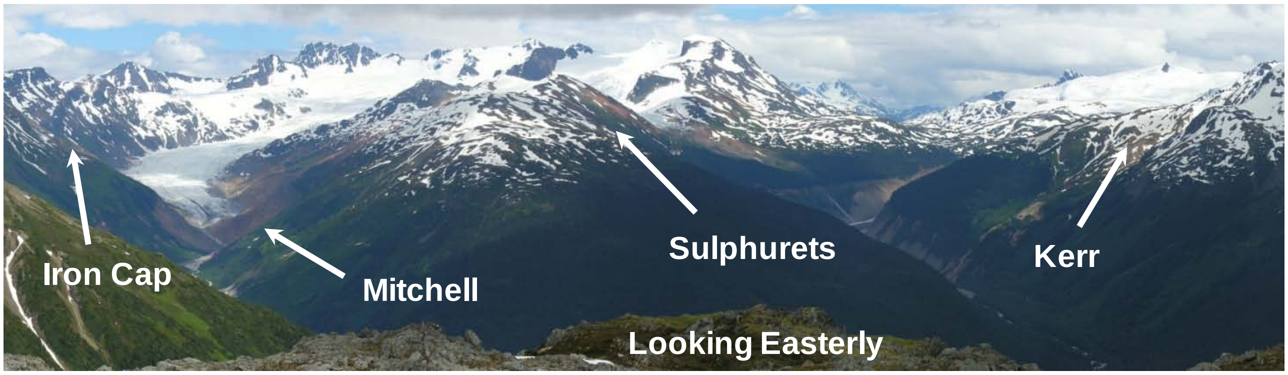
March 2011 Updated KSM Mineral Resources



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1.0 SUMMARY

1.1 Location and Ownership

The Kerr-Sulphurets-Mitchell-Iron Cap (KSM) property is located in northwest British Columbia at a latitude and longitude of approximately 56.52^{O}N and 130.25^{O}W, respectively. These four mineralized zones are located about 950 kilometers northwest of Vancouver, 65 kilometers north-northwest of Stewart and 21 kilometers south-southeast of the Eskay Creek Mine.

The KSM property is comprised of three discontinuous claim blocks. These claim blocks are referred to as 1) the KSM claim group, 2) the Seabee/Tina claims, and 3) the KSM placer claim block. The first two claim blocks (KSM and Seabee/Tina) contain 115 mineral claims, consisting of both cell and legacy claims. The total area of the three claim blocks covers an area of approximately 44,120 hectares. The Seabee/Tina claim block is located about 19 kilometers northeast of the Kerr-Sulphurets-Mitchell-Iron Cap mineralized zones. The Seabee/Tina claim block is currently being considered for proposed infrastructure siting. The claims are 100% owned by Seabridge Gold Inc. Placer Dome Inc. (now Barrick Gold) retains a capped 1% net smelter royalty on the property.

1.2 Geology and Mineralization

The property lies within an area known as “Stikinia”, which is a terrane consisting of Triassic and Jurassic volcanic arcs that were accreted onto the Paleozoic basement of the early North American plate. Early Jurassic sub-volcanic intrusive complexes are scattered through the Stikinia terrane and are host to numerous precious and base metal rich hydrothermal systems. These include several well known copper-gold porphyry systems such as Galore Creek, Red Chris, Kemess, Mt. Milligan in addition to the large cluster of deposits in the Sulphurets district, which hosts KSM and the adjacent Snowfield and Brucejack deposits.

At KSM, volcanics and sediments of the Triassic are assemblage belong the the Stuhini Group, which is disconformably to unconformably overlain by Jurassic volcanics and sediments of the Hazelton Group. The Stuhini Group includes turbidic siltstone, minor limestone, basaltic flows and tuffs, and thick sequences of conglomerate. These are interpreted to have formed in a deep marine environment transitioning to a shallow marine environment. They have been subjected to multiple deformation events and exhibit a low greenstone facies metamorphic grade with penetrative cleavage. The Hazelton Group consists of andesite flows, breccias, and pyroclastics, rhyodacitic welded tuffs, and interbedded sedimentary units. The Jack Formation is interpreted to be a basal conglomerate marking the beginning of the Hazelton Group. Hazelton Group rocks transition from shallow marine to a mixed marine and terrestrial environment, and are in turn conformably overlain by a thick back-arc assemblage of black siliclastic sediments of the Bowser Lake Group north and east of the property.

The property lies within the Skeena fold and thrust belt, which was formed during a Cretaceous deformational event. As a result, Triassic rocks have been thrust over Jurassic rocks at KSM, and a series of imbricate thrust sheets have dismembered much of the property and deposits into distinct structural panels. The principal thrust faults are the Sulphurets and Mitchell, which in general dip moderately northwest. However, geometry is complex as compressional stresses were preferentially accommodated by phyllic altered rocks with lower competency, and re-aligned around competent intrusive bodies behaving as buttresses. Later folding and normal faulting resulted in further geometrical complexity.

The Kerr, Sulphurets, Mitchell, and Iron Cap zones are separate and unique deposits each containing hundreds of millions of tonnes. All are classified as calc-alkaline gold-copper porphyries developed as a result of the shallow emplacement of Late Jurassic intrusives of monzonitic to diorite composition. Mineralization is finely disseminated and quartz veinlet stockwork hosted in both the intrusive and host volcanic-sedimentary rock assemblages of the Stuhini and Hazelton Groups. Gold and copper mineralization tends to be relatively low-grade but dispersed over a very large area. Most of the mineralization appears to be hypogene with the principal sulfides being pyrite and chalcopyrite with minor molybdenite and trace amounts of tennantite, bornite, sphalerite, and galena. Within the higher-grade cores of the deposits, gold and copper grades tend to correlate well with one another. Preliminary work indicates that gold is intimately associated with chalcopyrite, but later overlapping hydrothermal activity may remobilized and/or deposited additional metal. In general, within the currently drilled areas, gold and copper grades tend to be remarkably consistent between drill holes especially in the deeper and larger Mitchell zone.

1.3 Project Status

Seabridge Gold entered into the district with a gold-enriched copper porphyry target concept. To that extent, Seabridge assembled and reviewed all of the available previously collected data, performed their own reconnaissance level traverses, and have conducted drilling campaigns during the 2006, 2007, 2008, 2009, and 2010 field seasons. Since entering into the district, Seabridge has drilled 220 diamond core holes totaling about 81,000 meters. This drilling data augments an additional 42,625 meters of drilling information collected by other companies.

Mineral Resources were estimated for the Kerr, Sulphurets, Mitchell, and Iron Cap zones by creating three-dimensional block models. Gold, copper, silver, and molybdenum grades were estimated using 15-meter-long drill hole composites by inverse distance and nearest neighbor methods. The estimated block grades were validated using visual and statistical methods. Based on these tests, the grade models are globally unbiased and represent a reasonable estimate of in situ resources. A portion of the estimated blocks were classified into Measured (Mitchell only), Indicated and Inferred Mineral Resources based on mineralized continuity, the distance to drilling data coupled with the number of holes that were used in the estimate.

Table 1-1 summarizes the estimated global Kerr, Sulphurets, Mitchell, and Iron Cap Mineral Resources using a 0.50 g/t gold-equivalent cutoff grade. The gold equivalent grade was calculated using gold and copper prices of US$650 per ounce and US$2.00 per pound, respectively. In addition, gold and copper recoveries of 70% and 85% were used in the gold equivalency calculation, respectively (see Section 17.10 for a more detailed explanation of the gold equivalent calculation). Note that the KSM resources shown in Table 1-1 are inclusive of Mineral Reserves that were disclosed in 2010 (Wardrop, 2010).

Table 1-1: Summary of KSM Mineral Resources

Measured Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	677,600	0.64	13,943	0.17	2,539	3.2	69,713	58	86.6
Total	677,600	0.64	13,943	0.17	2,539	3.2	69,713	58	86.6

Indicated Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	1,069,500	0.59	20,287	0.17	4,007	3.2	110,033	60	141.4
Sulphurets	199,300	0.63	4,037	0.26	1,142	0.7	4,485	59	25.9
Kerr	241,200	0.25	1,939	0.47	2,499	1.2	9,306	n/a	n/a
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	1,871,700	0.52	31,380	0.23	9,322	3.1	186,620	57	204.8

Measured Plus Indicated Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	1,747,100	0.61	34,230	0.17	6,546	3.2	179,746	59	228.0
Sulphurets	199,300	0.63	4,037	0.26	1,142	0.7	4,485	59	25.9
Kerr	241,200	0.25	1,939	0.47	2,499	1.2	9,306	n/a	n/a
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	2,549,300	0.55	45,323	0.21	11,861	3.1	256,333	57	291.4

Inferred Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	551,000	0.43	7,617	0.14	1,700	3.1	54,917	47	57.1
Sulphurets	160,500	0.53	2,735	0.16	566	1.1	5,676	34	12.0
Kerr	91,500	0.23	677	0.30	605	0.7	2,059	n/a	n/a
Iron Cap	297,300	0.36	3,441	0.20	1,310	3.9	37,278	60	39.3
Total	1,100,300	0.41	14,470	0.17	4,181	2.8	99,930	49	108.4

Note: Mineral Resources which are not Mineral Reserves do no have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Seabridge has contracted a number of consulting groups that have collected a variety of data and performed a number of analyses in mining, processing, and permitting disciplines. In September 2009, Seabridge updated their December 2008 Preliminary Economic Assessment (PEA). At the end of March 2010, Wardrop, along with numerous other consulting groups completed a Prefeasibility Study (PFS) (Wardrop, 2010). Most of the consultants that participated in the 2010 PFS are currently working to update the PFS using the resources that are the subject of this report. The updated PFS is scheduled to be completed during the second quarter of 2011. The list of consultants currently involved in updating the PFS includes:

- Resource Modeling Inc. (RMI)
- TJS Mining-Met Service Inc. (TJS)
- Moose Mountain Technical Services (MMTS)
- WN Brazier Associates Inc. (Brazier)
- Klohn Crippen Berger Ltd. (KCBL)
- Bosche Ventures Ltd. (BVL)
- McElhanney Consulting Services, Ltd. (McElhanney)
- BGC Engineering Inc. (BGC)
- EBA Engineering Consultants Ltd. (EBA)
- Thyssen Mining Construction of Canada Ltd. (Thyssen)
- Allnorth Consultants Ltd. (Allnorth)
- Rescan Environmental Services Ltd. (Rescan)
- Golder Associates, Ltd. (Golder)
- Ventilation Services Inc. (VSI)
- PSI, a Division of Ausenco
- SGS-Canadian Environmental and Metallurical Inc. (SGS-CEMI)
- Wardrop Engineering Inc. (WEI)

1.4 Conclusions

Since entering into the district in 2004, Seabridge Gold has drilled 220 diamond core holes totaling about 81,000 meters. Their 2006 through 2010 drilling programs have confirmed the presence of large disseminated gold-copper systems referred to as the Kerr, Sulphurets, Mitchell, and Iron Cap zones which make up the KSM property.

The Kerr zone contains much lower gold grades than the neighboring Sulphurets and Mitchell deposits but has higher copper grades than those deposits. The higher Kerr copper grades may be beneficial for blending purposes.

The Sulphurets zone, while smaller than the Mitchell zone, represents an attractive target due to its proximity to the Mitchell zone but having higher gold and copper grades than Mitchell along with near surface exposures of mineralization. Additional drilling will be required to close off the deposit and to upgrade the current Inferred Mineral Resources to higher confidence categories. Drilling in 2010 helped to confirm previously recognized mineralization within the "Main Copper" zone, which had been identified in the 1990's by Placer Dome and earlier companies.

The geology, dimensions and metal distribution of the Mitchell deposit is consistent with those of a gold-enriched, low-grade copper porphyry model. For all practical purposes, the limit of the Mitchell zone has been closed off by drilling. While there still remains some potential in the down-dip direction to the north, lower grades and higher strip ratios associated with that mineralization would probably preclude the material being economically viable by open pit mining methods.

The Iron Cap zone was the focus of Seabridge's 2010 drilling program and resulted in the definition of a substantial resource. Ongoing metallurgical and geotechnical studies will be required to determine if any portion of this resource can be elevated to reserve status.

Substantial test results indicate that the mineral samples from the four mineralized zones are amenable to a combined flotation-cyanidation process. The process as currently envisioned consists of the following key points:

- Copper-gold-molybdenum bulk rougher flotation followed by gold-bearing pyrite flotation.

- Regrinding the resulting bulk rougher concentrate followed by three stages of cleaner flotation to produce a copper-gold-molybdenum bulk cleaner flotation concentrate.

- Molybdenum separation from the bulk cleaner flotation concentrate to produce a molybdenum concentrate and a copper/gold concentrate containing associated silver.

- Cyanide leaching of the gold-bearing pyrite flotation concentrate and the scavenger cleaner tailing to further recover gold and silver values as doré bullion.

Samples from the Mitchell and Sulphurets zones produced better metallurgical results with the chosen flotation circuit and cyanide leach extraction when compared to metallurgical results from samples taken from the Kerr and Iron Cap zones.

1.5 Recommendations

- A modest drilling campaign of 10-15 core holes totaling around 3,000 meters could potentially upgrade currently defined Kerr Inferred Resources to Indicated. This program is estimated to cost about $1,000,000.

- There is potential to increase resources within the Sulphurets zone. The area with potential is located between the Canyon Zone, located at the southwest end of the deposit and the main zone of mineralization to the northeast. It is estimated that 30-35 holes would be required totaling about 10,000 meters. This program is estimated to be cost approximately $3,500,000.

- Infill drilling should be completed within key areas of the Iron Cap zone. Approximately 10-15 core holes totaling about 5,000 meters would increase the overall confidence level of the resource. This program is estimated to cost about $1,750,000.

- Continue with geotechnical studies for determining possible pit slope angles for each of the four zones. Seabridge has been working several geotechnical consulting companies to determine appropriate pit slope angles. The author is unaware of the magnitude of costs associated with these activities.

- Additional metallurgical test work and mineralogical evaluations should be conducted to optimize process conditions and to establish design-related parameters for the next stage of study. The test work should include variability testing of samples from Sulphurets, Kerr and Iron Cap zones. The cost of the test work is estimated at $500,000.

- Further investigation of the separation between copper and molybdenum from the bulk concentrate should be included in the next study phase. The potential additional value of rhenium in the molybdenum concentrate should be evaluated at an estimated cost of $150,000.

- Further study should be conducted to optimize the proposed cyanide recovery and destruction methods. The cost is estimated to be approximately $100,000.

- Test work to confirm the slurry pumping arrangement to deliver the ore slurry from mine site and plant site should be conducted to confirm the current preliminary design. The cost for this is estimated to be approximately $200,000.

- Continue gathering environmental base line data for possible permitting of the project. Seabridge has contracted Rescan out of Vancouver, B.C. to manage and direct these efforts. The author is unaware of the costs estimated to complete these activities.

1.6 Risks and Uncertainties

- **Resource** - In RMI's opinion, there is little risk associated with the insitu Mineral Resources which are the subject of this report. These estimated resources are based on drilling data that have been verified by RMI and are supported by adequate QA/QC results. Diamond drilling has shown that mineralization tends to be fairly continuous and widespread, especially within the Mitchell zone. Gold and copper variograms suggest long ranges of mineralized continuity along preferential orientations. The estimated block grades have been demonstrated to be globally unbiased and provide a reasonable estimate of local grades. Back testing previous block models with newly obtained infill drilling results have been favorable. The resources which are the subject of this report were not confined to a conceptual pit. RMI used the same cutoff grade that has been used for past resource estimates for comparison purposes. That cutoff grade of 0.50 g/t gold equivalent is higher than a cutoff grade calculated using current prices or the average price over the past several years. RMI did generate a number of conceptual pits for each mineralized zone and compared resources captured by those pits versus the global inventory using the same cutoff grade. The "base case" conceptual pits captured nearly all the Measured (Mitchell only) and Indicated Mineral Resources for all four zones. The conceptual pits captured less Inferred material than the global inventory, especially for the Mitchell zone. Inferred material by its very nature is speculative and may never be upgraded into higher categories.

- **Mining** - Interim and final pit slope angles for each zone are currently being analyzed by several consulting groups. The south and north ultimate high walls of the Mitchell pit present a significant risk due to their overall heights, which are in excess of 1,500 meters. According to Moose Mountain Technical Services, the geotechnical design has been completed to a higher level of detail than is typical for a prefeasibility level study. However, walls of this height have not been built to date. Moose Mountain Technical Services also point out that for the first seven years of mill feed the Mitchell high wall height is less than 1,000m high, for which there is precedence. The current plan also shows that the high wall will be around 1,200m high for the first 16 years of mill feed. Another potential mining issue surrounds glacial ice. Currently a small portion of the Mitchell Glacier is located inside of the "ultimate" pit. However, the glacier has been retreating at a rate of approximately 30m/year. At that rate of melt back coupled with mine scheduling it is likely that no ice will need to be mined. Glacial melt water will need to be diverted from the pit and various diversion plans are being analyzed. The ice field above the Iron Cap zone is more problematic as more ice would have to be contended with than at Mitchell. Various mine planning scenarios are currently being studied. One scenario would call for Iron Cap to be the last KSM zone that is mined. That plan would call for mining ice above the deposit and placing it in the mined out Mitchell pit. Other solutions are being analyzed to deal with this issue including the potential of mining the zone using block caving methods.

- **Processing** - Metal recoveries for Sulphurets and Mitchell appear to be higher than those for the Kerr and Iron Cap zones. If ongoing test work results in showing lower recoveries a portions of the KSM resource could be reduced. Contracts for accepting concentrates from KSM will need to be secured.

- **Permitting** - At this juncture the authors are not aware of any fatal flaws associated with obtaining the various permits needed to construct and operate a mine at this site. However, permitting of any large undeveloped project represents an ongoing risk. RMI has held discussions with Mr. Clem Pelletier, CEO of Rescan Environmental Services Ltd. Mr. Pelletier has indicated that at this stage of the project there is no indication that the project cannot be permitted or that Seabridge will not gain its social license to operate, including the cooperation of the local Aboriginal peoples. However, Mr. Pelletier pointed out that a significant amount of work will need to be completed in order to obtain all required permits.

- **Capital Costs** - In the last KSM Prefeasibility Study (Wardrop, 2010), the capital cost estimate to develop this project was 3.3 billion dollars. This is a significant cost. Based on current estimates and economic studies, the project is not particularly sensitive to capital costs. However, if capital costs were to dramatically increase, the economic return of the project could be adversely affected.

- **Metal Prices** - Metal prices have been at record highs over the past few years, particularly gold. However, if prices were to dramatically fall, the overall project economics could be seriously impaired.

2.0 INTRODUCTION

The Kerr-Sulphurets-Mitchell-Iron Cap (KSM) copper-gold project is owned by Seabridge Gold Inc. The resource estimates that are the subject of this technical report were prepared at the request of Seabridge Gold Inc. The purpose of this report is to comply with disclosure and reporting requirements set forth in the Canadian Venture Exchange (CDNX) Corporate Finance Manual, National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1.

The scope of this study included a review of all available technical reports and data in the possession of Seabridge relative to the general setting, geology, mineralization, project history, previous exploration activities, drilling results, sampling/assaying methods, quality assurance/quality control (QA/QC) protocols, metallurgical, and environmental data.

Mr. Michael J. Lechner, P. Geo., President of Resource Modeling Inc. and Mr. Jianhui (John) Huang, P. Eng., Senior Process, Wardrop, a Tetra Tech Company are the qualified person's for this report. Mr. Lechner is responsible for all sections except 16. Mr. Huang is responsible for Section 16.

Mr. Lechner's primary mandate was to review newly acquired drilling data and to update the Kerr, Sulphurets, and Mitchell resource estimates so as to conform with National Instrument 43-101. In addition, Mr. Lechner estimated resources for the Iron Cap zone.

Mr. Huang's mandate was to review all past and ongoing metallurgical test work and to help develop a processing flow sheet.

Seabridge Gold provided Mr. Lechner with various electronic data including drill hole information, assay certificates, quality assurance quality control results, and various geologic interpretations. In addition, various Seabridge personnel (Mr. William Threlkeld, Mr. Mike Savell, Mr. Timothy Dodd, Mr. Peter Erwich, and Mr. Brent Murphy) have greatly contributed in the preparation of this document by providing detailed information about the location, history, geology, mineralization, exploration, and permitting activities associated with the KSM project. Additional support was provided by Mr. Huang regarding metallurgical testing and potential processing scenarios.

Mr. Lechner conducted a site visit of the KSM project from July 29 to August 1, 2009. The author spent three days on site visiting several operating drill rigs, examining mineralized exposures in the Mitchell Creek drainage, as well as examinations of the Sulphurets and Kerr deposits. The author was accompanied by several Seabridge Gold personnel, including Mr. Timothy Dodd, Senior Geologist and Mr. Peter Erwich, Senior Geologist. Mr. Dodd and the author spent one day surveying a number of pre-2009 Kerr, Sulphurets, and Mitchell drill hole collars using Seabridge's DGPS unit.

The Seabridge geologists provided the author with a detailed overview of the 2009

Seabridge drilling campaign. A thorough review was made of drilling, sampling procedures, assay sample chain of custody procedures, core logging, sample shipping, and core storage. The author also examined newly acquired drill core from the Mitchell zone and reviewed lithologic/alteration logging procedures.

Mr. Huang, site visited the KSM project on September 16, 2008. While on site, Mr. Huang performed a general site inspection and examined drill core.

Units of measure and various conversion factors used in this report include:

Linear Measure

1inch	=2.54 centimeters	
1 foot	=0.3048 meter	
1 yard	=0.9144 meter	
1 mile	=1.6 kilometers	

Area Measure

1 acre	=0.4047 hectare	
1 square mile	=640 acres	=259 hectares

Weight

1 short ton	=2000 pounds	=0.907 tonne
1 pound	=0.454 kilogram	=14.5833 troy ounces

Assay Values

1 oz per ton	=34.2857 gram/tonne	
1 troy ounce	=31.1035 grams	
1ppb	=0.0000292 oz per ton	

Rounding

Some apparent discrepancies in the calculation of gold ounces may occur due to the rounding of either tonnes and/or gold grades.

All currency amounts in this report are stated in terms of Canadian dollars unless otherwise stated.

3.0 RELIANCE ON OTHER EXPERTS

The Mineral Resource estimate that is discussed in this report was prepared by Mr. Lechner using data that were provided to him by Seabridge Gold. The author has personally verified the assay data that have been collected from Seabridge's 2006, 2007, 2008, 2009, and 2010 field seasons. A significant portion of the Kerr and Sulphurets data were collected by other companies prior to Seabridge's acquisition of the property.

In preparing this document, the author's did not check title to Seabridge's mining claims and hereby disclaims any responsibility for such matters. Seabridge has retained The Claim Group, an independent consulting firm based in Mississagua, Ontario, to confirm title to the claims (Brassard, 2010).

RMI held discussions with Mr. Jim Gray and Mr. Tracey Meintjes from Moose Mountain Technical Services regarding various mining scenarios including parameters for conceptual pits and potential glacial ice mining.

RMI has held discussions with Mr. Clem Pelletier, CEO of Rescan Environmental Services Ltd. Mr. Pelletier has indicated that at this stage of the project, there is no indication that the project cannot be permitted or that Seabridge will not gain it's social license to operate, including the cooperation of the local Aboriginal peoples. However Mr. Pelletier highlighted that significant work remains to be completed to permit this project."

RMI discussed capital cost estimates with Mr. Jim Smolik, President of TJS Mining-Met Services Inc. According to Mr. Smolik, as currently envisioned, the KSM project is not particularly sensitive to capital costs. However, Mr. Smolik points out that if capital costs were to dramatically increase, the economic return of the project could be adversely affected.

This report was prepared for Seabridge by the authors and is based in part on information not within the control of either Seabridge or the author's, although the majority of the Mitchell and Iron Cap data have been collected by Seabridge. While it is believed that the information contained herein is reliable under the conditions and subject to the limitations set forth herein, the author cannot guarantee the accuracy thereof. The author's are unaware of any existing technical data other than those that were provided to them by Seabridge and other consultants working on the project. The use of this report, or any information contained herein shall be at the user's sole risk, regardless of any fault or negligence of the author's.

4.0 PROJECT DESCRIPTION AND LOCATION

The KSM property is located in northwest British Columbia, at an approximate latitude of 56.50N and a longitude of 130.30W. The Mineral Resources that are the subject of this report are located relative to the NAD83 UTM coordinate system. The property is situated approximately 950 kilometers northwest of Vancouver, 65 kilometers north-northwest of Stewart, and 21 kilometers south-southeast of the Eskay Creek Mine (production ceased in 2009). Figure 4-1 is a general location map.

The KSM property is comprised of three discontinuous claim blocks. These claim blocks are referred to as 1) the KSM claim group, 2) the Seabee/Tina claims, and 3) the KSM placer claim block. The first two claim blocks (KSM and Seabee/Tina) contain 115 mineral claims, consisting of both cell and legacy claims. The total area of the three claim blocks covers an area of approximately 44,120 hectares. The Seabee/Tina claim block is located about 19 kilometers northeast of the Kerr-Sulphurets-Mitchell-Iron Cap mineralized zones. The Seabee/Tina claim block is currently being considered for proposed infrastructure siting.

The Kerr-Sulphurets-Mitchell mineral claims were purchased by Seabridge from Placer Dome in 2000. The mineral claims were converted from legacy claims to B.C.'s new Mineral Titles Online (MTO) system in 2005. In the MTO system, claims are located digitally using a fixed grid on lines of latitude and longitude with cells measuring 15 seconds north-south and 22.5 seconds east-west (approx. 460 by 380 meters at KSM). The legacy claims were located by previous owners by placing tagged posts along the boundaries; however the survey method employed in locating the legacy claims is not known. With the MTO system no markings are required on the ground and the potential for gaps and/or overlapping claims inherent in the old system is eliminated.

There is no record or evidence of any historical mining on the property. The B.C. Mineral Inventory (Minfile) contains 25 mineral occurrences in this area (mostly copper and gold). Also, within the claim group two non-compliant (pre-NI 43-101) Mineral Resources were reported by Placer Dome for the Kerr and Sulphurets deposits.

The original KSM claim group consisted of two contiguous claim blocks known as the Kerr and Sulphurets (or Sulphside) properties. The claims are 100% owned by Seabridge. Placer Dome Inc. (now Barrick Gold) retains a 1% net smelter royalty (NSR) that is capped at $4.5 million. Two of the pre-converted claims (Xray 2 and 6) are subject to a contractual royalty obligation in accordance with terms in the underlying Dawson Agreement. The lands covered by these claims are now contained within the converted Xray 1 claim (Tenure No. 516245). There is an additional underlying agreement whereby advance annual royalties payable to Dawson are being paid by Seabridge.

Since acquisition of the original KSM claim group, Seabridge has added to the project's property holdings through staking and purchase of several claim groups. These include the Seabee group, acquired by staking, the Tina and BJ groups purchased in

2009, and the New BJ group purchased in 2010. The Seabee and Tina groups are together referred to as the Seabee property, and the original KSM group, BJ and New BJ groups are referred to as the KSM property (see Figure 4-2). The Kerr-Sulphurets placer claims were part of the original property acquisition from Placer Dome Inc. Additional placer claims were acquired by staking in 2009 (see Figure 4-3).

Annual holding costs for all claims (lode and placer) are approximately $172,988, which the company has maintained since acquiring the project. In 2007, assessment work was filed to advance the expiry of the KSM property to 2018. Assessment work was completed on most of the Seabee property in 2010 with that work filed in February 2011 which advanced expiry dates to 2017. The BJ group of claims had assessment work from 2010 applied which advanced expiry dates to 2020. The Kerr-Sulphurets placer claims have been kept in good standing by paying fees in lieu of completing assessment work. The Claim Group Inc. (TCG) is the land manager and mineral tenure agent for Seabridge Gold Inc. Seabridge is provided with monthly 90-day forward reports of all land tenures (lode and placer) requiring action within that period. TCG files any work done on the properties, based on details provided by Seabridge, or files cash in lieu of work, for the company.

The author's are unaware of any environmental liabilities associated with the KSM project. It is the understanding of the author's that Seabridge has obtained permits for ongoing exploration work. Seabridge is in the process of obtaining other permits (see Section 18).

Table 4-1: KSM Claims - Lease Application EPC461 in Progress

Claim Number	Claim Name	Area (hectares)	Expiry Date	Annual Work (CIL) Due	Annual Fees	Cells or Units	TRIM Map No.
254756	ARBEE #35	25.0	June 16, 2018	$200.00	$10.00		104B059
254757	ARBEE #39	25.0	June 16, 2018	$200.00	$10.00		104B059
254758	ARBEE #54	25.0	June 14, 2018	$200.00	$10.00		104B059
254759	ARBEE #55	25.0	June 16, 2018	$200.00	$10.00		104B059
516236		303.3	June 30, 2018	$2,426.18	$121.31	17	104B059
516237		71.4	June 30, 2018	$571.03	$28.55	4	104B059
516238		624.5	December 10, 2018	$4,995.65	$249.78	35	104B059
516239		535.5	December 10, 2018	$4,284.10	$214.21	30	104B059
516240		107.0	June 30, 2018	$856.13	$42.81	6	104B059
516241		142.7	June 30, 2018	$1,141.67	$57.08	8	104B059
516242		71.4	September 23, 2018	$570.90	$28.55	4	104B059
516245		356.9	October 12, 2018	$2,855.37	$142.77	20	104B059
516248		142.7	August 26, 2018	$1,141.80	$57.09	8	104B059
516251		321.3	August 26, 2018	$2,570.75	$128.54	18	104B059
516252		125.0	August 26, 2018	$999.95	$50.00	7	104B059
516253		178.6	August 26, 2018	$1,428.98	$71.45	10	104B059
516254		285.8	August 26, 2018	$2,286.23	$114.31	16	104B059
516255		214.3	September 23, 2018	$1,714.77	$85.74	12	104B049
516256		53.6	August 26, 2018	$428.69	$21.43	3	104B049
516258		178.6	November 3, 2018	$1,428.58	$71.43	10	104B059
516259		107.2	November 3, 2018	$857.38	$42.87	6	104B049
516260		107.2	November 3, 2018	$857.58	$42.88	6	104B049
516261		464.6	December 20, 2018	$3,717.08	$185.85	26	104B049
516262		339.5	December 17, 2018	$2,716.21	$135.81	19	104B049
516263		643.9	December 17, 2018	$5,151.05	$257.55	36	104B049
516264		393.3	October 30, 2018	$3,146.75	$157.34	22	104B049
516266		178.8	December 17, 2018	$1,430.22	$71.51	10	104B049
516267		250.2	December 17, 2018	$2,001.94	$100.10	14	104B049
516268		321.8	December 17, 2018	$2,574.69	$128.73	18	104B049
516269		107.2	August 26, 2018	$857.66	$42.88	6	104B049
Total	n/a	6,726.4	n/a	$53,811.35	$2,690.57	n/a	n/a

Table 4-2: KSM Claims - Lease Application EPC462 in Progress

Claim Number	Claim Name	Area (hectares)	Expiry Date	Annual Work (CIL) Due	Annual Fees	TRIM Map No.
394782	BJ 7	500.0	December 11, 2020	$4,000.00	$200.00	104B059
394783	BJ 8	500.0	December 11, 2020	$4,000.00	$200.00	104B059
394784	BJ 9	400.0	December 11, 2020	$3,200.00	$160.00	104B059
394792	BJ 16	500.0	December 11, 2020	$4,000.00	$200.00	104B059
394793	BJ 17	400.0	December 11, 2020	$3,200.00	$160.00	104B059
394795	BJ 19	500.0	December 11, 2020	$4,000.00	$200.00	104B059
394796	BJ 20	375.0	December 11, 2020	$3,000.00	$150.00	104B059
394799	BJ 23	500.0	December 11, 2020	$4,000.00	$200.00	104B059
394800	BJ 24	300.0	December 11, 2020	$2,400.00	$120.00	104B059
394801	BJ 25	500.0	December 11, 2020	$4,000.00	$200.00	104B059
394802	BJ 26	250.0	December 11, 2020	$2,000.00	$100.00	104B059
394803	BJ 27	200.0	December 11, 2020	$1,600.00	$80.00	104B059
394804	BJ 28	100.0	December 11, 2020	$800.00	$40.00	104B059
394805	BJ 29	300.0	December 11, 2020	$2,400.00	$120.00	104B049
394806	BJ 30	400.0	December 11, 2020	$3,200.00	$160.00	104B049
394807	BJ 31	500.0	December 11, 2020	$4,000.00	$200.00	104B049
Total	n/a	6,225.0	n/a	$49,800.00	$2,490.00	n/a

Table 4-3: KSM Claims - No Lease Application in Progress

Claim Number	Claim Name	Area (hectares)	Expiry Date	Annual Work (CIL) Due	Annual Fees	TRIM Map No.
394780	BJ5	100.0	November 30, 2011	$800.00	$40.00	104B059
394781	BJ6	100.0	November 30, 2011	$800.00	$40.00	104B059
394786	BJ 11	500.0	November 30, 2011	$4,000.00	$200.00	104B059
394787	BJ 12	500.0	November 30, 2011	$4,000.00	$200.00	104B059
394788	BJ 13	100.0	November 30, 2011	$800.00	$40.00	104B059
394789	BJ 13A	25.0	November 30, 2011	$200.00	$10.00	104B059
394790	BJ 14	100.0	November 30, 2011	$800.00	$40.00	104B059
394791	BJ 15	250.0	November 30, 2011	$2,000.00	$100.00	104B059
394794	BJ 18	300.0	November 30, 2011	$2,400.00	$120.00	104B059
394808	BJ 31 A	375.0	December 11, 2011	$3,000.00	$150.00	104B049
394809	BJ 32	150.0	December 11, 2011	$1,200.00	$60.00	104B049
394810	BJ 33	450.0	December 11, 2011	$3,600.00	$180.00	104B049
394811	BJ 34	150.0	December 11, 2011	$1,200.00	$60.00	104B049
394812	BJ 35	450.0	December 11, 2011	$3,600.00	$180.00	104B049
705591	BJ GAP1	231.6	February 5, 2021	$926.48	$92.65	104B059
705592	BJ GAP2	160.5	February 5, 2021	$641.84	$64.18	104B059
Total	n/a	3,942.1	n/a	$29,968.32	$1,576.83	n/a

Table 4-4: Seabee/Tina Mineral Claims

Claim No.	Claim Name	Area (hectares)	Expiry Date	Annual Placer Work Due	Annual Fees	Cells or Units	TRIM Map No.
566467	BRIDGE1	445.8	February 8, 2017	$3,566.61	$178.33	25	104A052
566468	BRIDGE2	445.6	February 8, 2017	$3,564.58	$178.23	25	104A052
566469	BRIDGE3	427.8	February 8, 2017	$3,422.34	$171.12	24	104A052
566470	BRIDGE4	428.0	February 8, 2017	$3,423.82	$171.19	24	104A052
566471	BRIDGE5	445.7	February 8, 2017	$3,565.87	$178.29	25	104A052
566472	BRIDGE6	445.6	February 8, 2017	$3,564.62	$178.23	25	104A052
566473	BRIDGE7	427.9	February 8, 2017	$3,423.38	$171.17	24	104A052
566474	BRIDGE8	427.8	February 8, 2017	$3,422.08	$171.10	24	104A052
566475	BRIDGE9	427.6	February 8, 2017	$3,420.90	$171.05	24	104A052
566476	BRIDGE10	445.5	February 8, 2017	$3,564.25	$178.21	25	104A052/053
566477	BRIDGE11	302.9	February 8, 2017	$2,423.06	$121.15	17	104A052/053
566478	BRIDGE12	427.4	February 8, 2017	$3,419.45	$170.97	24	104A061
566479	BRIDGE13	445.2	February 8, 2017	$3,561.22	$178.06	25	104A061
566481	BRIDGE14	445.1	February 8, 2017	$3,560.49	$178.02	25	104A061
566482	BRIDGE15	444.8	February 8, 2017	$3,558.74	$177.94	25	104A061
566484	BRIDGE16	444.6	February 8, 2017	$3,556.50	$177.82	25	104A061
566485	BRIDGE17	426.7	February 8, 2017	$3,413.82	$170.69	24	104A061
566487	BRIDGE18	444.7	February 8, 2017	$3,557.69	$177.88	25	104A061
566488	BRIDGE19	444.8	February 8, 2017	$3,558.68	$177.93	25	104A061
566489	BRIDGE20	445.0	February 8, 2017	$3,559.75	$177.99	25	104A061
566490	BRIDGE21	427.3	February 8, 2017	$3,418.11	$170.91	24	104A061
566491	BRIDGE22	445.2	February 8, 2017	$3,561.34	$178.07	25	104A061
566492	BRIDGE23	427.3	February 8, 2017	$3,418.46	$170.92	24	104A061/104B070
566493	BRIDGE24	427.9	February 8, 2017	$3,423.39	$171.17	24	104A052
566494	BRIDGE25	427.9	February 8, 2017	$3,423.40	$171.17	24	104A052/053
566495	BRIDGE26	444.9	February 8, 2017	$3,559.03	$177.95	25	104A061/104B070
566496	BRIDGE27	391.3	February 8, 2017	$3,130.52	$156.53	22	104B070
566497	BRIDGE28	444.5	February 8, 2017	$3,555.66	$177.78	25	104A061/104B070
566567	BRIDGE29	427.5	February 8, 2017	$3,419.66	$170.98	24	104A052/062
571582	SEABEE1	408.8	February 8, 2017	$3,270.63	$163.53	23	104A061
571583	SEABEE2	373.1	February 8, 2017	$2,985.10	$149.25	21	104A061
571584	SEABEE3	444.1	February 8, 2017	$3,552.54	$177.63	25	104A061,071
571585	SEABEE4	426.1	February 8, 2017	$3,408.66	$170.43	24	104A071
571586	SEABEE5	372.6	February 8, 2017	$2,981.11	$149.06	21	104A071
571587	SEABEE6	159.6	February 8, 2017	$1,277.14	$63.86	9	104A071
573813	SEABEE7	213.3	February 8, 2017	$1,706.10	$85.31	12	104A071
575633	SEA 1	445.2	February 8, 2017	$3,561.59	$178.08	25	104A051
575635	SEA 2	445.3	February 8, 2017	$3,562.41	$178.12	25	104A061
575636	SEA 3	445.4	February 8, 2017	$3,563.28	$178.16	25	104A061
575638	SEA 4	445.4	February 8, 2017	$3,563.58	$178.18	25	104A061
575639	SEA 5	445.3	February 8, 2017	$3,562.70	$178.13	25	104A061
575642	SEA 6	445.1	February 8, 2017	$3,560.68	$178.03	25	104A051
575643	SEA 7	213.4	February 8, 2017	$1,707.52	$85.38	12	104A051
575645	SEA 8	427.1	February 8, 2017	$3,416.66	$170.83	24	104A051
575646	SEA 9	35.6	February 8, 2017	$284.78	$14.24	2	104B070
603133	SEABEE 8	426.6	February 8, 2017	$1,706.24	$170.62	24	104B070
603134	SEABEE 9	53.4	February 28, 2017	$213.52	$21.35	3	104B070
401548	TINA 1	500.0	February 28, 2017	$4,000.00	$200.00		104B070
401549	TINA 2	500.0	February 28, 2017	$4,000.00	$200.00		104B070
401550	TINA 3	500.0	February 28, 2017	$4,000.00	$200.00		104B070
401551	TINA 4	500.0	February 28, 2017	$4,000.00	$200.00		104B070
401552	TINA 5	500.0	February 28, 2017	$4,000.00	$200.00		104B070
401553	TINA 6	250.0	February 28, 2017	$2,000.00	$100.00		104B070
Total	n/a	21,477.7	n/a	$169,901.66	$8,591.07	n/a	n/a

Table 4-5: Seabridge Placer Claims

Claim Number	Claim Name	Area (hectares)	Expiry Date	Annual Placer Work Due	Annual Fees	Cells or Units	TRIM Map No.
516323	PLACER CLAIM	107.2	September 30, 2011	$1,071.91	$214.38	6	104B049
516325	PLACER CLAIM	125.0	September 30, 2011	$1,250.43	$250.09	7	104B049
516328	PLACER CLAIM	71.5	September 28, 2011	$714.53	$142.91	4	104B049
516330	PLACER CLAIM	107.2	September 28, 2011	$1,071.85	$214.37	6	104B049
516332	PLACER CLAIM	107.2	September 28, 2011	$1,071.79	$214.36	6	104B049
516333	PLACER CLAIM	89.3	September 28, 2011	$893.34	$178.67	5	104B049
516375	PLACER CLAIM	125.0	September 30, 2011	$1,250.23	$250.05	7	104B049
516676	PLACER CLAIM	17.9	September 30, 2011	$178.58	$35.72	1	104B059
516677	PLACER CLAIM	17.9	July 11, 2011	$178.58	$35.72	1	104B059
576658	KERR PL1	446.9	February 20, 2012	$4,468.61	$893.72	25	104B049
576659	KERR PL2	446.6	February 20, 2012	$4,466.19	$893.24	25	104B049
576660	KERR PL3	446.4	February 20, 2012	$4,463.94	$892.79	25	104B059
576661	KERR PL4	446.2	February 20, 2012	$4,462.29	$892.46	25	104B059
576662	KERR PL5	446.0	February 20, 2012	$4,460.32	$892.06	25	104B059
576663	KERR PL6	446.0	February 20, 2012	$4,460.18	$892.04	25	104B059
576664	KERR PL7	142.7	February 20, 2012	$1,427.33	$285.47	8	104B059
576665	KERR PL8	321.4	February 20, 2012	$3,213.96	$642.79	18	104B059
576666	KERR PL9	285.7	February 20, 2012	$2,856.99	$571.40	16	104B059
576667	KERR PL10	357.4	February 20, 2012	$3,573.99	$714.80	20	104B049
694483	KSM P1	357.4	January 5, 2012	$3,573.60	$714.72	20	104B049
694543	KSM P2	410.5	January 5, 2012	$4,104.90	$820.98	23	104B059
694683	KSM P3	427.9	January 5, 2012	$4,278.60	$855.72	24	104B059
Total	n/a	5,749.2	n/a	$57,492.14	$11,498.43	n/a	n/a

The KSM Project is located on provincial Crown land. The four gold-copper deposits, and the proposed waste rock storage areas, lie within the Unuk River drainage in the area covered by the Cassia Iskur-Stikine Land and Resource Management Plan, approved by the British Columbia Government in 2000. A part of the proposed ore transport tunnel lies within the boundaries of the South Nass Sustainable Resource Management Plan that is currently in development. The proposed sites for the tailing management and plant facilities lie outside of the boundaries of any land use planning process. Part of the Project, excluding the mineral deposits and their immediately-related infrastructure, lies within the boundaries of the Nass Area, as defined in the Nisga'a Final Agreement, where consultation is required with the Nisga'a Lisims Government under the terms of the Final Agreement. The Tahltan First Nation has an asserted claim over part or all of the area underlying the Project footprint. Additionally, the Gitanyow and Gitxsan Hereditary Chiefs, including wilp Ski Km Lax Ha, may have some interests within the broader region, particularly downstream of the plant site and tailing management facility, potentially affected by the Project.

Seabridge Gold is nearing completion of an extensive two year environmental baseline program initiated in 2007 in support of the Provincial and Federal Governments permitting process. Environmental studies are being conducted under the leadership of Clem Pelletier, President of Rescan Environmental Services Ltd. Rescan is a Canadian-based international consulting firm offering a wide range of environmental and engineering services to clients around the world including many of the largest mining companies.

In March 2010, an application was made for a Multi-Year Area Based (MYAB) permit which covers work at the KSM property for a five year period. Approval for this work, which covers drilling, geophysical surveys, and base line environmental studies, was granted on June 30, 2010 (Permit # MX-1-571, Approval #10-0100108-0630). Prior to this approval, an extension to their 2009 permit was granted by the Ministry of Energy Mines and Petroleum Resources on April 30th, 2010, in order to complete the approved program commenced in 2009. Work on the Seabee property is covered by a separate permit, MX-1-763. An application for an MYAB permit for Seabee has been submitted.

Figure 4-2 shows Seabridge's mineral claim blocks including the KSM, Seabee, and Tina groups. The location of the four mineralized zones (Kerr, Sulphurets, Mitchell, and Iron Cap) is depicted in the southwestern portion of the figure. Figure 4-3 shows Seabridge's placer claims.

Figure 4-1: General Location Map

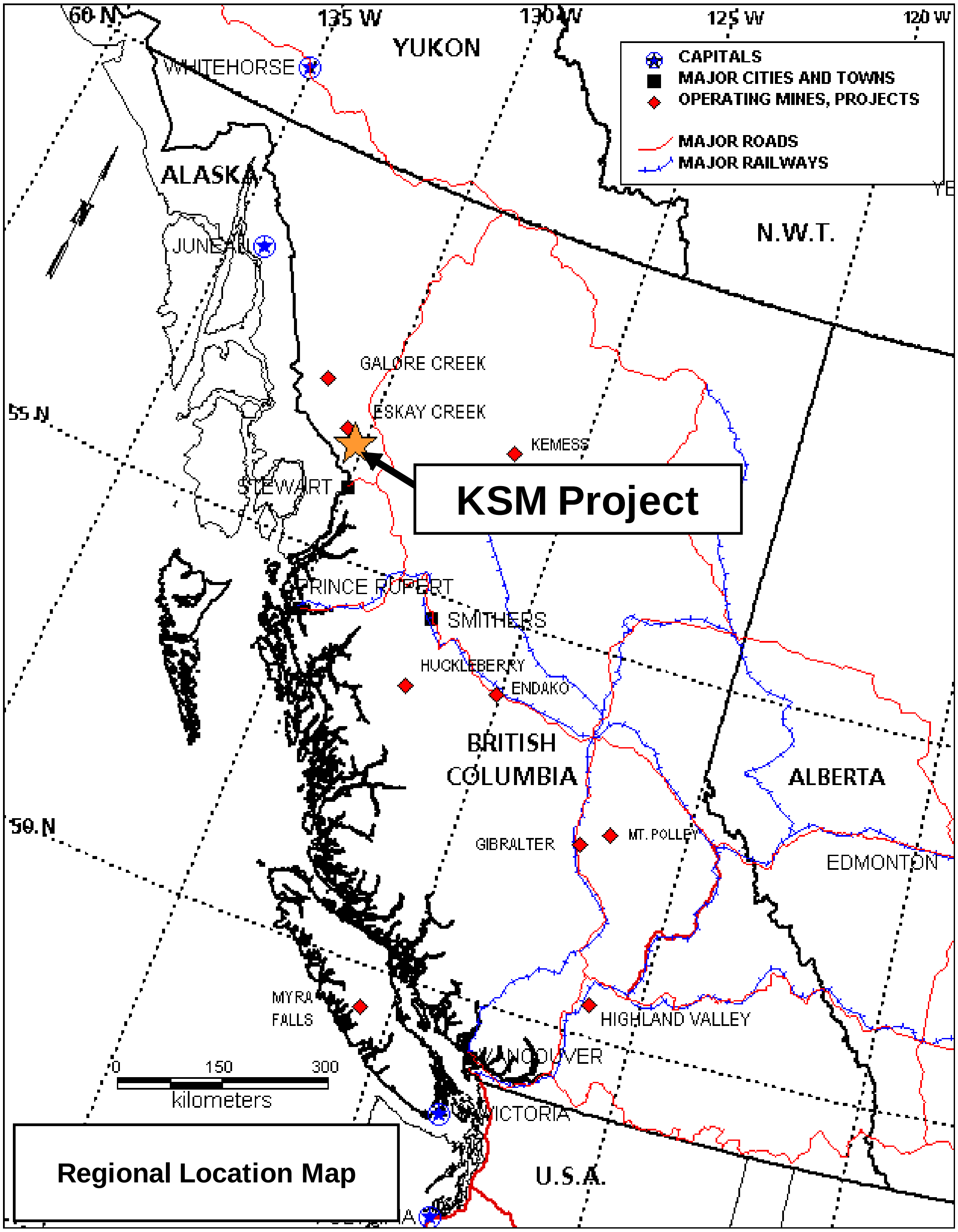


Figure 4-2: KSM Mineral Claim Map

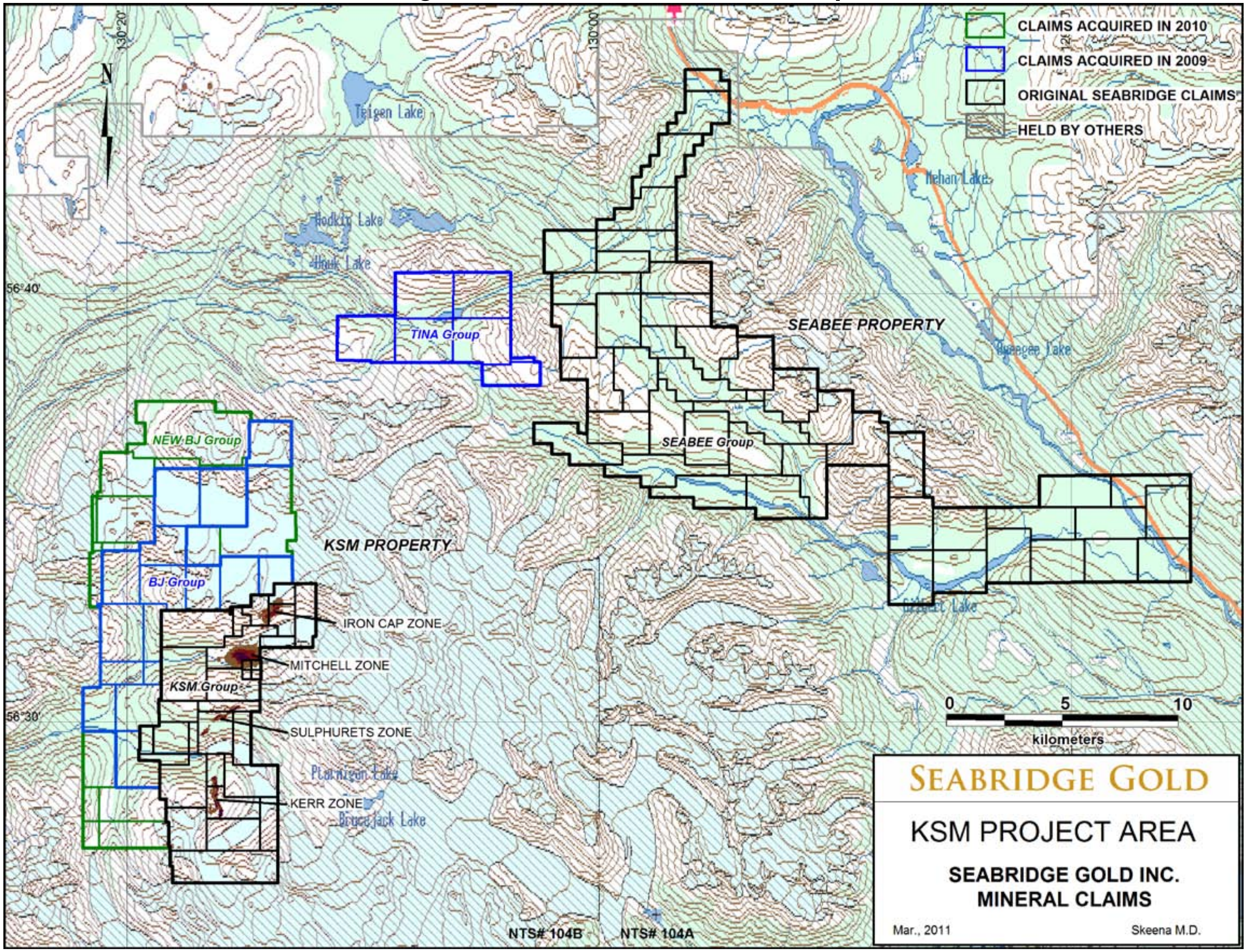
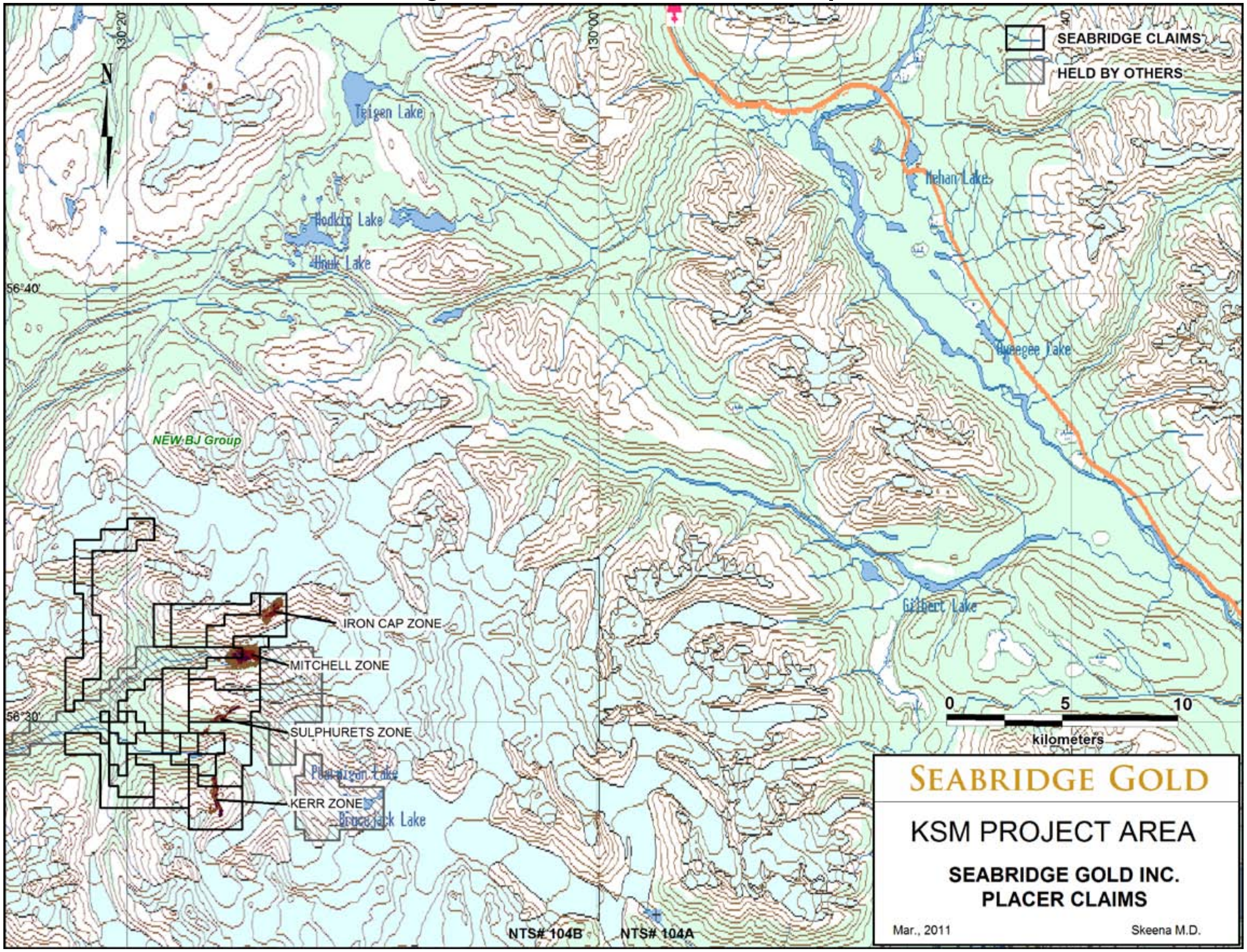


Figure 4-3: KSM Placer Claim Map



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following section was taken directly from RMI's April 6, 2007 NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia" (Lechner, 2007):

"The property lies in the rugged Coastal Mountains of northwest British Columbia, with elevations ranging from 520 meters in Sulphurets Creek valley to over 2,300 meters at the highest peaks. Valley glaciers fill the upper portions of the larger valleys from just below tree line and upwards. The glaciers have been retreating for at least the last several decades. Aerial photos from 1991 indicate the Mitchell Glacier has retreated almost a kilometer laterally and perhaps several hundred meters vertically since then.

The property is drained by Sulphurets and Mitchell Creek watersheds that empty into the Unuk River, which flows westward to the Pacific Ocean through Alaska. Tree line lies at about 1,240 meters above sea level, below which a mature forest of mostly hemlock and balsam fir abruptly develops. Fish are not known to inhabit the Sulphurets and Mitchell watersheds. Large wildlife such as deer, moose, and caribou are rare due to the rugged topography and restricted access however bears and mountain goats are relatively common.

The climate is generally that of a temperate or northern coastal rainforest, with sub-arctic conditions at high elevations. Precipitation is high with annual rainfall and snowfall totals estimated to be somewhere between the historical averages for the Eskay Creek Mine and Stewart B.C. These range from 801 to 1,295 millimeters of rain and 572 and 1,098 centimeters of snow, respectively (data to 2005). The length of the snow-free season varies from about May through November at lower elevations and from July through September at higher elevations.

Access to the property is via helicopter. Two staging areas for mobilizing crews and equipment were used. These are 1) an area located at kilometer 54 on the private Eskay Creek Mine Road, which is about 25 kilometers to the north-northwest of the property and 2) along the public Granduc Road, which is located about 35 kilometers to the south-southeast of the property, which in turn is about 40 kilometers north of the town of Stewart B.C. A section of this road passes through Alaska and the town of Hyder.

Stewart, a town of approximately 500 inhabitants, is the closest population center to the property. It is connected to the provincial highway system via paved, all weather highway (#37A). The larger population centers of Prince Rupert, Terrace, and Smithers, with a total population of about 32,000, are located approximately 270 kilometers to the southeast.

Deep water loading facilities for shipping bulk mineral concentrates exist in Stewart, and are currently used by both the Eskay Creek and Huckleberry Mines. The nearest railway is the CPR Yellowhead route, which is located approximately 220 kilometers to the southeast. This line runs east-west and terminates at the deep water port of Prince Rupert on the west coast of British Columbia.

The property lies on crown land, thus all surface and access rights are granted by the Mineral Tenure Act, the Mining Right of Way Act and the Mining Rights Amendment Act. There are no settlements or privately owned land in this area and no commercial or recreational activity is known to occur here. The closest power transmission lines run along the highway 37A corridor to Stewart, approximately 50 kilometers to the southeast. There are proposals to develop local hydroelectric power sources and extend the highway 37A transmission line northward.

AMEC of Vancouver, B.C. was commissioned by Noranda in 2004 to complete a scoping study to identify possible technical limitations for a conceptual large open-pit mining operation in the Kerr-Sulphurets area. The study recognized that within the claims, locating large plants, tailings and waste rock storage sites may be technically challenging, however ample space and favorable conditions exist in wide valleys approximately 20 kilometers to the east."

6.0 HISTORY

The following section was taken directly from RMI's April 6, 2007 NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia", (Lechner, 2007):

6.1 Exploration History

"The modern exploration history of the area began in the 1960's, with brief programs conducted by Newmont, Granduc, Phelps Dodge, and the Meridian Syndicate. All of these programs were focused towards gold exploration. Various explorers were attracted to this area due to the numerous large, prominent pyritic gossans that are exposed in alpine areas. There is evidence that prospectors were active in the area prior to 1935. The Sulphurets Zone was first drilled by Esso Minerals in 1969; Kerr was first drilled by Brinco in 1985 and Mitchell Creek by Newhawk Gold in 1991.

In 1989, a 100% interest in the Kerr deposit was acquired by Placer Dome from Western Canadian Mines and in the following year they acquired the adjacent Sulphurets property from Newhawk Gold Mines. The Sulphurets property also hosts the Mitchell Creek deposit and other mineral occurrences. In 2000, Seabridge Resources acquired a 100% interest from Placer Dome in both the Kerr and Sulphurets properties, subject to capped royalties.

There is no recorded mineral production, nor evidence of it, from the property. Immediately west of the property, small-scale placer gold mining has occurred in Sulphurets and Mitchell Creeks. On the Bruceside property immediately to the east and currently owned by Silver Standard Resources, limited underground development and test mining was undertaken in the 1990's on narrow, gold-silver bearing quartz veins at the West Zone. Table 6-1 summarizes the more recent exploration history of the Kerr zone."

Table 6-1: Exploration Summary of the Kerr Zone

Year	Activity
1982-1883	"Alpha JV" began prospecting and soil geochem surveys of the Kerr gossan focusing on gold
1984-1985	Brinco optioned the Kerr project, completed some geologic surveys and drilled 3 holes
1987-1989	Western Canadian Mines optioned Kerr and completed 59 drill holes and recognized Cu-Au porphyry
1989	Placer Dome (Placer) acquires Kerr property
1990-1992	Placer began delineation drilling of Kerr deposit at 50m centers by drilling 82 holes
1992-1996	Placer estimated resources (non NI 43-101), met testwork, and scoping studies
1996-2000	Project was dormant
2000	Seabridge Gold acquired a 100% interest in Kerr from Placer Dome
2002	Noranda Inc. acquired an option from Seabridge with the right to earn up to a 65% interest in Kerr
2003-2004	Noranda Inc. undertook various exploration surveys
2006	Seabridge Gold purchases Falconbridge (formerly Noranda) option
2009	Seabridge Gold drilled 7 holes totaling about 1,159m, conducted metallurgical testing, and permit work
2010	Seabridge Gold drilled 4 holes totaling about 1,453m, conducted metallurgical testing, and permit work

"Table 6-2 summarizes more recent exploration history of the Sulphurets/Mitchell zone."

Table 6-2: Exploration Summary of the Sulphurets/Mitchell Zone

Year	Activity
1880-1933	Limited placer gold exploration and mining
1935-1959	Placer gold prospecting, prospecting and staking of mining claims
1959-1960	Newmont and Granduc conducted surveys including airborne mag. Sulphurets and Iron Cap Au zones discovered. D. Ross, S. Bishop and W. Dawson prospected and stake claims in area.
1961-1968	Granduc Mines conducted geologic/geochem surveys, drilled 9 holes into Sulphurets zone. Ross-Bishop-Dawson claims optioned by Phelps Dodge in '62, Meridian Syndicate in '65, and Granduc in '68.
1963	R. Kirkham completed a M.Sc. thesis on the geology of Mitchell and Sulphurets areas.
1981	T. Simpson completed a M.Sc. thesis on the geology of the Sulphurets gold zone.
1971-1977	Granduc Mines conducted additional exploration surveys targeting molybdenum & drilled 6 holes into Snowfield zone (Bruceside)
1979-1984	Esso Minerals optioned Sulphurets property and completed early stage exploration including drilling 14 holes (2275 meters).
1985-1991	Granduc optioned Sulphurets to Lacana (later Corona) and Newhawk Gold Mines. Lacana-Newhawk JV spends ~ $21M developing West Zone and other smaller precious metal veins on Bruceside property. Drilled 11 holes at Sulphurets. Homestake undertook exploration after acquiring Corona.
1991	Arbee prospect optioned by Newhawk from D. Ross.
1992	Arbee prospect optioned by Placer Dome from Newhawk.
1991-1992	Newhawk commissioned AB geophysical survey over Sulphurets. Newhawk subdived Suphurets property into Sulphside and Bruceside. Placer Dome acquires Sulphside (Sulphurets, Mitchell, Iron Cap,
1992	Placer Dome undertook delineation drilling of Sulphurets deposit at 50 meter centers (23 holes).
1993	J. Margolis completed a PhD thesis on the Sulphurets district. Newhawk-Corona drilled 3 holes in the Snowfields and Josephine zones east of Sulphurets.
1992-1996	Placer Dome completed geologic modeling, resource estimation (not NI 43-101 compliant), preliminary met testwork, and scoping studies.
1999	Silver Standard Resources acquired Newhawk Gold Mines.
1996-2000	Sulphurets project was dormant.
2000	Seabridge Gold acquired a 100% interest in the Sulphurets/Mitchell properties from Placer Dome.
2002	Noranda Inc. acquired an option to earn up to 65% from Seabridge.
2003-2004	Noranda Inc. undertook various exploration surveys.
2005	Falconbridge Ltd. (formerly Noranda) completed 4,092 meters of diamond drilling in 16 holes.
2006	Seabridge Gold purchased Falconbridge's option and drilled 29 holes totaling about 9,129m at the Sulphurets and Mitchell zones.
2007	Seabridge Gold puchased Arbee prospect from D. Ross and drilled 37 holes totaling 15650m.
2008	Seabridge Gold puchased Arbee prospect from D. Ross, drilled 37 holes totaling 15,177m, started metallurgical testing, obtained new topographic data, and initiated permit related activities.
2009	Seabridge drilled approximately 13,000m (resource definition, geotechnical and water monitoring), conducted metallurgical testing, and intensified permit data collection.
2010	Seabridge drilled 29 holes totaling about 9,725m (resource definition and geotechnical), conducted metallurgical testing, and intensified permit data collection.

The majority of Seabridge's 2010 drilling campaign concentrated on the Iron Cap zone. Thirty-eight (38) diamond core holes were drilled to augment the previous 11 holes to define Mineral Resources. An additional three holes were drilled for geotechnical purposes.

6.2 Historical Resource Estimates

The author is unaware of any publicly disclosed historical resource estimates for the KSM deposits prior to Seabridge's entry into the district. The author has prepared NI 43-101 compliant Mineral Resources for the Mitchell zone (Lechner, 2007, Lechner, 2008b, Lechner, 2009, and Lechner, 2010). The author has prepared NI 43-101 compliant Mineral Resources for the Sulphurets zone (Lechner, 2008, Lechner, 2009, and Lechner, 2010). The author has prepared NI 43-101 compliant Mineral Resources for the Kerr zone (Lechner, 2008, Lechner, 2009, and Lechner 2010).

6.3 History of Production

There is no known production from the Kerr, Sulphurets, Mitchell, or Iron Cap deposits.

7.0 GEOLOGICAL SETTING

The following section was taken directly from RMI's April 2008 NI- 43101 report (Lechner, 2008b):

"The region lies within "Stikinia", a terrane of Triassic and Jurassic volcanic arcs that were accreted onto the Paleozoic basement of the North American continental margin in the Middle Jurassic. Stikinia is the largest of several fault bounded, allochthonous terranes within the Intermontane belt, which lies between the post-accretionary, Tertiary intrusives of the Coast belt and continental margin sedimentary prisms of the Foreland (Rocky Mountain) belt. In the Kerr-Sulphurets area, Stikinia is dominated by variably deformed, oceanic island arc complexes of the Triassic Stuhini and Jurassic Hazelton groups. Back-arc basins formed eastward of the property in the Late Jurassic and Cretaceous were filled with thick accumulations of fine black clastic sediments of the Bowser Group. Folding and thrusting due to compressional tectonics in the late Cretaceous generated the area's current structural features. Remnants of Quaternary basaltic eruptions occur throughout the region.

Early Jurassic sub-volcanic intrusive complexes are common in the Stikinia terrane, and several host well-known precious and base metal rich hydrothermal systems. These include copper-gold porphyry deposits such as Galore Creek, Red Chris, Kemess, Mt. Milligan, and Kerr-Sulphurets. In addition, there are a number of related polymetallic deposits including skarns at Premier, epithermal veins and subaqueous vein and replacement sulfide deposits at Eskay Creek, Snip, Bruceside, and Granduc.

At Kerr-Sulphurets, Triassic rocks include marine sediments and intermediate volcanics of the Stuhini Group. The lowermost Stuhini Group is dominated by turbiditic argillite and sandstone, which are overlain by volcanic pillowed flows and breccias. The upper portion consists of turbidites and graded sandstones similar to the base strata. The Stuhini Group is separated by an erosional unconformity from the overlying Jurassic sediments and volcanics of the Jack Formation and Hazelton Group. The Jack Formation is comprised of fossiliferous, limey sediments, mudstones and sandstones. The base is marked by a granodiorite and limestone cobble bearing conglomerate. Overlying the Jack Formation is the Hazelton Group, dominated by andesitic flows and breccias deposited in a volcanic chain with high paleotopographic relief. Distinct felsic welded tuff horizons of the Mount Dilworth Formation are an important stratigraphic marker in the Hazelton Group, as they are closely associated with the Eskay Creek deposit.

A variety of dikes, sills, and plugs of diorite, monzodiorite, syenite, and granite are found in the area. Radiometric dating indicates these are of Early Jurassic age and they are collectively referred to as the "Mitchell Intrusions". Below the Sulphurets and Mitchell thrust faults, pre- and intra-mineral intrusives have historically been very difficult to differentiate due to intense hydrothermal alteration. Above the faults there are a number of sills and plugs of coarse-grained feldspar porphyritic monzonite to low-silica granite that intruded siliceous hornfelsed sediments and volcanics. Copper and gold mineralization is *typically best developed at the margins of these intrusions. There appear to be both pre-, intra-, and post-mineral phases of mineralization.*

Figure 7-1 is a generalized geologic map of the KSM district showing lithology, alteration, major structures, drill hole collar locations, and gold equivalent mineralized zones.

SEABRIDGE GOLD
Figure 7-1
KSM GEOLOGIC MAP
UTM Nad83 Zn.09
Scale = 1:12,500
ms / 20/03/11
0
0.5
1
kilometers
N
Altered and Mineralized Units
Iron Cap - Porphyritic Monzonite dykes, intrusive breccias, and altered wall rocks, generally moderately mineralized with both Au and Cu from "Iron Cap" mineralizing event
Iron Cap - A distinct zone of relatively high Mo, average Cu, low Au, and high Ag mineralization beneath the Iron Cap fault, in rocks of the North Mitchell panel
Iron Cap - Steep, meter-scale, massive to semi-massive quartz veins, tend to pinch and disappear at depth
Iron Cap - lower zone mineralization, disseminated, stockwork, and vein hosted fine grained chalcopyrite, generally ~0.2% Cu and ~0.5g/t Au, forms bulk of Iron Cap zone. includes indistinguishable volcanics, sediments, and intrusive rocks, usually intensely silicified, minor remnant potassic alteration.
Iron Cap - upper zone mineralization, disseminated and stockwork hosted fine grained chalcopyrite, generally >0.1% Cu with variable Au, silicified volcanics, sediments, and intrusive rocks.
Iron Cap - low grade mineralization, mostly below lower zones. Cu and Au values are quite variable, but on average are lower than Upper and Lower zones. Intensely silicified volcanics, sediments, and intrusive rock.
Iron Cap - fine grained intermediate intrusive and/or related flows, generally finely porphyritic, chloritized, patchy remnant potassic alteration, variably mineralized, probably closely related to similar intrusive observed in deeper portions of Mitchell zone
Mitchell Zone - fine grained intermediate volcanics and/or related intrusive, disseminated, stockwork, and vein hosted fine grained chalcopyrite with gold, generally finely porphyritic, chloritized, patchy remnant potassic alteration
Kerr Zone - Indistinguishable fine grained intermediate volcanics and/or related intrusive, sediments, disseminated, stockwork, and vein hosted fine grained chalcopyrite with gold, chloritized, rare remnant potassic alteration, variably mylonitized
Sulphurets Breccia Gold zone - mostly gold bearing pyritic mineralization with minor chalcopyrite and sulphosalts in a K-feldspar-siliceous hydrothermal, rotational to crackle breccia, with an intense phyllic overprint
Sulphurets Leached zone - a halo of altered rocks surrounding the Sulphurets Breccia Gold zone depleted in gold and copper, probably due to hypogene leaching
Sulphurets Lower Gold zone - a zone of weaker alteration and mineralization beneath the Raewyn Copper zone, with a distinctly higher Au to Cu ratio
Raewyn Copper-Gold zone - mostly porphyry style disseminated chalcopyrite and associated gold mineralization in moderately quartz stockworked, chlorite-biotite-sericite magnetite altered volcanics and sediments, alteration and mineralization are centered on a conformable body of porphyritic quartz monzonite
Silicified and Hornfelsed cherty sediments above the Sulphurets zone
Silicified and Hornfelsed andesitic volcanics above the Sulphurets zone
Intensely quartz-sericite-pyrite-clay altered rock, typically schistose and mylonitized. At the Mitchell zone, it may host multiple phases of deformed quartz stockwork veining with associated copper and gold mineralization
Altered, mineralized rock with >50% by volume of quartz stockwork veinlets, generally deformed, crackled, sulfide bearing
Snowfield zone advanced argillic alteration
Propylitic hornfelsed, weakly mineralized, volcanics and sediments of uncertain age, beneath MTF
Propylitic hornfelsed, weakly mineralized, volcanics and sediments of uncertain age, between STF and MTF
Propylitic hornfelsed, weakly mineralized, volcanics and sediments of uncertain age, above STF
Gossanous weathering
Late Jurrasic Intrusions
Fine grained to aphanitic dioritic dykes
Coarsely feldspar porphyritic monzonitic
Coarsely feldspar-hornblende porphyritic "Premier" type intrusions
Plagioclase-augite diorite
Plagioclase-hornblende diorite to microdiorite or massive andesite
Jurassic Hazelton Group
Porphyritic dacite flow dome complex
Mt. Dilworth Fm. felsic pyroclastics
Green and maroon laminated tuff
Green and maroon pyroclastic flow
Siltstone, limestone, chert
Andesite and basalt
Undivided Hazelton Group sediments and volcanics
Silicified, pyritic, hornfelsed Hazelton Group
Silicified, pyritic, hornfelsed, mostly Jack Formation sediments
Jack Formation sediments
Triassic Stuhini Group
Andesite
Sediments, volcaniclastics
Undivided Stuhini Group sediments and volcanics
Mineralized Zones
Near Surface Block Model
Au Equivalent (g/t)
> 1.5
1.0 - 1.5
< 0.5
IRON CAP ZONE
IRON CAP FAULT
MITCHELL THRUST FAULT
MITCHELL ZONE
SNOWFIELD ZONE
BRUCEJACK FAULT
SULPHURETS ZONE
SULPHURETS THRUST FAULT
Sulphurets Lake
CAMP
SEABRIDGE PROPERTY BOUNDARY
KERR ZONE
BRIDGE/BRUCEJACK ZONES
Modified from Margolis, 1993
Modified from Tech. Rpt. and Prelim. Assess. of the Snowfield-Brucejack Project, Pretium Res., 2010
Modified from MDRC Spec. Pub. No. 1, Ch. 22, Bridge, 1993

8.0 DEPOSIT TYPES

The following section was taken directly from RMI's April 6, 2007 NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia" (Lechner, 2007):

"The KSM property hosts an extensive alteration and mineralization system that was developed as a result of hydrothermal activity focused on hypabyssal, Early Jurassic "Mitchell" intermediate, porphyritic intrusions. The model is best described as a gold-enriched copper porphyry system controlled by a series of dikes, sills and plugs rather than a single stock. Mineralization is typically associated with quartz veinlet stockworks and sheeted quartz veinlet arrays mainly in altered host rocks adjacent to the intrusions. Drilling and surface rock chip sampling confirms that the alteration and mineralization is continuous over distances of hundreds of meters. Less commonly, mineralized intrusive-hydrothermal breccias cut through previously veined and mineralized rocks. Principal sulfides are pyrite and chalcopyrite, with minor molybdenite, and trace amounts of tennantite, bornite, sphalerite, and galena. All mineralization is hypogene, except for a small remnant of preserved supergene mineralization at the south end of the Kerr deposit which hosts some chalcocite enrichment, and at the Main Copper (Sulphurets) occurrence where a remnant of leached capping and oxide mineralization is preserved at the highest elevations.

At Mitchell and Sulphurets, copper-gold mineralization is fine grained, pervasive, homogeneous, and continuous for several hundred meters along strike and depth extents. Preliminary work indicates gold is intimately associated with chalcopyrite. The unusually homogeneous nature of the mineralization over large extents may be the result of post-mineral metamorphism and re-distribution of metals during Early Jurassic or Cretaceous deformational events. At Sulphurets, mineralization is somewhat less continuous than Mitchell, where sharp contrasts in grade occur between structurally controlled hydrothermal breccias and alteration zones."

9.0 MINERALIZATION

The Jurassic island arc setting was conducive to shallow emplacement of intrusives and hydrothermal cells. At KSM, large, coalescing hydrothermal alteration haloes developed around nested volcanic-intrusive complexes.

9.1 **Kerr Zone**

The Kerr deposit has been delineated by over 29,020 meters of core drilling in 155 drill holes spaced at intervals of 50 to 100 meters by Seabridge and six previous operators between 1987 and 2010. In 2010, Seabridge drilled an additional 1,453 meters in four core holes. These drill holes were primarily completed to assist with ongoing geotechnical engineering studies. Two of the four holes intersected the previously designed resource. Geologic and assay results were consistent with existing models of geology and metal distribution as described below.

Fine disseminated, fracture and veinlet controlled chalcopyrite mineralization, with minor bornite, chalcocite and tennanite, is associated with intrusion of Early Jurassic monzonite porphyry into Triassic sediments and volcaniclastics, and accompanying hydrothermal alteration. There is a strong phyllic overprint with a high pyrite content, generally 5 to 20%. In many respects, the deposit bears little resemblance to a classic porphyry deposit; however it has been referred to as a porphyry-type deposit since 1987. Later studies (see bibliography) indicated that mineralization was localized around one or more previously unrecognized monzonite intrusions, and is adequately described as a modified porphyry deposit. Most of the following description has been extracted and modified from the paper by Ditson, et al, 1995."

The Kerr deposit is a strongly deformed copper-gold porphyry, where copper and gold grades have been upgraded due to remobilization of metals during later and/or possibly syn-intrusive deformation. Alteration is the result of a relatively shallow, long lived hydrothermal system generated by intrusion of monzonite. Subsequent regional deformation along the Sulphurets thrust was diverted into Kerr area along pre-existing structures and altered rocks with low competency.

The mineralized area forms a mostly continuous, north-south trending and westerly dipping, irregular body at least 1700 meters long, and up to 200 meters thick. Higher grades are associated with crackled quartz stockwork, anhydrite veining, and chlorite alteration. It is enveloped by a schistose, pyrite rich phyllic alteration with low to moderate grades. Mineralization is open at depth and along strike.

The surface expression of the deposit is a large, strongly leached schistose, pyritic gossan. Soil geochemistry shows elevated anomalous gold values over the deposit, and a halo of anomalous copper values. Induced polarization detects high chargeability and low resistivity coincident with mineralization.

9.1.1 **Lithology and Structure**

The majority of the host volcaniclastic and sedimentary rocks belong to the Stuhini Group which is highly schistose within the deposit. Where they are undeformed, the sedimentary rocks consist primarily of coarse conglomerate, siltstone, mudstone and minor greywacke. Undeformed volcaniclastic rocks are not present within the deposit but outcrops nearby contain well-bedded, sandy tuffs to coarse volcanic conglomerate. The presence of strongly flattened clasts was used to assign a volcaniclastic origin. Within the core of the deposit, deformation and alteration preclude assignment of protolith, and either "sericite schist" or "chlorite schist" is usually the most appropriate term.

Monzonite intrusions are plagioclase-hornblende-biotite porphyries with common apatite microphenocrysts. Primary hornblende and biotite are not observed, but are recognized as hydrothermal chlorite and sericite pseudomorphs. Plagioclase phenocrysts are variably altered to sericite and have diffuse boundaries. Where alteration and deformation are intense, identification of monzonite may hinge on the recognition of plagioclase or hornblende phenocrysts alone. Several intrusive phases appear to be present, including breccias at the margins, but cannot be distinguished clearly by their mineralogy.

Monzonite is probably part of the "Mitchell Intrusions", which belong to the Early Jurassic Texas Creek plutonic suite. This age is inferred by previous workers from the close relationship between monzonite and porphyritic dikes. Monzonite appears to be most abundant in the lower reaches of the deposit, but it is also the suspected protolith for much of the strongly altered material in the upper central portions.

A large area of barren plagioclase porphyry and intrusive breccia occurs in the southeastern corner of the deposit. Alteration includes pervasive chlorite, epidote, sericite and carbonate. K-feldspar is a primary component in the groundmass of some porphyries. The contact between these rocks and mineralizing monzonite is probably a fault.

Plagioclase hornblende porphyry dikes and intrusions similar to the host monzonite are most abundant in the southern half of the deposit. They are generally massive and barren or only weakly mineralized and are inferred to be late phases of the same magma.

Meter-scale, barren albite megacrystic porphyry dikes intrude the deposit along generally north-south trends. Hyalophane megacrystic dikes intrude along east-west trends. These dikes likely correlate with "Premier porphyry" dikes of the Texas Creek plutonic suite commonly associated with copper and gold mineralization throughout the region. Aphanitic andesite dikes are common throughout the deposit, and are highly altered, massive, dark green, and composed of plagioclase, chlorite, ilmenite and sericite. These dikes generally cross-cut schistosity, but many folded dikes have been observed on the surface.

Eocene kersantite, andesite and monzonite dikes up to 3 m wide intrude the deposit along the northerly foliation trend. These are composed of highly variable amounts of biotite, fine-grained plagioclase, chlorite, tremolite/actinolite, quartz and K-feldspar. Coarse white carbonate and possible barite occur as local amygdules, especially along contacts.

The Kerr deposit occurs within a major northerly trending structural zone with strong foliation and widespread shearing. Individual structures within the deposit are masked by pervasive alteration and deformation.

9.1.2 **Alteration**

Abundant pervasive sericite occurs throughout the deposit, which is accompanied by chlorite replacement of mafic minerals in the main monzonite intrusion. Outward from this, strong chlorite-sericite alteration contains more pervasive chlorite than sericite.

Yellow and grey sericite alteration types occur peripheral to these two chlorite-bearing types. Sericite is commonly twice as abundant as chlorite. In drill core, zones of pale green sericite-dominant alteration are common. Patchy quartz is present in amounts varying from 5% to 15%. Pyrite content is generally less than 10%.

Dark green, pervasive chlorite-dominant alteration occurs around the margins of the main monzonite intrusion. It most commonly occurs between sericite-chlorite and intense grey sericite zones and may represent an alteration front. Up to 60% dark chlorite is accompanied by up to 30% sericite. Patchy quartz (5% to 15%) may locally represent dismembered veins. Anhydrite is most visible as white to pink coarsely crystalline veins up to several centimeters wide. Pyrite content is only 1% to 7%. Primary biotite phenocrysts have been replaced by chlorite. Apatite grains up to 15 mm are locally present in some of the most strongly altered zones.

Pervasive grey sericite alteration is characterized by 40% to 60% grey sericite with 5% to 10% quartz and 0% to 7% chlorite. Fine-grained plagioclase is commonly present in amounts varying from 20% to 50%, but much less where quartz is dominant. Intensity of alteration and deformation are such that the rock is best described as sericite or quartz-sericite schist. The pyrite content can be as high as 15%, especially in volcaniclastic rocks.

Pervasive yellow sericite alteration is a peripheral assemblage affecting only the Stuhini Group, primarily in the footwall below the main stockwork zone. This has the lowest average copper grade of all the pervasive alteration types. This style typically contains 5% to 15% original plagioclase, 30% to 60% yellow sericite, 10% to 20% quartz, and 10% to 20% pyrite. Yellow sericite commonly wraps around rounded quartz fragments, giving these rocks an augen-like, granular appearance. Green sericite commonly occurs in minor amounts as a replacement of selected clasts. As alteration and deformation weaken, pervasive sericite changes from yellow to green, and gradually disappears as sedimentary textures become clear.

Anhydrite veining is most commonly associated with chlorite bearing alteration types. It is characteristic of texturally destructive chlorite-sericite alteration and the upper portions of sericite-chlorite altered monzonite. Anhydrite veins locally carry minor chalcopyrite. During deformation, anhydrite was remobilized into irregular, crosscutting networks of veinlets that post-date all other vein types. Anhydrite has hydrated to gypsum to depths of up to 250 meters, and leaching by groundwater has produced large areas of voids and broken rock called "rubble." Core recovery in these zones is poor.

9.1.3 **Mineralization**

The most important mineralization type is quartz stockwork, which drapes over the main monzonite intrusion and extends a considerable distance down the eastern side, along the footwall of the deposit. Deformation of mineralized quartz veins has resulted in segregation of sulphides into interstices between granular recrystallized quartz, resulting in a 'crackled' texture. Chalcopyrite also occurs as fracture fillings in an earlier generation of coarse vein pyrite. Narrow veins and veinlets are commonly highly contorted. The quartz stockwork veins may contain any combination of pyrite, chalcopyrite, bornite, tetrahedrite, tennantite or rare enargite. Thin films of secondary digenite and chalcocite are also present, but are only locally significant near the surface. Small flakes of possibly primary crystalline covellite are locally abundant, especially in rubbled zones and near-surface areas.

In addition to crackled quartz stockwork, mineralization is hosted by several other types of veinlets. Ditson et al, suggest the following vein classification for Kerr:

- pyrite±quartz, sericite, minor chalcopyrite (predeformation)
- quartz±pyrite, carbonate, anhydrite, sericite, chlorite, chalcopyrite (predeformation)
- anhydrite±chalcopyrite (predeformation)
- carbonate±minor chalcopyrite, bornite (syn/postdeformation)
- quartz+carbonate, chlorite, chalcopyrite (postdeformation)

Chlorite-bearing alteration types host the greatest variety of vein types. Mineralization grading over 0.4% Cu is generally located within or adjacent to crackled quartz stockwork, however there are significant tonnages in non-stockwork mineralization grading over 0.4% Cu in the northern sector in monzonite below the stockwork. All mineralization grading over 1% Cu occurs within stockwork. The Au:Cu ratio (g/t:%) for all rocks grading over 0.4% Cu averages 0.4.

Molybdenum values were analyzed are most commonly less than 100 ppm, but range up to 423 ppm. Molybdenite is associated with chloritic alteration, and in the northern sector yellow sericite altered rocks below monzonite.

9.1.4 **Structure**

The Kerr deposit occurs within a major northerly trending structural zone with strong foliation and widespread shearing. Individual structures within the deposit are masked by pervasive alteration and deformation.

9.1.5 Revised Three-dimensional Geologic Models

In 2010 revised geological models were interpreted by Seabridge geologists on sections spaced at approximately 50 meter intervals, and assembled to construct three-dimensional wireframes for grade estimation. Models for lithology, alteration, and the “rubble” zone are based on the framework of Bridge, 1995. The model incorporates data from Seabridge’s core logging of drill holes from 2009 and 2010, and re-logging of historical holes by Placer Dome geologists in 1992 which the work of Bridge incorporates.

Brief descriptions of revised geological units using wireframe coding follow:

Lithology

OVERBRDN – Unconsolidated soils, talus, moraine, and slumped rock.

HW_INTR – Fine grained to porphyritic late dioritic intrusions in the hanging wall, generally barren or poorly mineralized.

HBL_DYKE – Hornblende porphyritic late dioritic dyke in the footwall, generally barren

PREMDYKE – Coarse feldspar porphyritic late monzonite dyke (“Premier” type), mostly barren, meter to decimeter scale, that cuts through and parallels the center of the Kerr zone, and dips similarly at about 55 to 65 degrees west.

HW_MIXED – Hydrothermally altered, mineralized, deformed zone of almost indistinguishable sediments, volcanics, and monzonitic dykes, comprising the majority of the Kerr deposit. The dykes are intra-mineral and associated with the main alteration-mineralizing phase. HW (hanging wall) denotes Kerr zone above the PREMDYKE.

LW_MIXED – Similar to HW_MIXED, but comprises rocks of the Kerr zone below the PREMDYKE.

HW_UNCAT – Uncategorized rocks above the PREMDYKE, mostly weakly altered and mineralized sediments and volcanics, interpreted to belong to the Triassic Stuhini Group. Portions of this unit are sufficiently mineralized to be included in the resource.

FW_UNCAT – Same as HW_UNCAT but below the PREMDYKE.

STUHVOLC – Unaltered and unmineralized volcanics and sediments of the Stuhini Group, stratigraphically and topographically above the Kerr zone.

Alteration

CL_ALTN – Chlorite dominant alteration, with remnants of earlier potassic alteration, and patchy sericitic alteration. Closely associated with chalcopyrite and gold.

QSP_ALTN – Quartz-sericite-pyrite alteration, with patchy chlorite. Generally strongly deformed, schistose and mylonitized, with minor chalcopyrite and gold.

DFRMD_ZN – Grossly defined envelope of weaker deformation, alteration and mineralization surrounding above alteration types.

QTZ_CRACK – Defines limits of best developed and continous quartz-sulfide vein stockwork, closely associated with chalcopyrite and gold. Veins typically show strong crackling related to syn- or post-mineral deformation. Crackles lined with sulfides.

RUBBLE – Zone of rubbleized rock within and close to the Kerr zone. Due to conversion of anhydrite to gypsum lining fractures and in veinlets and subsequent dissolution near surface.

ANHY_GYP – Mapped extents of preserved anhydrite and gypsum veinlets at depth.

<u>Ore Types</u>

Table 9-1 summarizes the currently recognized ore types from the Kerr zone.

Table 9-1: Kerr Ore Types

Ore Type	**Description/Source**
Rubble Zone	Kerr zone mineralization with anhydrite/gypsum, secondary chalcocite, poor rock quality, dominant at south end of zone
Gold leach breccia	Kerr zone mineralization, qtz-ser-chl-py-cpy altered crackle quartz stockwork veintlets, mylonitized, relatively competent
Uncategorized	Undrilled areas

Figure 9-1 is a geologic plan map of the Kerr zone showing lithology and alteration. Assay grades are shown as histograms along the drill hole traces (copper shown in green and gold grades shown in orange). Figure 9-2 is an east-west cross section through the southern portion of the Kerr zone showing drill hole traces/grades, lithology, alteration, and gold mineralization. Figure 9-3 is a similar cross section through the Kerr deposit showing copper mineralization. Both cross sections also show Indicated Resource block model grades (inside of the heavy green line). The line of section for the two cross sections is shown on the plan map (Figure 9-1) in reddish-brown.

Figure 9-1: Kerr Geologic Plan Map

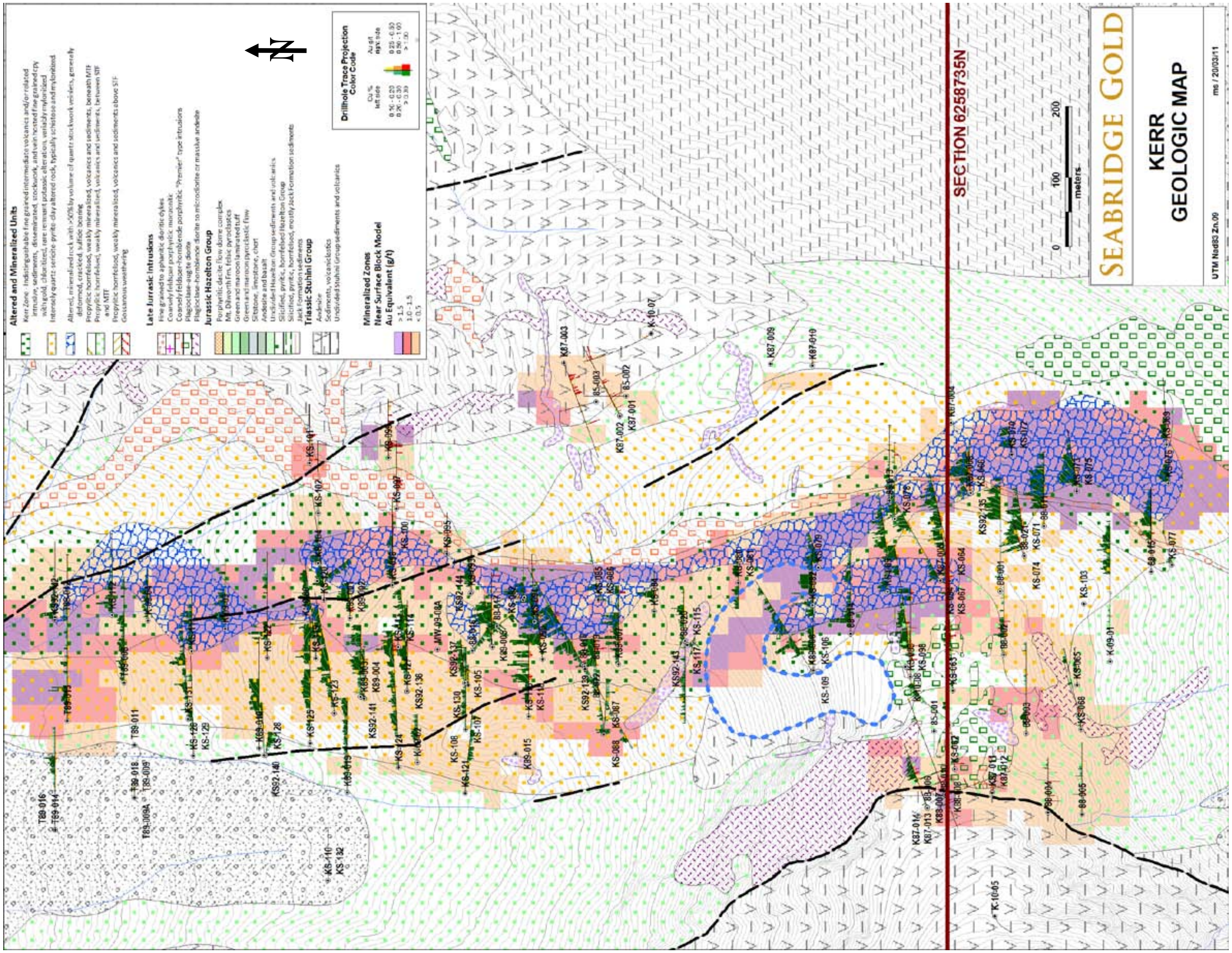


Figure 9-2: Kerr Geologic Section 6,258,375 N (Au)

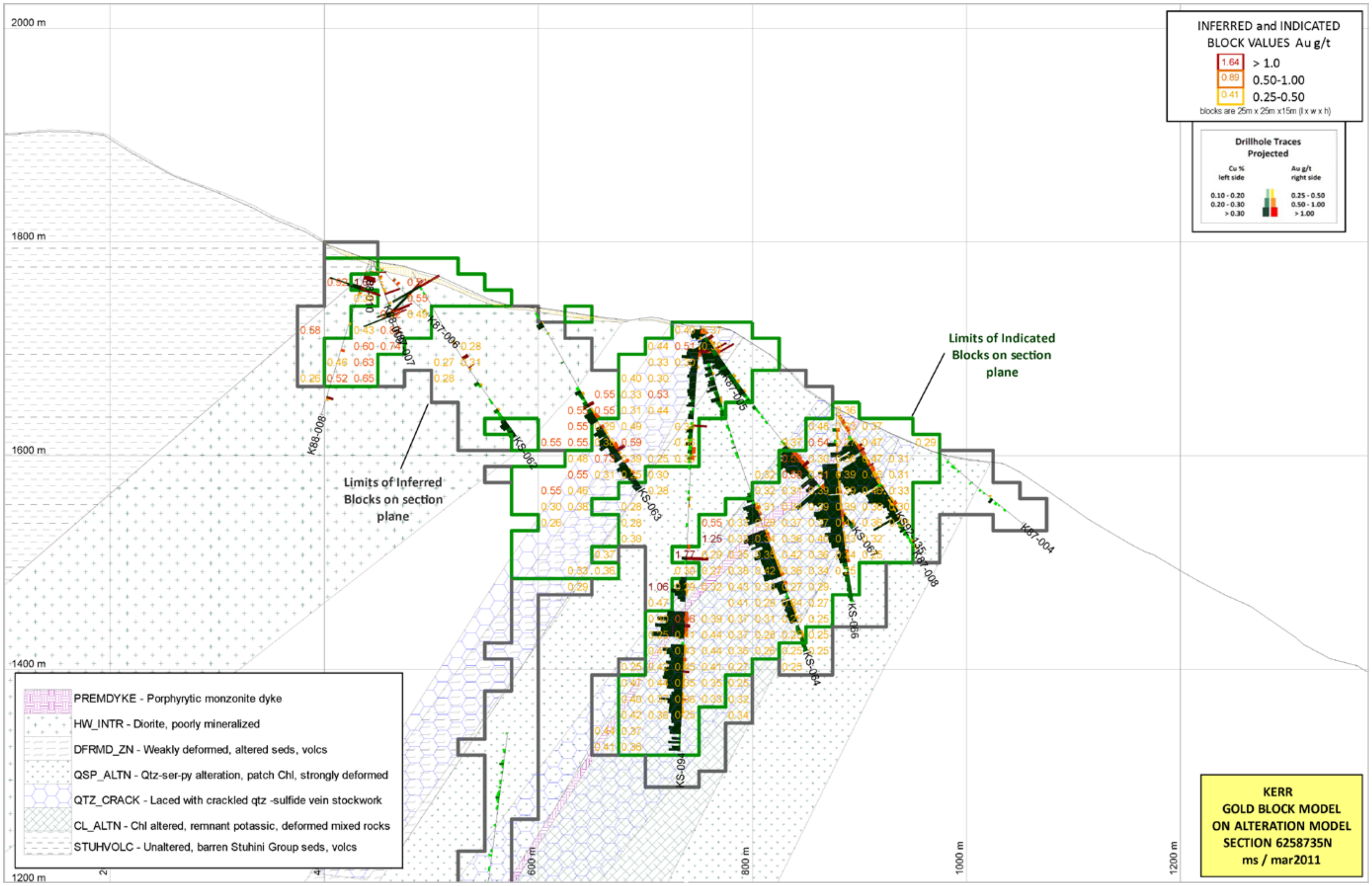
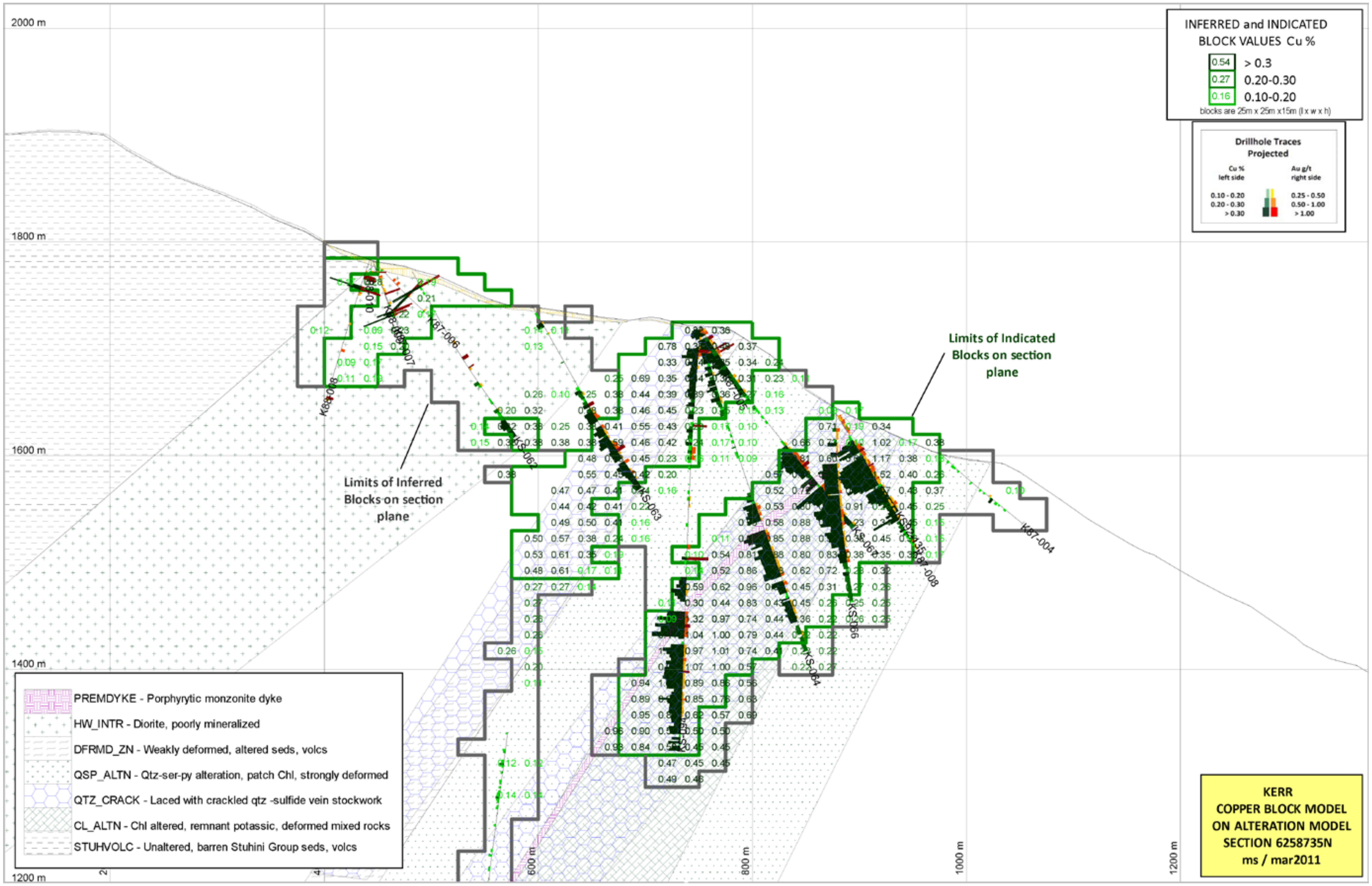


Figure 9-3: Kerr Geologic Section 6,258,375 N (Cu)



9.2 Sulphurets Zone

The Sulphurets deposit has been delineated by over 25,281 meters of core drilling in 94 drill holes spaced at intervals of 50 to 100 meters; in total six different operators drilled the project between 1968 and 2010.

In 2010, Seabridge drilled 6,538.9 meters in 18 core holes to upgrade Inferred Resources to Indicated Resources and to test the continuity of mineralization down-dip and along strike to the southwest. Geological and assay results were consistent with existing models for geology and metal distribution as described below.

The deposit is comprised of two distinct zones, Raewyn and Breccia Gold. The Raewyn Copper-Gold zone hosts mostly porphyry style disseminated chalcopyrite and associated gold mineralization in moderately quartz stockworked, chlorite-biotite-sericite-magnetite altered volcanics. The alteration and mineralization are centered on a narrow, apparently conformable body of porphyritic quartz monzonite. It has an apparent northeasterly strike and dips about 45 degrees to the north. It may be offset in en echelon style by several north-northeasterly trending vertical structures. The mineralization is open at down-dip and along strike to the southwest. The Breccia Gold zone hosts mostly gold bearing pyritic mineralization with minor chalcopyrite and sulfosalts in a K-feldspar-siliceous hydrothermal breccia that apparently crosscuts the Raewyn porphyry copper-gold deposit. It comprises altered intrusive clasts in a matrix of mainly silica and sulfides. Both zones have an intense phyllic overprint that nearly masks all earlier alteration phases. According to Fowler, et al (1995), the breccia zone has an apparent northerly strike and dips to the west, and is open down dip. A late, barren, pyritic monzogabbro cuts off the Breccia zone on the northwest side. Most of the following description has been extracted and modified from the paper by Fowler and Wells, 1995. Figure 9-4 is a generalized geological plan showing the surficial geology of the Sulphurets deposit along with drill holes and the approximated surface trace of 0.30% copper mineralization. Figure 9-5 is a northwest-southeast trending cross section through the Sulphurets deposit.

9.2.1 Lithology and Structure

The Sulphurets deposit (or Sulphurets Gold zone) formed in a high level, transitional porphyry copper-gold system that was over thrust by the deeper levels of a syenite-centered porphyry copper-gold deposit (Main Copper zone) along the Sulphurets Thrust Fault (STF). Volcanic sequences on either side of the thrust have been assigned to Hazelton Group. Below the STF the volcanics consist of propylitic to potassic altered, massive to tuffaceous trachyandesites, with local sediments, intruded by northerly-trending feldspar porphyry dikes. Trachyandesite crystal and ash tuffs, flows, and breccias are interlayered with dark argillites, volcanic derived sandstones, cherts, and cherty tuffs. Generally, in areas of intense alteration and mineralization, the protolith cannot be assigned accurately. Late hornblende phyric monzonite to monzogabbro dikes and sills intrude the area.

The Sulphurets Gold zone is centered along the Raewyn Fault, a zone of strong faulting and phyllic-quartz-sericite-pyrite, intermediate argillic, and potassium silicate alteration. The Raewyn Fault trends northeasterly, subparallel to the STF, and is well exposed for much of its length along the main cliff, forming a prominent gossan. Copper-gold mineralization is usually coincident with areas of strongest fracturing and potassium silicate alteration. At the southern end of the “Raewyn panel”, auriferous hydrothermal breccias constitute the Breccia Gold zone.

Above the STF, intermediate volcanics, massive green flows and tuffs are intruded by feldspar porphyry quartz syenites and potassic monzonite dikes. Rocks in the periphery of the dikes are K-feldspar altered and contain disseminated and fracture controlled chalcopyrite. The dikes are grouped with the Mitchell intrusions that correlate with late Jurassic Texas Creek intrusions common throughout the region.

Brittle fracturing typical of hornfelsed aureoles is widespread in the upper plate rocks, and numerous northerly to north-northeasterly striking, steep-westerly dipping fractures and fracture zones are present. Below the STF, the most prominent feature is the subparallel, northeasterly-dipping Raewyn structural-alteration panel. This panel is separated from the STF by a 100 m to 200 m wide section of less deformed and less altered volcanic rocks. It is transected by shallowly and steeply dipping fault sets, some of which are intra-mineral and others post-mineral. Bedding, where visible, dips at fairly steep angles to the north and northwest but it is not as steep as the sub-vertical foliation.

9.2.2 **Alteration/Mineralization - Raewyn Copper-Gold Zone**

Gold and copper mineralization here is associated with the main Raewyn dike. Average copper and gold values from the mineralized zones below and within the Raewyn panel are fairly consistent. Copper values range from 0.3% to 0.7% and gold values are 0.4 g/t to 1.2 g/t. Strong quartz-sericite-pyrite (phyllic) alteration largely overprints pre-existing assemblages, however a considerable amount K-feldspar is present from an early widespread potassic alteration event. Outboard from the quartz-sericite-pyrite alteration the volcanic rocks are chlorite-altered and locally contain epidote, magnetite and variable carbonate (propylitic).

Multiphase brecciation, alteration, veining and widespread recrystallization characterize the zone. Vein assemblages include:

1) chalcopyrite, quartz, chlorite, sericite ± albite and carbonate,
2) chalcopyrite, quartz, pyrite, biotite, sericite, minor chlorite and molybdenite,
3) milky quartz veins with coarse blebby chalcopyrite, minor pyrite and chlorite

Below the Raewyn panel, biotite alteration with chalcopyrite may extend for ten or more meters from the intrusion into the wallrocks, and overprints earlier K-silicate assemblages. Locally, siliceous-biotite hydrothermal breccias occur within the panel. Heterolithic, siliceous hydrothermal breccias have significant gold values and little copper and may have associated dark tourmaline. Late high-angle quartz veins, up to 3 m wide, occur throughout most commonly close to faults and cross-cut all alteration domains. They contain coarse chalcopyrite, elevated gold grades, pyrite, tetrahedrite ± arsenopyrite and molybdenite.

9.2.3 **Alteration/Mineralization -The Breccia Gold Zone**

Ditson, et al (1995) suggest the following sequence of events in the Breccia zone area:

1) intrusion of Raewyn monzonite followed by
2) main phase of hydrothermal breccias with K-feldspar alteration and
3) late-stage siliceous hydrothermal activity with local breccia pipes.

The K-feldspar hydrothermal breccias are characterized by numerous, mm scale, subangular to rounded, groundmass supported mono to heterolithic fragments in a K-feldspar rich groundmass. Pyrite content ranges from 5% to 20%, and gold content ranges from 0.12 g/t to 5.6 g/t, averaging 1.16 g/t; avg. copper content 0.10%. The siliceous breccias are dominated by aphanitic, siliceous and pyritic groundmass, rare chalcopyrite, and variable gold content ranging from 0.10 g/t to 21.20 g/t, averaging 1.52 g/t. Both breccias locally contain significant amounts of dark coloured tourmaline aggregates and rosettes.

9.2.4 **Revised Three-dimensional Geologic Models**

In 2010 revised geological models were interpreted by Seabridge geologists on sections spaced at approximately 50 meter intervals, and assembled to construct three-dimensional wireframes for grade estimation. The models reflect distinct mineralized domains controlled by lithology, alteration, and structure. The model incorporates data from Seabridge's core logging of drill holes from 2006 to 2010, and re-logging of historical holes by Placer Dome geologists in 1992.

Brief descriptions of revised geological units using wireframe coding follow:

UP_RAECU – Upper Raewyn Copper-Gold zone, mixed zone of mainly chloritic, potassic, and silica altered volcanics, with minor sediments, and thin monzonitic sills. Tabular, moderately dipping, stratigraphically controlled alteration and chalcopyrite-gold mineralization. Comprises bulk of Sulphurets resource.

LW_RAECU – Lower Raewyn Copper-Gold zone, similar to above, but generally thinner and lower grade. Extends southwest towards and encompasses part of Canyon zone. Structurally down-dropped relative to Upper Raewyn Copper zone along several steep west-northwest dipping, short displacement, normal faults in en-echelon pattern.

SULPH_AU – Sulphurets Breccia Gold zone, interpreted to be a late pipe-like feature cutting through the Upper Raewyn Copper-Gold zone which incorporates intense, matrix-supported rotational breccia and peripheral jigsaw and crackle breccia.

LCH_AU – A distinct halo surrounding portions of the Sulphurets Breccia Gold zone, with much lower average gold and copper grades, attributed to hypogene leaching associated with hydrothermal fluid flow patterns associated with emplacement of the breccia.

LW1_AU – A peripheral zone of weaker alteration and mineralization, with a higher Au to Cu ratio, roughly parallel to and beneath the Upper Raewyn Copper-Gold zone. Potassic alteration mostly absent.

LW2_AU – Similar to above, surrounds the Lower Raewyn Copper-Gold zone and encompasses the Canyon zone.

MC_MONZ – Feldspar porphyritic monzonite above the Sulphurets thrust fault. Interpreted to intrude previously mineralized, hornfelsed volcanics and sediments. The core of the intrusive is generally barren, but weak mineralization has been assimilated in the contact areas especially where brecciated.

HAZLVOLC – Undivided volcanics and sediments above the Sulphurets thrust fault, which earlier interpretations considered part of the Jurassic Hazelton Group. Hornfelsing is widespread, and patchy areas of stronger chlorite-magnetite-pyrite alteration also contain some chalcopyrite and associated gold.

9.2.5 **Ore Types**

Table 9-2 summarizes the currently recognized ore types from the Sulphurets zone.

Table 9-2: Sulphurets Ore Types

Ore Type	**Description/Source**
Gold breccia	Hydrothermal Sil-kspar sulphide breccia, includes wall rock and intrusive - wireframe model
Gold leach breccia	Gold breccia wall rocks with distinctly lower grades due to hypogene leaching - wireframe model
Raewyn Copper	Propylitic altered, hornfelsed volcanics and sediments, and some altered intrusive generally above 0.15% Cu - wireframe model
Hazelton Volcanics	Propylitic altered to skarn hornfelsed volcanics and sediments, Main Copper Zone above Sulphurets thrust fault
Monzonite	Monzonite wireframes above Sulphurets thrust fault
Late mafic intrusions	Diorite wirefame model
Uncategorized	Undrilled areas

Figure 9-4 is a geologic plan map of the Sulphurets zone showing lithology, alteration, and structure. Assay grades are shown as histograms along the drill hole traces (copper shown in green and gold grades shown in orange). Figure 9-5 is a northwest-southeast trending cross section through the central portion of the Sulphurets zone showing drill hole traces/grades, lithology, alteration, and gold mineralization. Figure 9-6 is a similar cross section through the Sulphurets deposit showing copper mineralization. Both cross sections also show Indicated Resource block model grades (inside of the heavy green line). The line of section for the two cross sections is shown on the plan map (Figure 9-4) in reddish-brown.

Figure 9-4: Sulphurets Geologic Plan Map

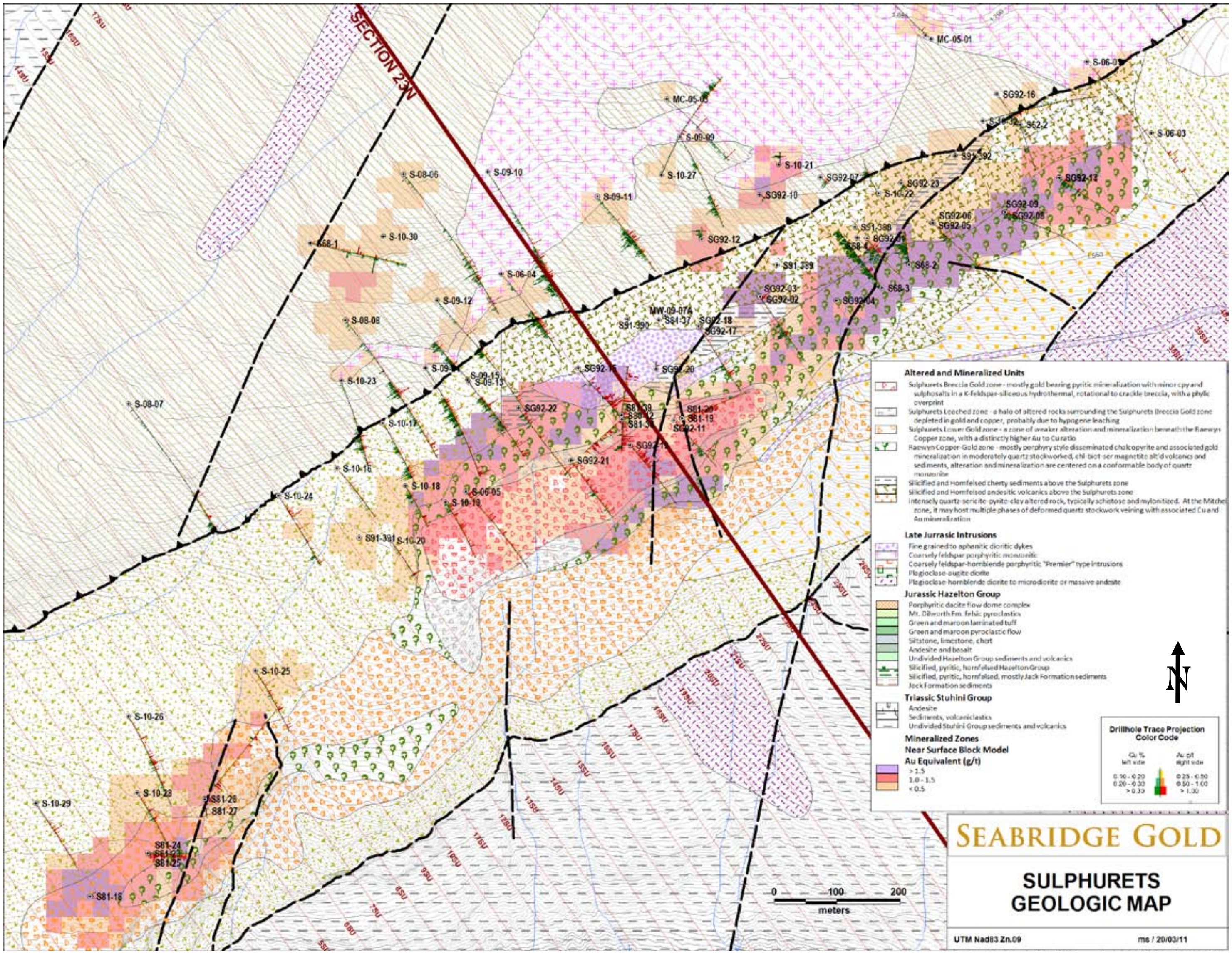


Figure 9-5: Sulphurets Geologic Section 23 (Au)

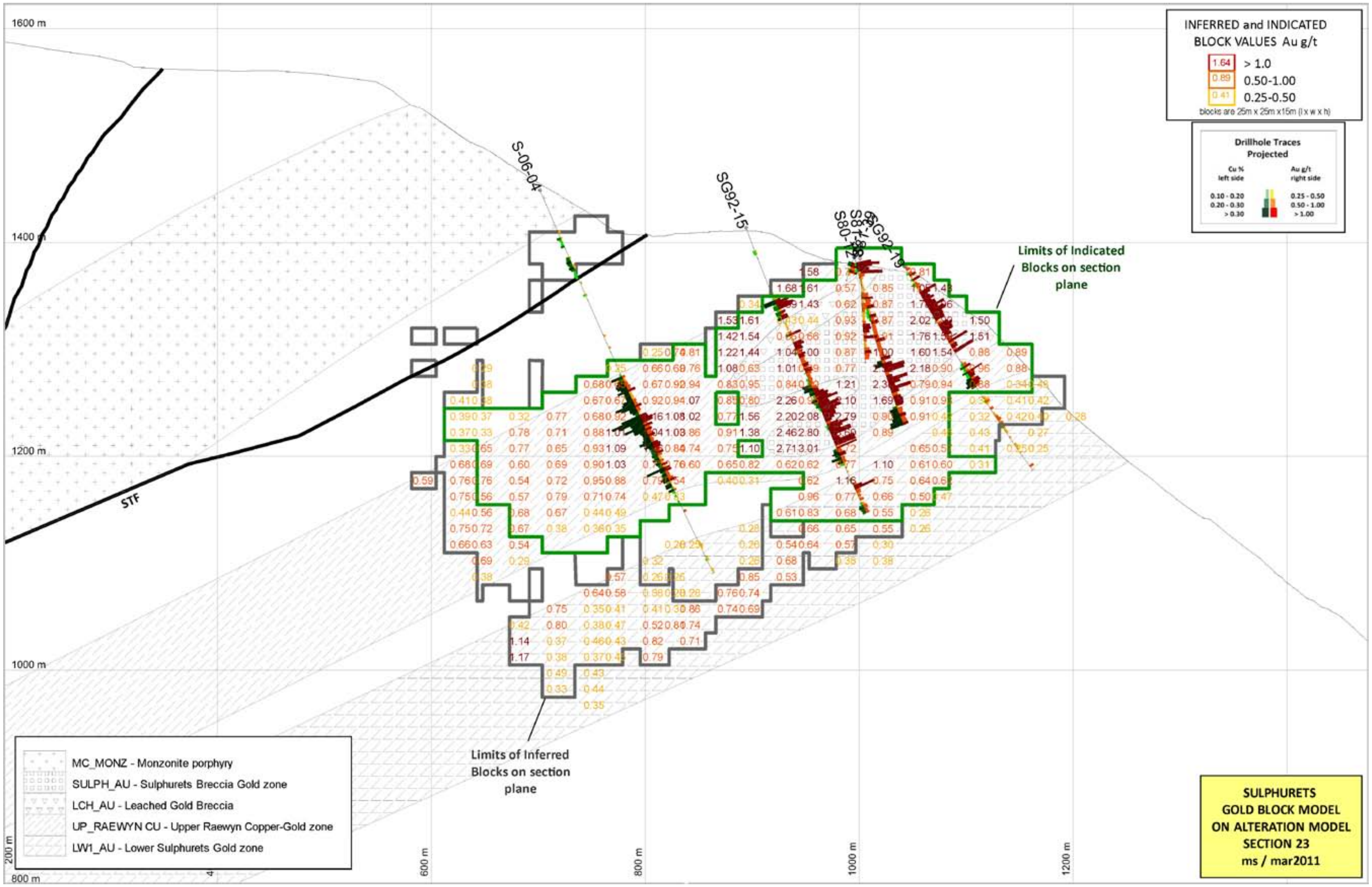
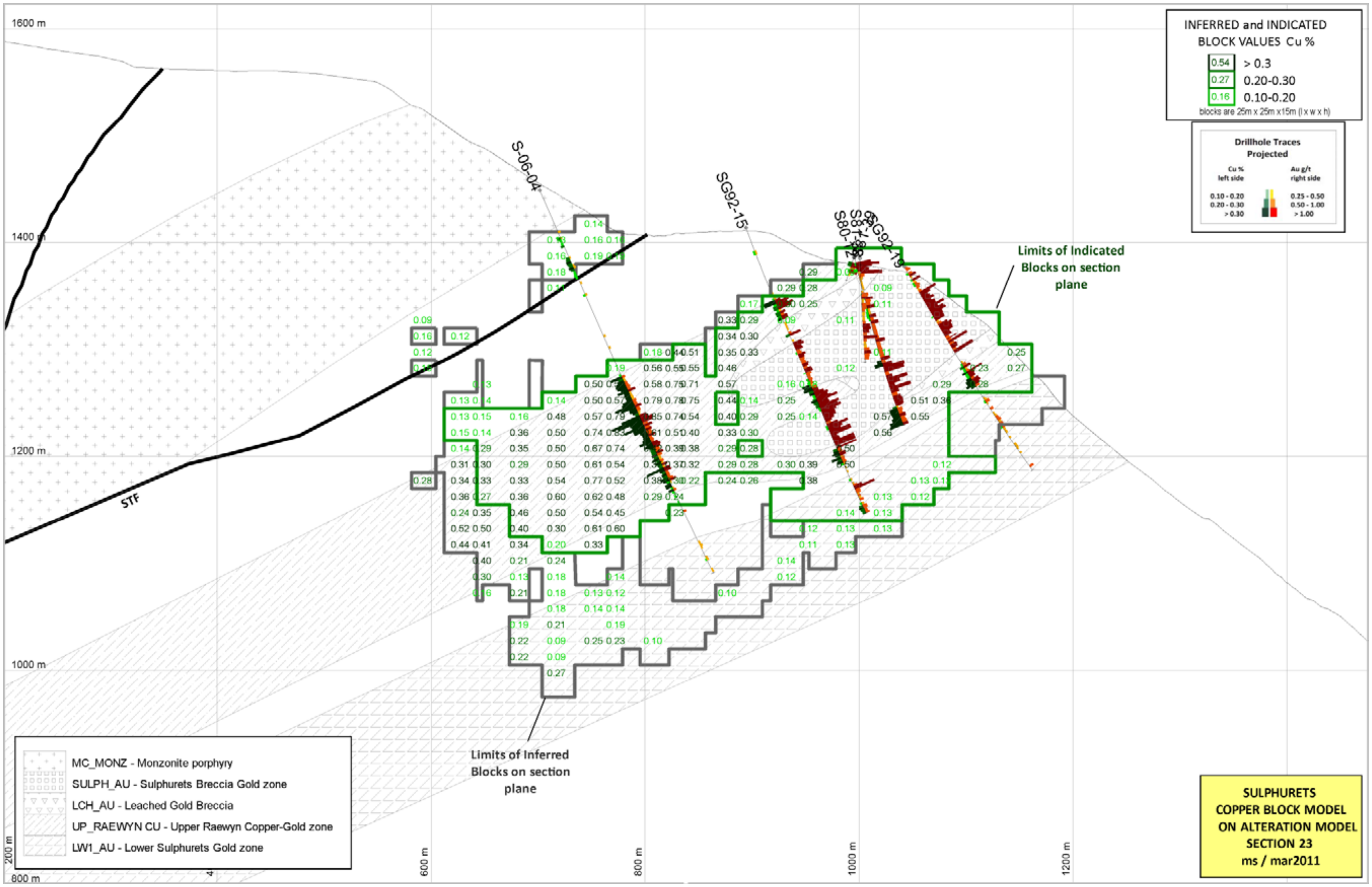


Figure 9-6: Sulphurets Geologic Section 23 (Cu)



9.3 Mitchell Zone

Eleven core holes totaling 3,186.11 meters were drilled by Seabridge in 2010 within the Mitchell. The majority of these holes were drilled along the northern and eastern flanks of the mineralized zone in order to upgrade Inferred Resources to Indicated Resources. The drill hole spacing at Mitchell is somewhat variable but within the core portion of the Mitchell zone drill hole spacing varies between 50 to 100 meters. The total drill pattern has tested a volume measuring roughly 2,000 meters by 1,200 meters by 600 meters. Geological and assay results were generally consistent with existing models for geology and metal distribution, but some revisions were made to the geometry of solid models to reflect new data.

The Mitchell zone is exposed in Mitchell valley through an erosional window exposing the footwall of the Mitchell Thrust Fault. The zone is a moderately dipping, roughly tabular gold-copper deposit measuring approximately 1,600 meters along strike, 400 to 900 meters down dip, and at least 300 to 600 meters thick. It consists of a foliated, schistose or mylonitic zone of intensely altered and sulfide bearing rocks, with a variably distributed stockwork of deformed and flattened quartz veinlets. The schistosity generally follows an east-southeast direction, and dips moderately steep to the north. In general, the core area of mineralization has a moderate plunge to the north or northwest, and is lineated in a east-southeast direction.

Recent glacial melt back has provided exceptional surface exposure of a relatively fresh gold-copper porphyry system. A zone of intense quartz and sulfide veining ("High Quartz") forms resistant bluffs in Mitchell valley. However, the higher grade core area is mostly covered by talus and moraine west of the bluffs. Active oxidation and leaching of sulfides has produced prominent gossans and extensive copper sulfate precipitates at the surface.

The Mitchell zone is considered to lie within the spectrum of the gold-enriched copper porphyry environment. Metals, chiefly gold and copper (in terms of economic value), are generally at low concentrations, finely disseminated, stockwork or sheeted veinlet controlled, and pervasively dispersed over dimensions of hundreds of meters. Grades diminish slowly over large distances; sub-economic grades are encountered at distances of several hundreds of meters beyond the interpreted centre of the system. This is distinct from the Sulphurets and Kerr zones, where there are more abrupt breaks in grade due to higher structural complexity and juxtaposition of weak and moderate grade domains by faulting, both syn-mineral structures controlling breccia contacts, and post-mineral faulting and displacements.

9.3.1 Lithology and Structure

Due to the intensity of hydrothermal alteration and strong post-mineral shearing, especially at the Mitchell zone, it is difficult to impossible to determine the original protolith. This is especially true in phyllic-argillic or quartz-sericite (illite)-pyrite altered rocks. In chlorite-sericite and propylitic altered rocks, a homogeneous, tuffaceous texture is often observed, and the host is believed to be intermediate volcanic tuffs or volcaniclastics. However, these textures may in part be shear related. Petrographic studies indicate the host was possibly a sequence of fine grained andesitic volcaniclastics, crystal tuffs, and porphyritic flows with coeval, fine dioritic dykes and sills throughout. Diffuse, ghost porphyritic textures may reflect dikes of the Mitchell intrusions. Rare, meter-scale, aphanitic intermediate dykes are post-alteration and unmineralized. Rarer monzonitic intrusives have been recognized as well.

Where not obliterated by alteration, fine to coarse, lithic to crystal, tuffaceous, intermediate volcanics are dominant, followed by vaguely bedded, fine grained volcaniclastics and argillites, more common to the west. Government mappers have assigned the stratigraphy under the Mitchell fault to the Jurassic Hazelton Group, however in many ways it more closely resembles descriptions of the Triassic Stuhini Group. Within the central and eastern portions of the drilled area, intervals of bleached, vaguely coarse porphyritic textured rocks may be altered dikes of the Mitchell intrusive suite.

Above the Mitchell thrust fault, alteration is mainly confined to siliceous hornfelsed zones adjacent to porphyritic monzonite and granitic Mitchell intrusions. The host rocks are mostly dark, fine grained volcaniclastics, argillites and vaguely porphyritic andesites and basaltic flows assigned to the Triassic Stuhini Group. The intrusions appear to have thick, sill-like geometries, with thin, anastomizing dykes in the contact zones. Similar intrusives and surrounding siliceous alteration zones have been mapped above the Mitchell Thrust Fault on both sides of Mitchell valley.

9.3.2 **Alteration and Mineralization**

Alteration and mineral zoning patterns have been modified by syn- and post-mineral deformation, however logging and petrographic examinations have been able to demonstrate the system generally follows established models observed at other gold-copper porphyry districts. The coding system utilized here was modified from codes used at Sulphurets by Ditson, et al.

- **Primary Hypogene Assemblages**

The dominant primary hypogene alteration mineral assemblage is propylitic, with quartz-chlorite-pyrite-chalcopyrite, often with magnetite and carbonate, and more rarely with anhydrite and molybdenite, and very rarely with bornite. It is characterized by pervasive chloritization of mafics, and quartz-pyrite alteration of most other silicates. This mineralogy is found in stockwork veins and the altered host rocks. Microscopic examination suggest much of the chlorite is replacing original hornblende, and to a lesser extent biotite. Occasionally there is textural evidence that suggest some of the replaced biotite may have been hydrothermal and related to earlier potassic alteration. Chalcopyrite precipitated after most of the pyrite. As quartz, pyrite, and chalcopyrite are generally ubiquitous, this alteration gets assigned to one of the following codes based on the following criteria:

- CL - chlorite dominant
- CL2 - chlorite with magnetite and carbonate, no epidote
- CL2STW - chlorite with magnetite and carbonate, no epidote, with >60% quartz veins
- CLSTW - chlorite alteration >60% quartz veins

There are occasional remnants of an earlier core potassic alteration. The geometry is uncertain, and it appears that a large portion of potassic alteration has been propylitized to some degree. It is found in stockwork veins, wall rocks, and early hydrothermal breccias. Some of the veins have a remnant wormy, pegmatitic texture. It is characterized by the presence of brownish-pink orthoclase and adularia typically in veins or vein haloes. Magnetite and very dark chlorite or biotite are usually present, rarely anhydrite. Quartz, pyrite, and chalcopyrite are ubiquitous. This type gets coded as KP.

Mainly peripheral to the CL and KP assemblages is a distal propylitic assemblage in andesitic and dioritic host rocks characterized by the presence of epidote, chlorite, calcite and ubiquitous quartz and pyrite. Veining and associated chalcopyrite and molybdenite are lower in abundance. This gets assigned a code of PR.

Where the host is a sedimentary rock, the distal propylitic assemblage is similar but the rock has a more hornfelsic texture, and if epidote is absent it gets assigned a code of HFLS. Banded and spotted ("diseased") hornfelsic textures are common especially in the footwall of the Mitchell zone. Often silica and pyrite are the only alteration minerals present, indicating the absence of a mafic component in the original sediment, and the alteration gets coded as SIH, or in the case of magnetite-rich hornfels, as MTH. Rare calcareous sediments have typical hornfelsic skarn assemblages.

- **Secondary Hypogene Assemblages**

The propylitic and potassic assemblages are overprinted by secondary phyllic assemblages. Towards the east and higher areas of the Mitchell zone, the overprint is intense and pervasive, but is variable and intermittent to the west and at depth. The phyllic assemblage is characterized by complete loss of mafics, introduction of mm to cm scale, deformed quartz veinlets in stockwork and sheeted arrays, with mostly creamy white to grey sericite and/or illite and pyrite as the interstitial vein component. The phyllic assemblages may reflect a type of high sulphidation, downward penetrating, structurally controlled overprint where fluids in the upper portion of the hydrothermal cell reacted with acidic meteoric water.

Vein relationships suggest multiple pulses of overprinting phyllic veins, together with contemporaneous development of propylitic alteration and veining in new fractures over the development of the hydrothermal cell.

There are clearly multiple stages of veining. Later veins have abundant coarse pyrite, often with molybdenite, and cm scale, near massive coarse pyrite veins are common. The phyllic alteration has a strong foliation best manifested in sericite rich intervals. Sheeted quartz veinlets often follow the foliation, and may indicate deformation of pre-existing veins, or perhaps contemporaneous formation of quartz veinlets and deformation. In some surface exposures, intensely deformed zones contain coarse clasts of rotated, previously veined material, and strong shear textures are noted in microscopic thin sections. The highest concentrations of pyrite and quartz veinlets are generally strongly coincident with phyllic-argillic alteration. These assemblages are coded as follows:

- QSP - creamy to grey fine sericite and/or clay, strongly schistose or mylonitic. Quartz vein stockwork usually intense, and are mostly foliation parallel or oblique. Generally 5to 20% fine disseminated pyrite, lesser chalcopyrite, minor molybdenite, rare tourmaline.
- QSPSTW - similar to above with >60% quartz veinlets, in general it is forms the core area of the QSP, but appears to be fragmented or dismembered
- IARG - intermediate argillic characterized by pale green sericite and/or chlorite (which in large part may be illite or other clays), abundant pyrite, common molybdenite, late pyrite only veins. In general, it forms a crude partial halo around the east, north, and south sides of the Mitchell zone. It has lower than average copper and gold concentrations, and higher than average molybdenum.

- **Other Secondary Late Veins and Minerals**

Coarse, centimeter scale, purple tinted anhydrite veins occasionally are found throughout the Mitchell zone, more typically at depth and along the north side or hanging wall of the deposit. These are distinct from the sub-millimeter anhydrite filled fractures that are found in isolated parts of the propylitic altered areas.

Relatively coarse grained, sub-centimeter pyrite veins are common especially in the upper portions of the Mitchell deposit. These tend to have a distinctly paler tone than earlier pyrite.

A variety of micron-scale silver sulfosalt occurrences have been identified in microscopic examination of polished thin sections. These are usually found along the north side of Mitchell. Although the core of the Mitchell zone contains elevated silver values on the range of 3 to 6 ppm, based on observations to date the silver here probably occurs as a contaminant within chalcopyrite, not as sulfosalts.

Trace amounts of galena, sphalerite, arsenopyrite, and tetrahedrite or semseyite have been observed, mostly occurring in secondary alteration phases.

Centimeter to sub-meter scale, discontinuous, bulbous, boudinaged, coarse, pegmatitic, quartz-chlorite-calcite veins are common throughout the Mitchell zone. These are almost always mineralized with chalcopyrite that is typically coarser grained than in the host rocks. The calcite is often tinted orange when exposed and is probably ankeritic. These veins appear to have been emplaced in dilatent zones at the last stages of the regional deformation event.

- **Bornite Breccia**

The Bornite Breccia is a late, cross-cutting pipe or dilatent structure within the Mitchell zone. In this structure, bornite replaces earlier aggregates of pyrite and chalcopyrite which occur in the matrix of a silica and anhydrite rich mass. The texture is chaotic and deformed, and is tentatively interpreted as a breccia vein subsequently sheared during regional deformation. It is postulated to have formed from acidic fluids related to the high sulphidation overprint descending along fractures and precipitating in cross-cutting or dilatent structures. Gold grades are lower than average, and there is a halo surrounding the structure from which gold and copper have been leached. The structure has dimensions of approximately 300 by 300 meters with a maximum thickness of about 50 meters, and dips steeply to the north cross-cutting the general shape of the Mitchell zone. The leached halo is about 10 to 30 meters wide. The interpretation is tentative as it has only been intersected in a few drill holes, and has not been observed in outcrop.

- **Hypogene Leaching**

Petrographic examination of polished thin sections from the Mitchell zone indicates that chalcopyrite accompanied all of the hypogene alteration assemblages to some degree. Also, there are indications that the secondary phyllic alteration leached some metal, including copper and gold, from earlier phases, and redistributed that metal in new veinlets. As the system matured, re-fracturing and multi-phase primary and secondary alteration episodes would, through leaching and re-precipitation, have the effect of homogenizing metal distribution especially considering the density and homogeneity of the fracture (vein) patterns over much of the Mitchell zone.

- **Gold**

Gold has not been observed at Mitchell except under microscopic examination of polished thin sections and metallurgical test concentrates. When observed, gold grains are generally less than 10 microns, and occur within both pyrite and chalcopyrite grains, on sulfide grain surfaces, and as grains isolated in minute fractures in gangue. Preliminary metallurgical testing indicates about 60% of the gold is recoverable and would report to a chalcopyrite concentrate using standard flotation methods. Cyanide leaching of a pyrite concentrate to produce doré bars could bring total gold recovery to about 78%.

- **Supergene Processes**

Supergene processes of oxidation, leaching, and re-precipitation are essentially absent at the Mitchell zone, due to the high rate of erosion and glaciation. Along the higher areas of the slopes of Mitchell valley, oxidation has penetrated to several 10's of meters along a few fractures and copper oxide coatings have been observed in areas of the mineralized material above the Mitchell Thrust Fault. Below the fault, oxidation is rare and has only been observed in fractures within a few meters of the surface in the most southerly holes. Minor chalcocite coatings on chalcopyrite and pyrite have also been observed in these holes, to a maximum depth of a few meters.

- **General Observations**

At the property scale, gold and copper are generally coincident. An area marking consistent grades mostly above 0.75 g/t Au and 0.2% Cu has dimensions of about 500 x 1000m at the surface. To the east, the gold grade tends to fall off at a lower rate than copper. Gold and copper grades are closely related to the density of quartz stockwork veining, as sulfides are disseminated in minute “crackle” fractures within the veins, as well as coalescing vein haloes in wall rock. Zones of intense to massive quartz stockwork and sheeted veining where veins make up more than half of the rock volume (High Quartz) are contained within the central area of the deposit, but there is no consistent correlation between vein density and gold and copper grades. The “High Quartz” zones occur mostly within areas of intense phyllic or QSP alteration, but extend to the west and at deeper levels into propylitic altered areas. Molybdenum occurs in distinct halo that is stronger on east side. Analyses of molybdenite concentrates from metallurgical test sampling indicate anomalously high Rhenium concentrations.

9.3.3 **Structure and Metamorphism**

Regional mapping by government geologists demonstrate that Jurassic Hazelton Group rocks exhibit overturned folds that are southeast vergent in the region of KSM. The thrusts are also southeast vergent. The area occurs within the regional Skeena fold and thrust belt that was formed in the Cretaceous. Triassic Stuhini Group rocks above the faults form the east side of a broad north plunging anticlinorium. Triassic rocks were thrust over Jurassic rocks and truncated the upper portion of the Mitchell deposit. Less competent phyllic-argillic altered rocks at Mitchell and Kerr zones appear to have provided the least resistance and were the focus of shearing and faulting during this event.

High temperature and pressure conditions during post-mineralization deformation is thought to have promoted re-mobilization of metals and contributed to the homogeneity of grades over large distances.

Petrographic examinations of polished thin sections of selected core samples show ample evidence of post-mineral deformation, including ribbon textured quartz due to shearing, crushed and sutured quartz due to strain, shear fabrics, muscovite wrapping quartz boudins and quartz-sericite crystals filling pressure shadows. Most of the core samples from within the Mitchell zone have been categorized as mylonites or mylonitic. Primary textures are rare.

Figure 9-7 is a geologic plan map of the Mitchell zone showing lithology, alteration, and structure. Assay grades are shown as histograms along the drill hole traces (copper shown in green and gold grades shown in orange). Figure 9-8 is a northeast-southwest trending cross section through the west central portion of the Mitchell zone showing drill hole traces/grades, lithology, alteration, and gold mineralization. Figure 9-9 is a similar cross section through the Mitchell deposit showing copper mineralization. Both cross sections also show Indicated Resource block model grades (inside of the heavy green line). The line of section for the two cross sections is shown on the plan map (Figure 9-7) in reddish-brown.

Figure 9-7: Mitchell Geologic Plan Map

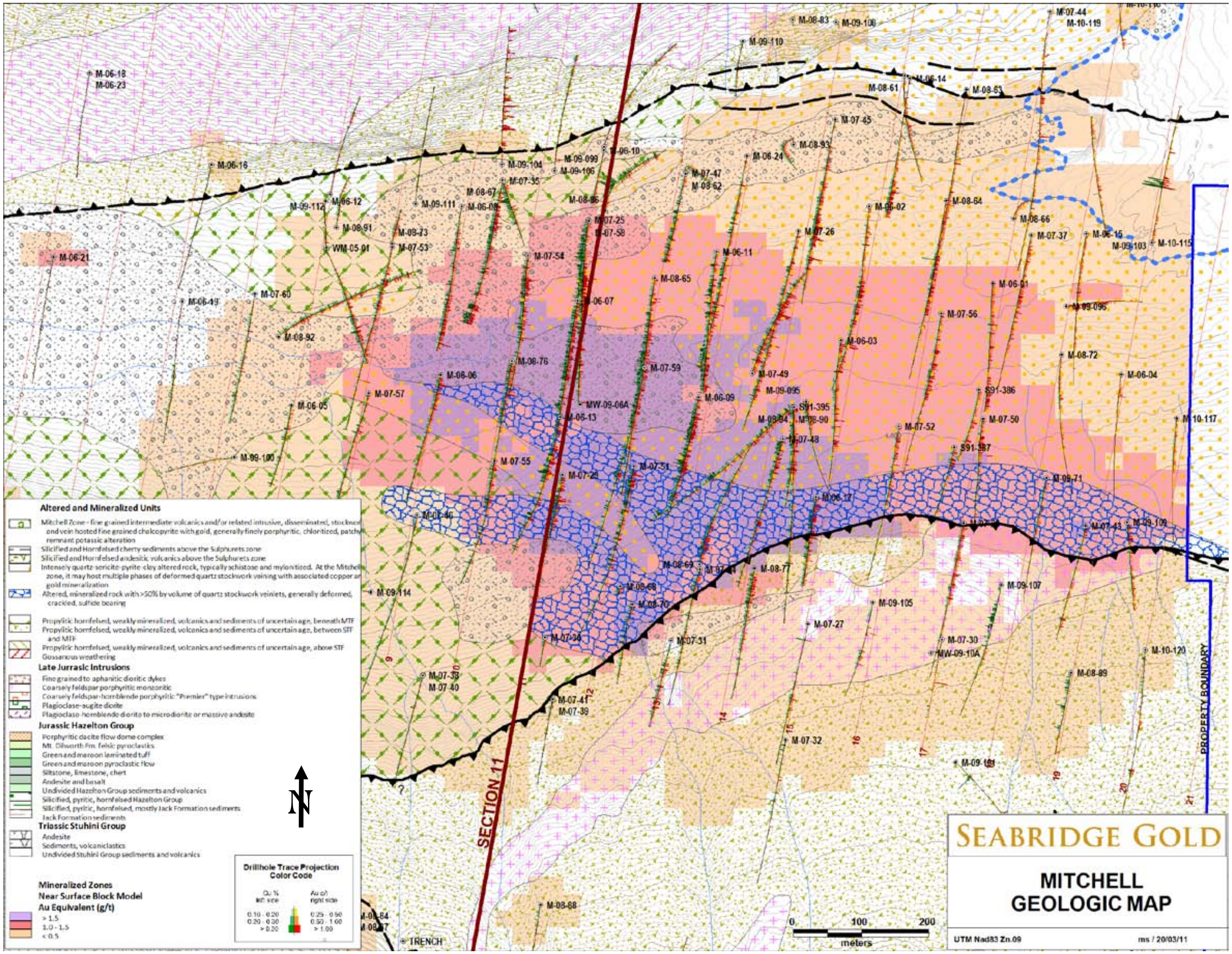


Figure 9-8: Mitchell Geologic Section 11 (Au)

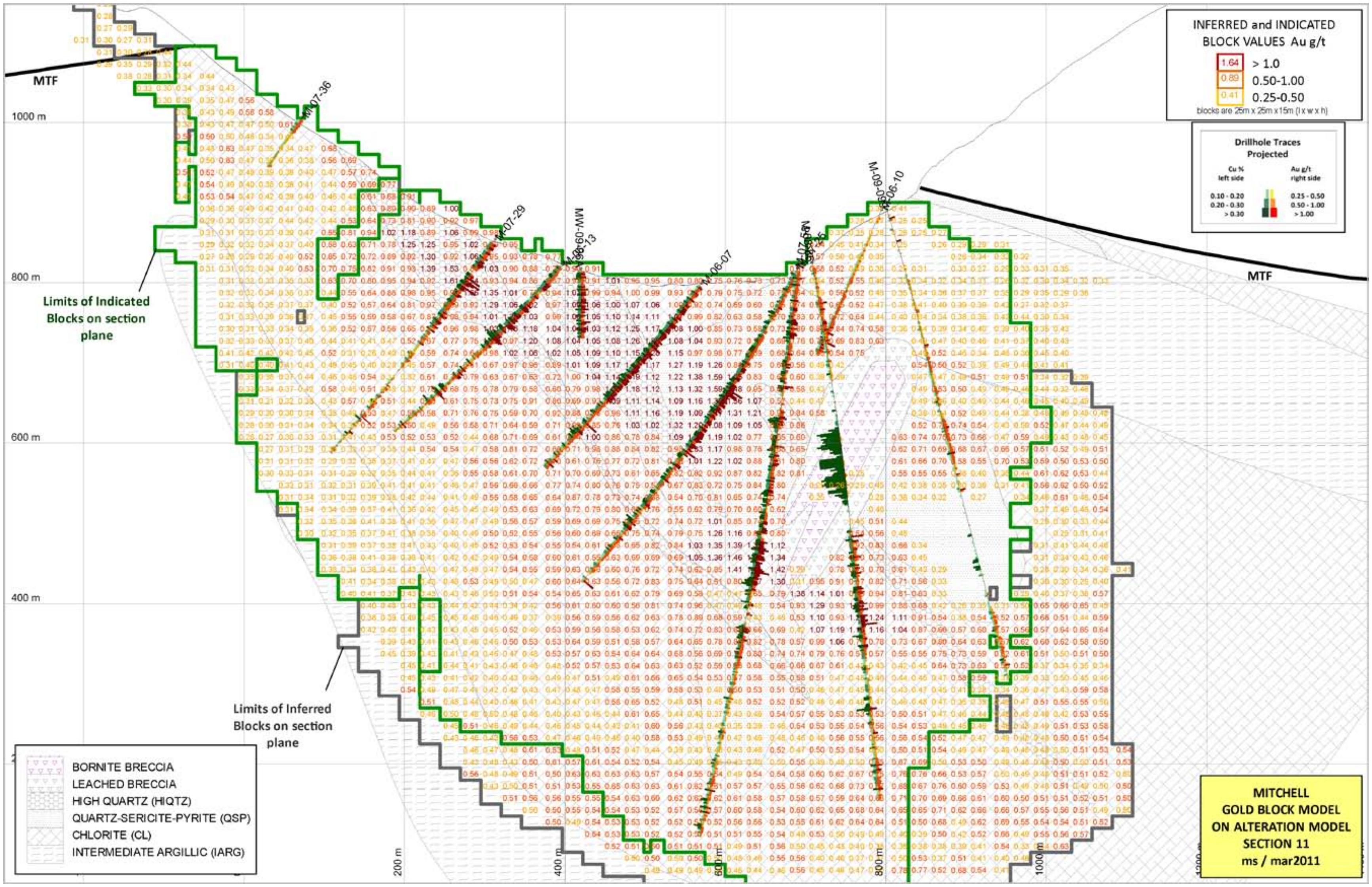
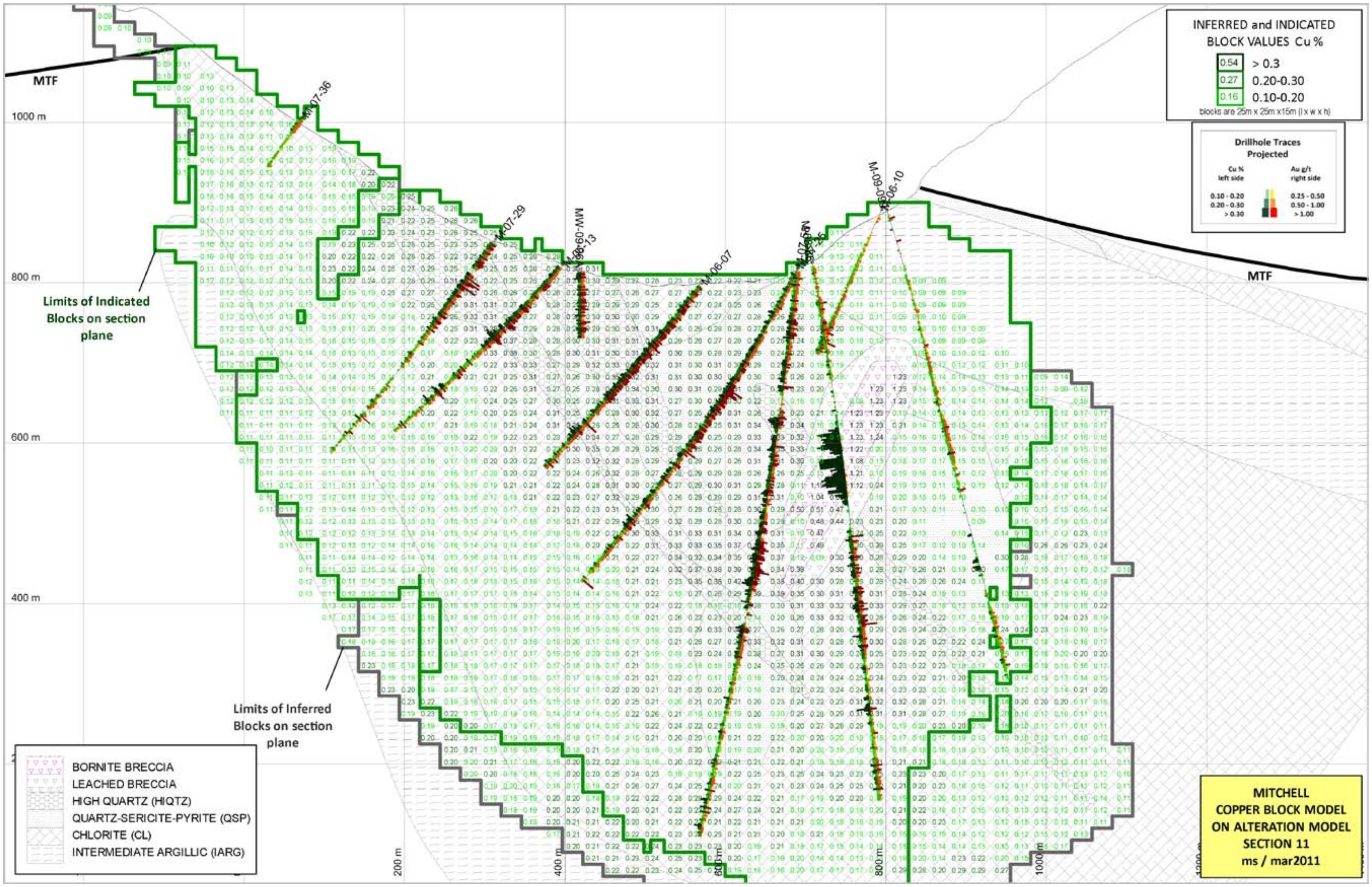


Figure 9-9: Mitchell Geologic Section 11 (Cu)



- **Ore Type Categories**

Ongoing engineering and metallurgical test work at KSM requires that the mineralized zones be categorized for sampling on the basis of hardness, work indices, density and mineralogy. As the hydrothermal alteration mineral assemblages and post-mineral deformation effects contribute the most in terms of identifying contrasts between rock properties and mineralogy styles, alteration coding and solid modeling was used to differentiate the “end members”. In relatively unaltered rocks, the lithology determines the contrasts.

Table 9-3 summarizes and briefly describes the alteration and lithologic characteristics associated with the various ore types for the Mitchell zone.

Table 9-3: Mitchell Ore Types

Ore Type	Description/Source
QSP	Quartz-sericite-pyrite alteration - QSP wireframe model
IARG	Intermediate argillic alteration - carved from sericite and CL-PR models where IARG is logged and Cu < 0.11% and Mo > 55 ppm
CL-PR	Chlorite-propyltic alteration - CL-PR wireframe model
High-quartz	+60% quartz wireframe model
Hornfels	Altered seds and volcanics - All other rocks except monzonite and bornite models
Monzonite	Monzonite wireframes (upper plate)
Bornite breccia	Bornite breccia wireframe (Cu > 0.50%)
Uncategorized	Undrilled areas

The CL-PR (chloritic-propylitic) ore type contains both propylitic (PR) and CL-PR (chlorite-propylitic) alteration types, which have a similar mineral assemblage (quartz-chlorite-pyrite). The PR alteration (usually characterized by weaker veining and the appearance of epidote and higher calcite) is generally peripheral and indicates direction away from main zone of mineralization. The transition from CL-PR to PR is gradual and may be imperceptible in a drill hole at an oblique angle to the transition. Thus the logged position of the contact is approximate and difficult to align from hole to hole and section to section.

Similarly, the transition from the QSP to CL-PR alteration is typically gradual. The revised alteration model has the QSP / CL-PR contact repositioned a bit further to the east (an indication of how subtle the transition is). In general, the QSP alteration model at deeper levels contains some meter-scale CL-PR intervals, where as the QSP alteration at shallow depths (especially to the east) contains rare intervals of CL-PR alteration, and is more strongly mylonitized (deformed and schistose).

9.4 **Iron Cap Zone**

The Iron Cap Zone is a roughly 600 by 1,500 meter area of well-exposed, intensely and pervasive quartz-sericite-pyrite altered intrusive, sedimentary and volcanic rock approximately 2,300 meters northeast of the Mitchell zone. Quartz-sulfide-gold bearing veins within Iron Cap zone attracted previous explorers and were the focus of blast trenching and three short drill holes drilled by Esso Minerals in 1980 which intersected wide intervals of low grade copper-gold mineralization. The Iron Cap deposit has been delineated by over 17,790 meters of core drilling in 52 drill holes spaced at intervals of 50 to 100 meters; in total three different operators drilled the project between 1980 and 2010. In 2010, Seabridge drilled 15,400.6 meters in forty-one core holes to explore, delineate and model the deposit. This resulted in the first resource estimation of the Iron Cap deposit, which contains substantial volumes of both inferred and indicated resources.

9.4.1 **Lithology, Alteration and Structure**

The Iron Cap deposit is a separate but related mineralized system within the KSM district, and occurs structurally above the Mitchell deposit, in the panel of rocks between the Mitchell and Sulphurets thrust faults. It differs from the Mitchell deposit primarily in that much of the host rock is hydrothermally altered intrusive (porphyritic monzonite to diorite) rather than volcanics and sediments. The volcanics are mostly andesitic porphyry, generally similar to the main host of the Mitchell deposit. There is a high degree of silicification which overprints earlier potassic and chloritic alteration. Intense phyllic alteration and high density stockwork veining, which are pervasive at Mitchell are less pervasive at Iron Cap. Copper bearing zones at Iron Cap demonstrate higher grades than Mitchell, which is consistent with the intrusive setting and potassic alteration indicating a deeper and hotter environment.

Associated with the silicification are wide zones of hydrothermal brecciation, scattered meter-scale quartz-pyrite-chalcopyrite veins and centimeter-scale quartz-pyrite-chalcopyrite-sphalerite-galena-tetrahedrite veins that are interpreted to be superimposed on earlier stockwork and disseminated mineralization associated with the intrusion. Microscopic examinations of polished thin sections confirm that Iron Cap was also subjected to a post-mineral deformational event evidenced by widespread mylonitic textures. “Mylonite” and Ultramylonite” are terms used as rock names in petrographic descriptions of several Iron Cap mineralized samples.

Generally intense silicification at the higher, eastern portions gives way to chloritization with some preserved k-spar alteration at depth and towards the west which correlates with increasing proportion of intrusive rock. Relative to Mitchell, stockwork veining is much weaker. There is a distinct overprint of structurally controlled, centimeter scale quartz-carbonate veins with chalcopyrite, galena, sphalerite, and tetrahedrite, but the distribution is not clear. It does not seem to effect the gold and copper distribution on a large scale, but at the vein scale there is often correlation. High silver values are generally associated with presence of galena and sphalerite.

Petrographic study of selected core samples suggests the following possible sequence of events and alteration mineral assemblages at Iron Cap:

- Porphyry gold-copper style hydrothermal alteration with chalcopyrite-gold mineralization including silicification, phyllic alteration (sericite-silica-pyrite), propylitic alteration (albite-carbonates-chlorite-pyrite), and weak potassic alteration (secondary kspar)

- Contact metamorphism or hornfelsing (magnetite-diopside-clinozoisite)

- Base metal sulfides (chalcopyrite-galena-sphalerite-tetrahedrite-pyrite overgrowths and silver-bearing sulfides (freibergite-acanthite-proustite)

- Medium temperature, low pressure metamorphism (cordierite-pyrophyllite)

- Regional metamorphism, high pressure (kyanite-rutile-chalcedony) with pressure shadows, strained quartz, contorted and offset veins, fractures, carbonates and sulfates

- Precious metals (native silver and gold, quartz and adularia)

- Weathering (Kaolinite-iron oxides)

Chalcopyrite, silver-bearing galena, freibergite, and tetrahedrite are found in tiny vugs of vuggy pyrite in early pyrite. Later pyrite overgrowths are not vuggy suggesting a leaching event occurred between precipitation of the first and second pyrite generations.

Secondary magnetite is intergrown with pyrite and chalcopyrite, suggesting that the formation of these minerals overlapped that of iron metasomatism.

Later regional metamorphism resulted in the formation of mylonite, which has pressure shadows on pyrite and other hard grains. Pressure shadows are most commonly filled by chalcedony and pyrophyllite. Pyrite that has pressure shadows may be the same generation of pyrite that formed overgrowths on earlier pyrite. Many protoliths have been so altered by dynamic metamorphism that the original rocks are not recognizable. The presence of pyrophyllite in mylonite indicates relatively high temperatures (300-400 oC) of formation. Pressure shadows (or former voids) by pyrite grains that are now filled by various secondary minerals indicate directions of extensional stress during metamorphism.

Precious metals and sulfosalts precipitated after base metal sulfides and after a leaching and fracturing event, because they occur in vugs and fractures. Late quartz and adularia may have been associated with native gold and silver.

Given the relatively complex hydrothermal and metamorphic history of the Iron Cap zone, it is not unexpected that the distribution of hydrothermally deposited metals shows little correlation with lithology, alteration, or veining. The clearly defined metal zonation present at Mitchell is likely a function of its deeper position in the porphyry profile, (i.e. a stable, mature hydrothermal event with low thermal gradient) and host rock consistency. At Iron Cap, and to some degree at Sulphurets, the positioning of the system is higher in the porphyry profile (and topographically at present), where thermal gradients are interpreted to be more abrupt due to stronger lithological and structural contrasts and more complex fluid pathways. Shallower hydrothermal systems are also subject to a higher degree of mixing with meteoric waters, phreatic events, hydro-fracturing, and brecciation. This is the environment of high sulfidation events, characteristics of which portions of Iron Cap display, including cockscomb textured veins, base metals, acid leaching, high silicification, and late stage gold and silver precipitation.

9.4.2 **Three Dimensional Geologic Models**

In 2010, geological models were interpreted by Seabridge geologists on sections spaced at approximately 100 meter intervals, and assembled to construct three-dimensional wireframes for grade estimation. Mineralized domains were built independently of lithology and alteration models as no clear correlation exists. The model incorporates data from Seabridge's core logging of drill holes from 2010, and re-logging of historical holes from 2005.

Brief descriptions of revised geological units using wireframe coding follow:

Lithology

STUHVOLC – Unmineralized, unaltered Stuhini group bedded sediments and volcaniclastics above the Sulphurets thrust fault.

PMON – Porphyritic Monzonite, generally barren to low grade. This is considered post-mineral relative to Iron Cap and Mitchell. A distinct mineralized contact zone has also been modeled (PMONAUCU).

IC_DIOR – Fine grained intermediate intrusive and/or related flows, generally finely porphyritic, chloritized, probably closely related to similar intrusive observed in deeper portions of Mitchell zone. Usually mineralized with disseminated chalcopyrite and associated gold.

IC_QZVNS – A few steep, narrow massive to semi-massive quartz veins that correlate to mapped veins on the surface were intersected in drilling. They tend to pinch and disappear at depth. These are late in the mineralizing sequence at Iron Cap.

IC_UNCAT – These comprise undifferentiated, intensely altered sediments, volcanics, and possibly intrusive rock and hydrothermal breccias in the structural panel between the Sulphurets thrust fault and Iron Cap fault. Intense alteration renders them indistinguishable and results in inconsistent lithological assignments by different loggers.

FW_UNCAT – This comprises undifferentiated, mostly hornfelsed sediments, volcanics, and monzonite dykes considered equivalent to the North Mitchell panel geology. There is not sufficient drilling to resolve the geology. A distinct zone of higher Mo mineralization has been identified in these rocks, and wireframed as a mineralized domain (IC_MO_ZN). Beneath the Mo zone a distinct interval of near massive, hydrothermal magnetite skarn or breccia with coarse clots of pyrite and chalcopyrite was intersected in hole IC-10-044 from 389 to 402 (end-of-hole) which assayed 3.00g/t Au and 0.37% Cu. This is the only intercept of this material.

Alteration

CL – Chlorite alteration dominant, with variable silicification and phyllic overprints, traces of remnant potassic alteration, chalcopyrite with gold almost always present. Generally correlates with intrusive and volcanic host rock

CL_SIL – Mixed, transitional assemblage between CL and SIL

SIL – Silica-pyrite alteration dominant, sericite and remnant chlorite common, usually some chalcopyrite and associated gold, with scattered, late centimeter scale quartz-base metal veins more common than other units

KP – Closely similar to CL, but with minor remnant potassic alteration, usually as preserved, pink-brownish K-feldspar veinlets and “bleeding” around veinlets. Stockwork quartz-sulfide veining tends to be stronger, but still weak compared to the Mitchell zone.

KP_PMON – Potassic alteration associated with the late PMON monzonitic porphyry.

SIH – Siliceous and propylitic hornfels alteration, with disseminated pyrite, minor chalcopyrite, with later mm to cm scale carbonate veinlets. Generally effects all other rocks in panel between Sulphurets and Iron Cap faults

FW_SIH – Siliceous and propylitic alteration beneath Iron Cap fault similar to SIH

Mineralization

LOW_AUCU – Lower zone Iron Cap mineralization, disseminated, stockwork, and vein hosted fine grained chalcopyrite, variable but on generally >0.2% Cu and 0.5g/t Au, but the boundaries are not always well defined. This is hosted both by altered volcanics and intrusive rocks, and comprises the bulk of the Iron Cap zone. There is no clear grade boundary between volcanic and intrusive, even in the rare case where the lithological distinction is clear.

MID_AUCU – Low grade Iron Cap mineralization, sandwiched between UP_AUCU and LOW_AUCU.

LOW_AUCU. Boundaries are usually well defined. Cu and Au values are quite variable, but on average are lower than Upper and Lower zones. There are some sharp grade boundaries at faults within this domain but there is insufficient resolution to satisfactorily map them out. The Au to Cu ratio increases northeastward until there is very little Cu, but the transition is gradual, and probably reflects increasing dominance of silicified volcanic-sedimentary rock versus intrusive rock.

UP_AUCU – Upper zone Iron Cap mineralization, disseminated and stockwork hosted fine grained chalcopyrite, generally >0.1% Cu with variable Au. Intruded by the barren PMON and PMONAUCU mineralized contact zone. Pinches out moving east.

PMONAUCU – Contact zone of Porphyritic Monzonite, with dykes, intrusive breccias, and altered wall rocks, generally moderately mineralized with both Au and Cu from the previous Iron Cap main mineralizing event.

IC_MO_ZN – A distinct zone of relatively high molybdenum, average Cu, low Au, and high Ag mineralization beneath the Iron Cap fault, in rocks of the North Mitchell panel. Not much drilling here, but continuity is clear. May be more closely associated with higher molybdenum mineralization at the periphery of the Mitchell zone than with the Iron Cap zone, although it is structurally above the Mitchell thrust fault.

<u>Structure</u>

IC FAULT – Iron Cap fault, an east-southeast trending, steep north dipping fault, revised from previous version to accommodate two drill hole intercepts. Iron Cap zone proper is situated above this fault, and the Iron Cap moly zone (IC_MO_ZN) lies below it. underneath.

There are several faults and sheared structures logged in drill core and mapped on the surface between these two major faults, but there does not appear to be any major offsets, and have not been modeled due to poor resolution. Most are probably steep, north-northeast trending, minor normal faults, several of which are clearly observed as small lineaments in surface exposures.

Oxide Facies

IC_OXIDE – A clearly defined blanket of weakly oxidized rock, roughly parallel with topography, ranging from a few meters to several tens of meters thick. Characterized by weak oxidation of sulfides, typically to jarosite, hematite or goethite, usually along fracture faces, especially in brittle, siliceous rocks. Rare coatings of chalcocite and covellite have been observed, but there is no obvious leaching or enrichment within this zone.

Figure 9-10 is a geologic plan map of the Iron Cap zone showing lithology, alteration, and structure. Assay grades are shown as histograms along the drill hole traces (copper shown in green and gold grades shown in orange). Figure 9-11 is a northwest-southeast trending cross section through the Iron Cap zone showing drill hole traces/grades, lithology, alteration, and gold mineralization. Figure 9-12 is a similar cross section through the Iron Cap deposit showing copper mineralization. Both cross sections also show Indicated Resource block model grades (inside of the heavy green line). The line of section for the two cross sections is shown on the plan map (Figure 9-10) in reddish-brown.

Figure 9-10: Iron Cap Geologic Plan Map

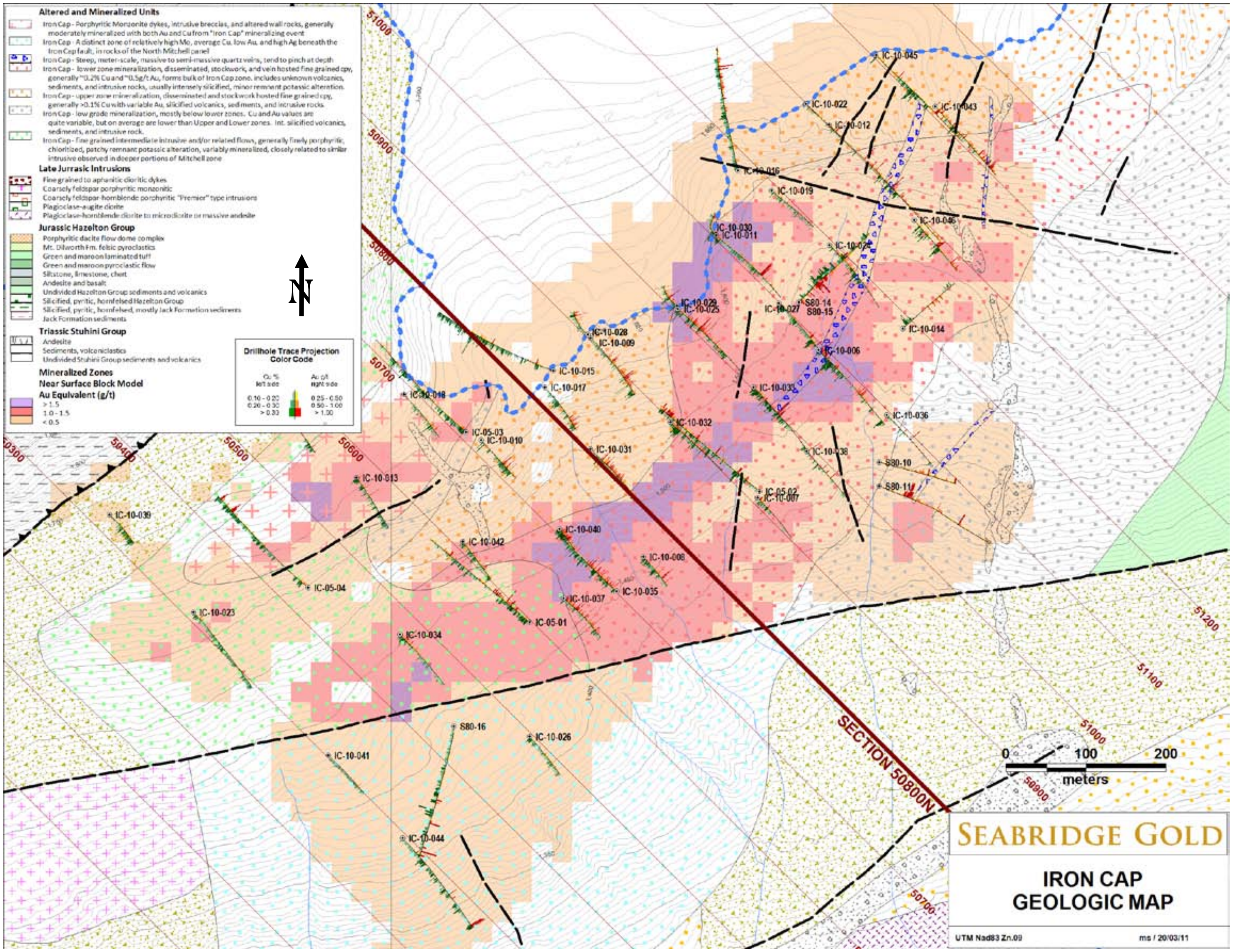


Figure 9-11: Iron Cap Cross Section 50800 (Au)

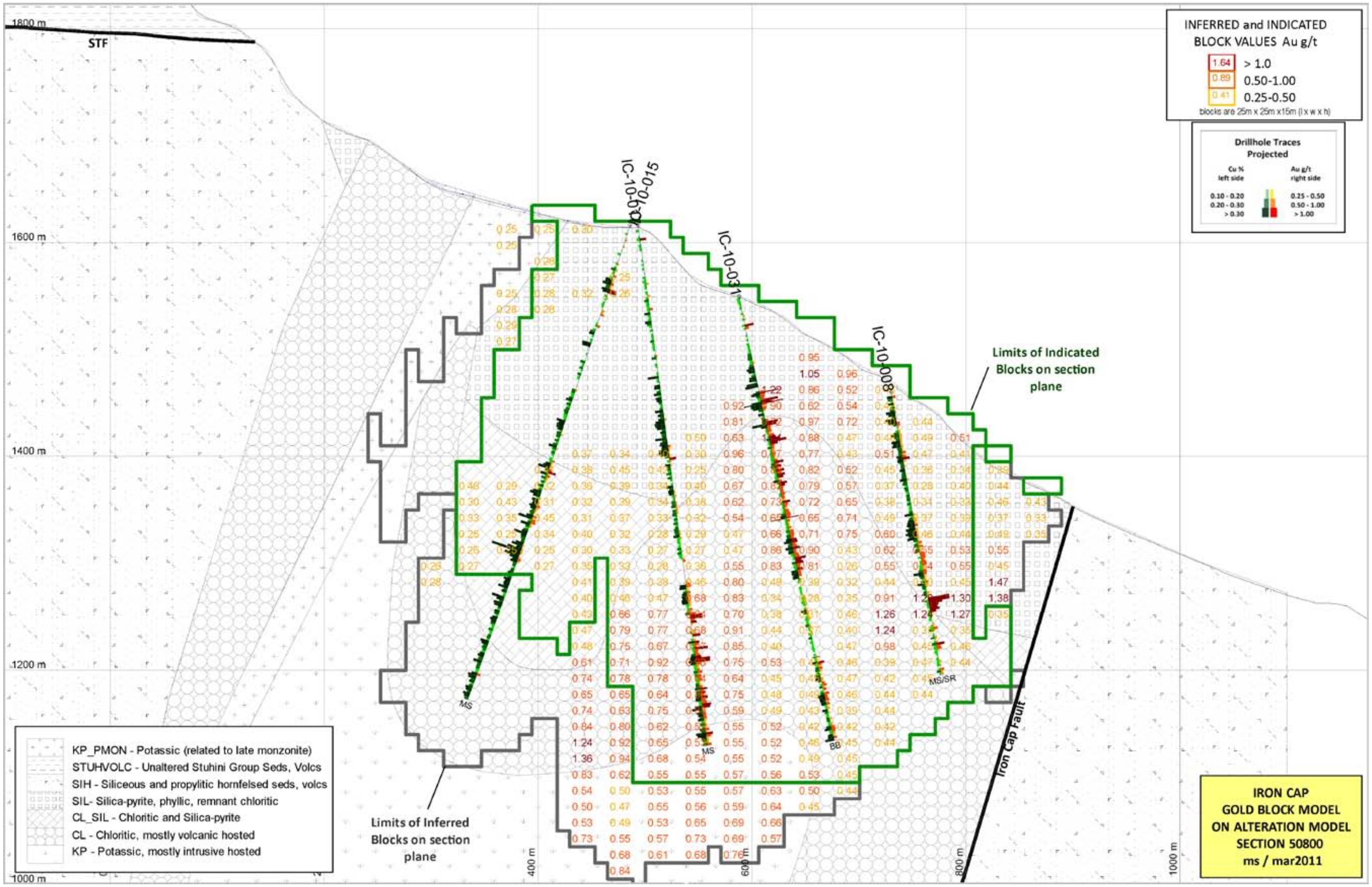
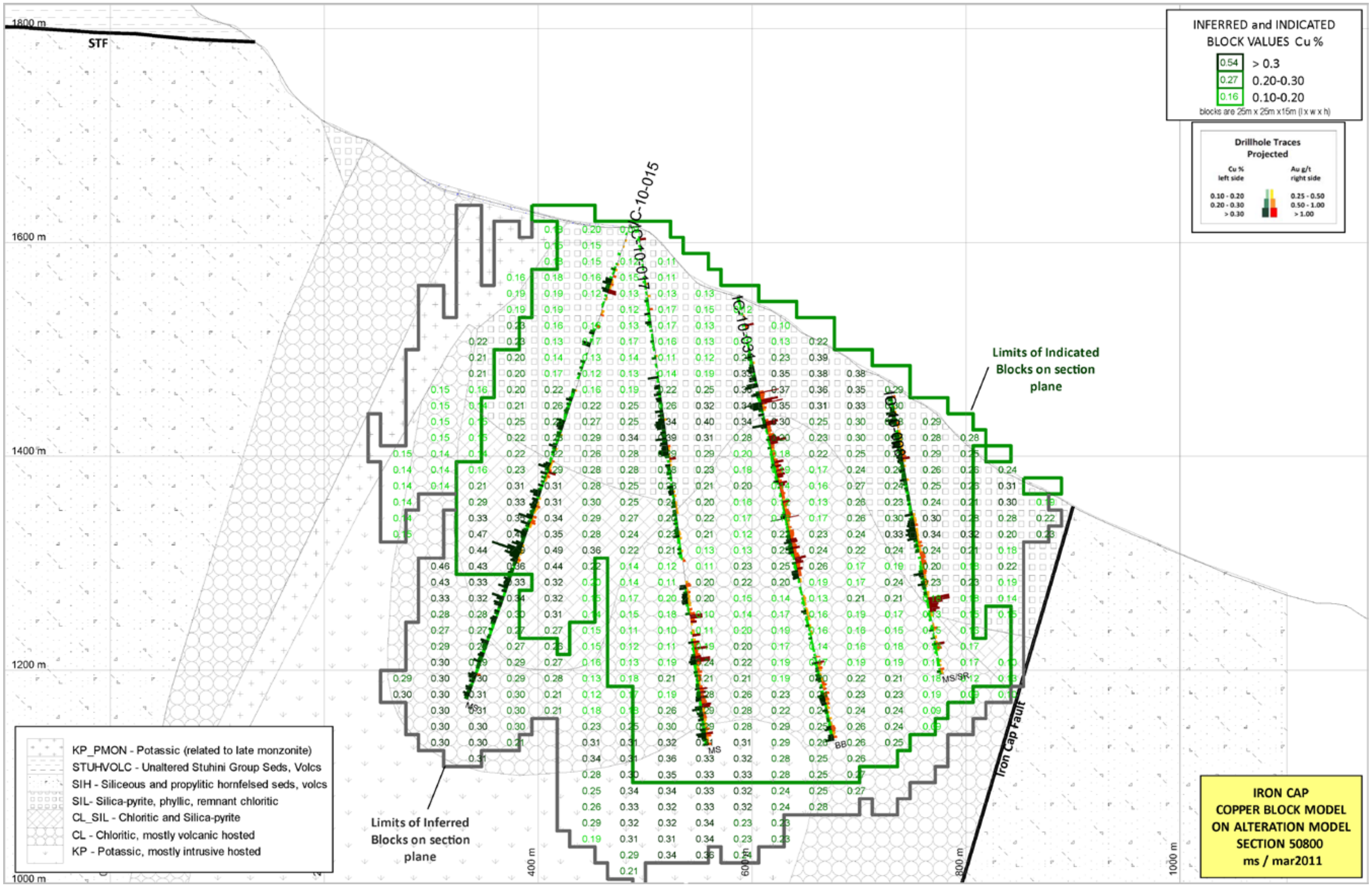


Figure 9-12: Iron Cap Cross Section 50800 (Cu)



10.0 EXPLORATION

This section describes Seabridge's 2010 exploration program at KSM. Prior exploration activities have been described in various Technical Reports prepared by RMI (Lechner 2007, Lechner 2008a, Lechner 2008b, and Lechner 2009).

10.1 2010 KSM Exploration Program

Seabridge's 2010 exploration efforts were directed towards:

- Infill drilling within the Mitchell and Sulphurets deposits in order to upgrade resource categories within current pit designs to at least an indicated level.

- Exploration and delineation drilling at the Iron Cap zone to model the deposit and facilitate a resource estimation.

- Geotechnical core drilling to provide data for engineering studies at Kerr, Sulphurets, Mitchell, and Iron Cap. Some of these holes intersected the mineralized zones and contributed to the resource database.

- Geotechnical overburden and core drilling in areas of proposed infrastructure well beyond the mineralized zones. This work is documented in a report prepared by Klohn Crippen Berger Ltd.

The 2010 Kerr, Sulphurets, Mitchell, and Iron Cap drilling programs are tabulated in Table 11-1. Individual mineral zone drilling statistics are summarized in Tables 11-3 through 11-6. Sixteen holes within the four deposits totaling 5,020.7 were drilled using triple tube tools that facilitated detailed geotechnical data collection and testing including down-hole digital photography. In addition, 17 shallow, large diameter holes were drilled at various locations throughout the property to provide data for engineering studies of potential mine support infrastructure.

The drill core was logged on site by Seabridge geologists who collected a variety of information including lithology, alteration, mineralization, and geotechnical attributes like core recovery, RQD, and fracture frequency. After photographing the core, it was sawn in half with primarily 2-meter-long samples collected and sent to Eco Tech Laboratories, Stewart Group, a commercial laboratory located in Kamloops B. C. Seabridge has used Eco Tech for their prior drilling campaigns. Recently Eco Tech was purchased by the Stewart Group. RMI will refer to Eco Tech Laboratories, Stewart Group as "Eco Tech" throughout the remainder of this report. The samples were analyzed for gold, copper and a suite of other elements. Additional bulk density determinations were completed by Seabridge geologists from all rock types and alteration assemblages.

Geotechnical data collection and studies were contracted to BGC Engineering Inc., Klohn Crippen Berger Ltd., and Rescan Environmental Services Ltd., all based in Vancouver, B.C.

10.2 **Results of 2010 Exploration Program**

The previous geologic interpretations of the Kerr, Sulphurets, and Mitchell zones were updated using the 2010 core hole data. The author notes the updated geologic interpretation remains virtually unchanged from the previous interpretation (see Section 7, 8, and 9). Approximately 58% of the 2010 drilling program focused on exploration and delineation of Mineral Resources for the Iron Cap zone. The assay results and subsequent geologic interpretation of the new Iron Cap drilling allowed for an estimate of Mineral Resources to be completed.

The drilling, sampling, and assay procedures employed for the 2010 exploration program were adopted from previous years and are discussed in Sections 11 and 12, respectively.

10.3 **Interpretation of Exploration Data**

The author combined the 2010 drill hole information with the previously collected data so that an updated geologic model and estimate of Mineral Resources could be made. The steps involved and results from those activities are discussed in Section 17.

10.4 **Statement Regarding Nature of Investigations**

All of the exploration activities that were conducted at KSM in 2010 were either directly carried out by Seabridge's geologic staff or directly supervised by Seabridge personnel.

11.0 DRILLING

This section describes Seabridge's 2010 drilling program at KSM. Previous drilling programs have been described in various NI 43-101 Technical Reports prepared by the author for the Kerr, Sulphurets, and Mitchell deposits (Lechner 2007, Lechner 2008a, Lechner 2008b, Lechner 2009, and Lechner 2010).

11.1 2010 Drilling Campaign

Seabridge Gold completed a helicopter supported diamond drilling program at the KSM project in 2010 as previously summarized. Hy-Tech Drilling Ltd. from Smithers, B.C. drilled all of the resource core holes using a Tech¬5000 Fly Rig using NQ and HQ tools.

Helicopter support was provided by two Eurocopter A-Star model 350B2 that were contracted from Lakelse Air Ltd. of Terrace B.C. The drilling operations were conducted from the Sulphurets Creek camp which is located southwest of the Mitchell deposit.

Approximately 13,931 diamond core samples were collected from the 2010 Kerr, Sulphurets, Mitchell, and Iron Cap drilling program and analyzed by Eco Tech Laboratories, Stewart Group out of Kamloops, B.C. for gold, copper and a suite of other elements.

934 quality control samples (blanks, standards and duplicates) were submitted with the core samples. From these core and control samples, 1,484 pulps (10%) were selected and analyzed by ALS Chemex Laboratory in Vancouver, B.C. as per the QAQC protocol. Additional QA/QC samples accompanied the Eco Tech same pulp assays that were sent to ALS Chemex, for a total of 1,033 QA/QC samples analyzed in 2010.

The 2010 exploration and geotechnical drilling program at the KSM project is summarized in Table 11-1 by zone.

Table 11-1: 2010 KSM Drilling Campaign

Mineral Zone/Area	No. Exploration Holes	Total Exploration Meterage	No. Geotech Holes	Total Geotech Meterage	Total Holes	Total Meterage
Kerr			4	1,453.00	4	1,453.00
Sulphurets	13	4,588.60	5	1,950.30	18	6,538.90
Mitchell	7	2,591.60	4	594.51	11	3,186.11
Iron Cap	38	14,377.80	3	1,022.85	41	15,400.65
Infrastructure (KSM)			17	1,681.02	17	1,681.02
Infrastructure (Seabee)			27	1,769.45	27	1,769.45
Total	58	21,558.00	60	8,471.13	118	30,029.13

11.2 Drill Hole Surveying

The procedures used for spotting the drill holes, surveying collars and down-hole surveying methods are the basically the same as those described for the 2007 drilling campaign (Lechner, 2008). The following section briefly describes how the drill hole collar locations were initially acquired and what steps were undertaken to translate those locations into the new coordinate system.

- Kerr Deposit - Previous to Seabridge's ownership of the property, the drill hole collars were located in a local mine grid system that was tied to the NAD27 datum by Placer Dome in the early 1990's. Seabridge personnel located nine Placer Dome drill hole collars and surveyed them with their handheld Trimble DGPS instrument. These re-surveyed locations along with the "original" coordinates for all Kerr holes were provided to Aero Geometrics. The drill hole collars were adjusted by Aero Geometrics from their original local grid to NAD27 using affine transformation and then further transformed into NAD83 using Canadian National Transformation v2.0. No elevation adjustments were made by Aero Geometrics and when the transformed drill hole coordinates were compared with the new Lidar based topographic surface it was apparent that some adjustment was required. The Kerr drill hole collars were adjusted to match the new NAD83 based topo surface.

- Sulphurets Deposit - Holes drilled prior to Seabridge's entry into the district were treated in the same manner as described for the Kerr deposit. Seabridge era drill holes were located in the field using a Trimble handheld DGPS unit. Depending on terrain, satellite coverage and other factors it is possible to achieve sub-meter accuracy. All of the Seabridge drill hole collars were originally located in NAD27 coordinates. These data were sent to Aero Geometrics who converted the drill hole collars to NAD83 coordinates. The translated drill hole collars were compared with the new Lidar topographic surface. This elevation of drill holes did not always conform to the Lidar survey and were adjusted to topography like was done for the Kerr drilling.

- Mitchell Deposit - The same procedures were used to locate Seabridge's Mitchell drill holes as was described for the Sulphurets holes. Falconbridge drill holes were located in the field using a standard DGPS unit. Like the other two deposits, the elevation for some of the drill holes was adjusted to match the new NAD83 Lidar topography.

All except three drill holes completed in 2010 were surveyed by McGladrey & Associates Professional Land Surveyors using control station corrected DGPS and processed using the CSRS PPP Service. Three holes (M-10-117, IC-10-028, IC-10-030) were surveyed by Seabridge personnel using a standard uncorrected DGPS unit. Comparisons on holes surveyed by both methods indicate that the surveys of these three holes are acceptable.

Table 11-2 summarizes total KSM drilling by company and mineral zone. Tables 11-3 through 11-6 breakdown the drilling for the Kerr, Sulphurets, Mitchell, and Iron Cap zones by company and year, respectively.

Table 11-2: KSM Drill Hole Summary by Company

Kerr			
Company	**Number**	**Meters**	**% of Total**
Brinco	3	189.90	1%
Western Canadian	36	5,324.56	18%
Newhawk Gold	2	115.21	0%
Sulphurets Gold	20	4,365.35	15%
Placer Dome	83	16,413.57	57%
Seabridge	11	2,611.75	9%
Total Kerr	***155***	***29,020.34***	***100%***

Sulphurets			
Company	**Number**	**Meters**	**% of Total**
Granduc	6	1,016.02	4%
Esso	14	2,275.23	9%
Newhawk Gold	7	1,306.30	5%
Placer Dome	23	5,577.34	22%
Falconbridge	7	1,648.09	7%
Seabridge	37	13,457.79	53%
Total Sulphurets	***94***	***25,280.77***	***100%***

Mitchell			
Company	**Number**	**Meters**	**% of Total**
Esso	1	210.00	0%
Newhawk Gold	4	647.30	1%
Falconbridge	4	1,197.29	2%
Seabridge	130	49,478.87	96%
Total Mitchell	***139***	***51,533.46***	***100%***

Iron Cap			
Company	**Number**	**Meters**	**% of Total**
Esso	5	1,051.26	6%
Falconbridge	5	1,246.60	7%
Seabridge	42	15,492.27	87%
Total Iron Cap	***52***	***17,790.13***	***100%***

Total KSM Project			
Company	**Number**	**Meters**	**% of Total**
Granduc	6	1,016.02	1%
Esso	20	3,536.49	3%
Brinco	3	189.90	0%
Western Canadian	36	5,324.56	4%
Newhawk Gold	13	2,068.81	2%
Sulphurets Gold	20	4,365.35	4%
Placer Dome	106	21,990.91	18%
Falconbridge	16	4,091.98	3%
Seabridge	220	81,040.68	66%
Grand Total	***440***	***123,624.70***	***100%***

Table 11-3: Kerr Drill Hole Summary by Company

Company	Year Drilled	Hole Pre-fix	No. Holes	No. Meters	% of Total
Brinco	1985	85-nnn	3	189.90	0.7%
Western Canadian	1987-1988	K87-nnn, K88-nnn, 88-nn	36	5,324.56	18.3%
Newhawk Gold	1988	T88-nnn	2	115.21	0.4%
Sulphurets Gold	1989	K89-nnn, T89-nnn	20	4,365.35	15.0%
Placer Dome	1992	KS-nnn, KS92-nnn	83	16,413.57	56.6%
Seabridge	2009	K-09-nn, MW-09-nna	7	1,158.75	4.0%
Seabridge	2010	K-10-nn	4	1,453.00	5.0%
Total	n/a	n/a	155	29,020.34	100.0%

Table 11-4: Sulphurets Drill Hole Summary by Company

Company	Year Drilled	Hole Pre-fix	No. Holes	No. Meters	% of Total
Granduc Mining Corp.	1962, 1968	S62-n, S68-n	6	1,016.02	4.0%
Esso Resources	1980, 1981	S80-nn, S81-nn	14	2,275.23	9.0%
Newhawk Gold Mines	1991	S91-nn	7	1,306.30	5.2%
Placer Dome	1992	SG92-nn	23	5,577.34	22.1%
Falconbridge	2005, 2006	MC-05-nn, MQ-05-nn, IF-05-nn	7	1,648.09	6.5%
Seabridge Gold	2006, 2008, 2009	S-06-nn, S-08-nn, S-09-nn, MW-09-nna	19	6,918.89	27.4%
Seabridge Gold	2010	S-10-nn	18	6,538.90	25.9%
Total	n/a	n/a	94	25,280.77	100.0%

Table 11-5: Mitchell Drill Hole Summary by Company

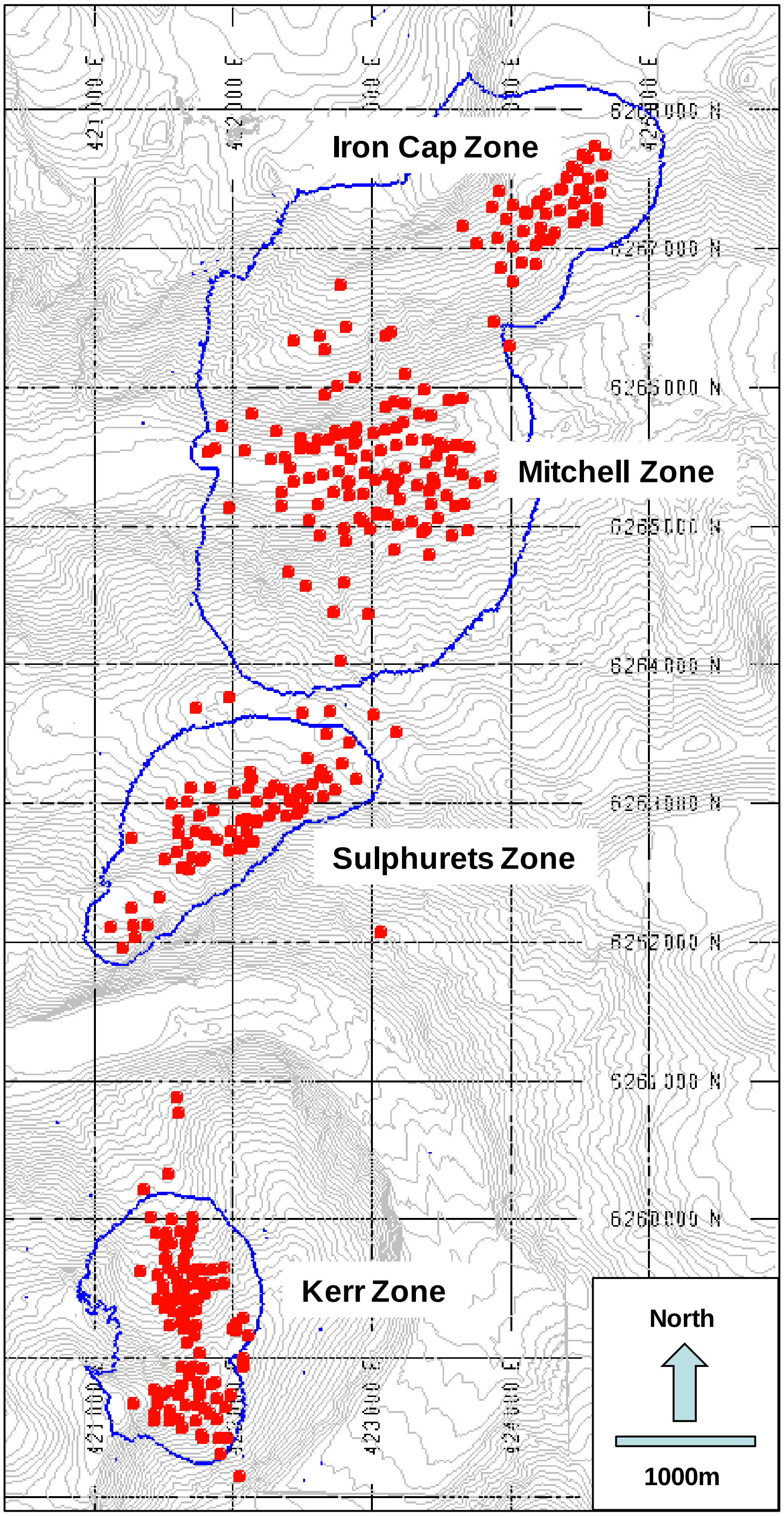
Company	Year Drilled	Hole Pre-fix	No. Holes	No. Meters	% of Total
Esso	1980	S80-9	1	210.00	0.4%
Newhawk Gold	1991	S91-nnn	4	647.30	1.3%
Falconbridge	2005	NM-05-nn, WM-05-nn	4	1,197.29	2.3%
Seabridge	2006	M-06-nnn	24	7,505.80	14.6%
Seabridge	2007	M-07-nnn	37	15,650.32	30.4%
Seabridge	2008	M-08-nnn	34	15,415.75	29.9%
Seabridge	2009	M-09-nnn, MW-09-nnA	24	7,720.89	15.0%
Seabridge	2010	M-10-nnn, KC10-nn	11	3,186.11	6.2%
Total	n/a	n/a	139	51,533.46	100.0%

Table 11-6: Iron Cap Drill Hole Summary by Company

Company	Year Drilled	Hole Pre-fix	No. Holes	No. Meters	% of Total
Esso	1980	S80-nn	5	1,051.26	5.9%
Falconbridge	2005	IC-05-nn	5	1,246.60	7.0%
Seabridge	2009	MW-09-nnA	1	91.62	0.5%
Seabridge	2010	IC-10-nnn	41	15,400.65	86.6%
Total	n/a	n/a	52	17,790.13	100.0%

Figure 11-1 is a drill hole collar plan map for the entire KSM project showing the areal distribution of drilling (collars shown as red dots) relative to the four mineralized areas.

Figure 11-1: KSM Drill Hole Locations



Figures 11-2 through 11-5 are drill hole collar maps for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively showing the collar location and hole trace in red. RMI's conceptual pits (pit number 3 as defined in Table 17-32) are shown in blue. For reference purposes, each drill hole collar map contains a reference line of section line for drill hole and block model cross sections shown in Section 17.8.

Figure 11-2: Kerr Drill Hole Locations

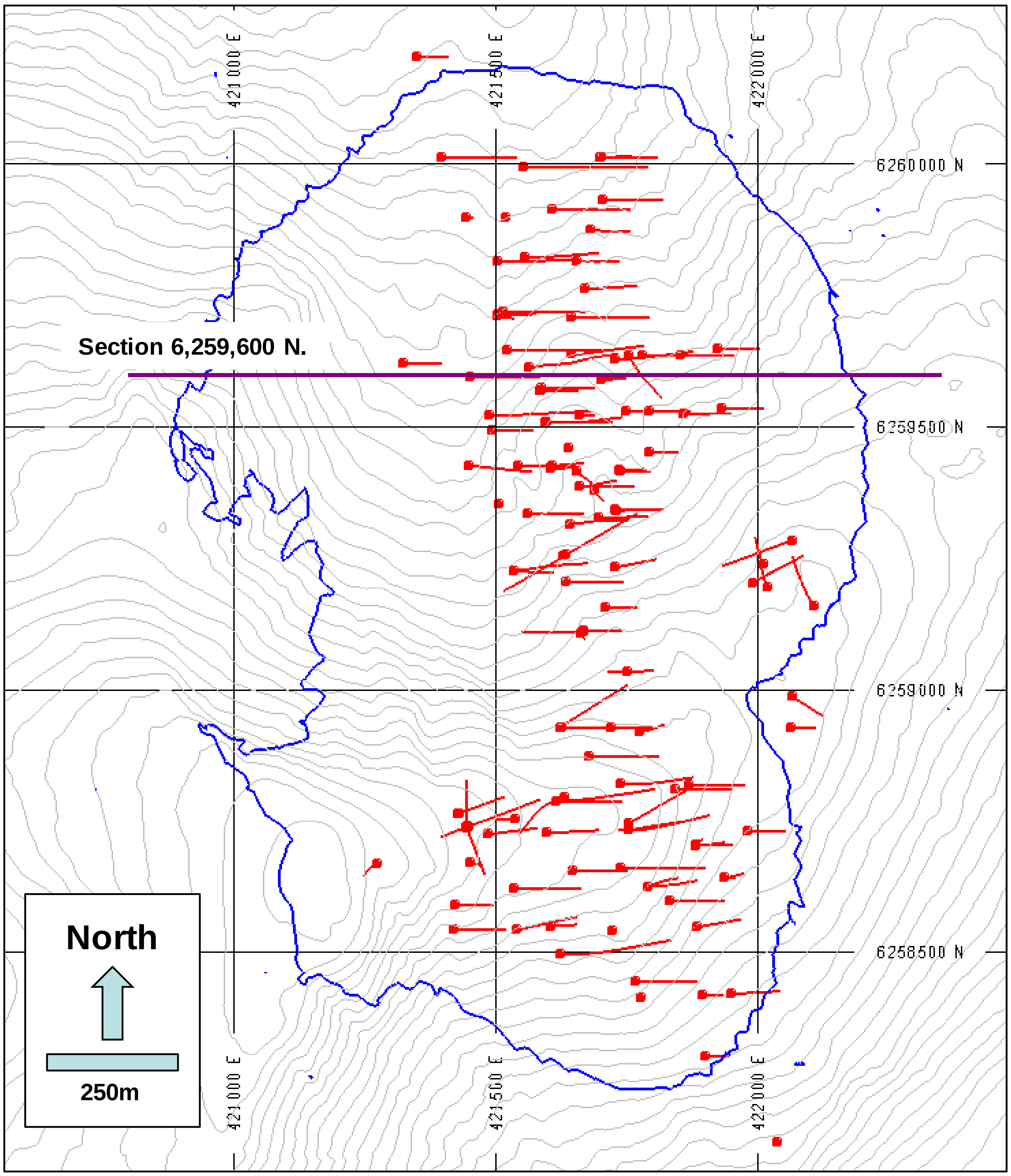


Figure 11-3: Sulphurets Drill Hole Locations

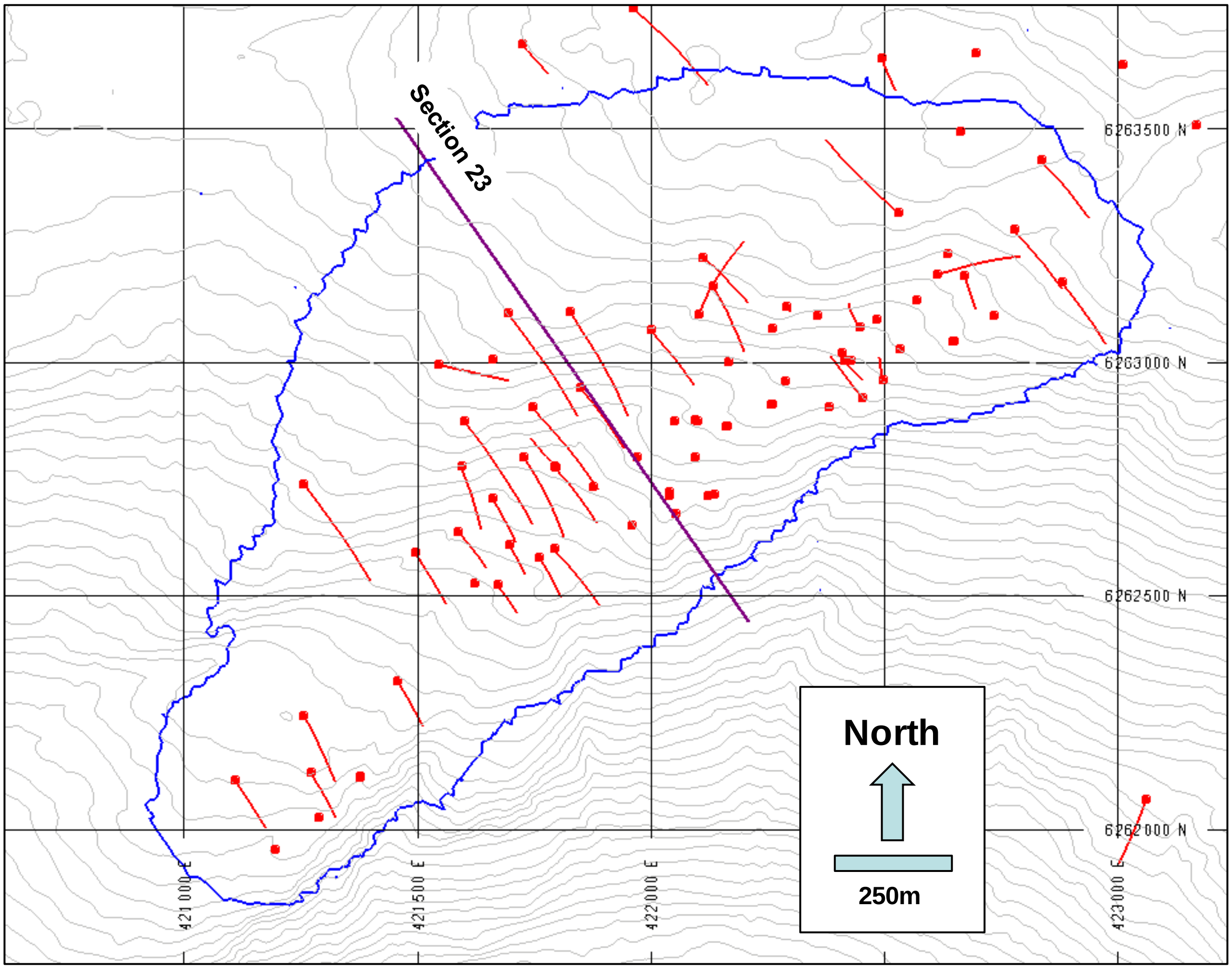


Figure 11-4: Mitchell Drill Hole Locations

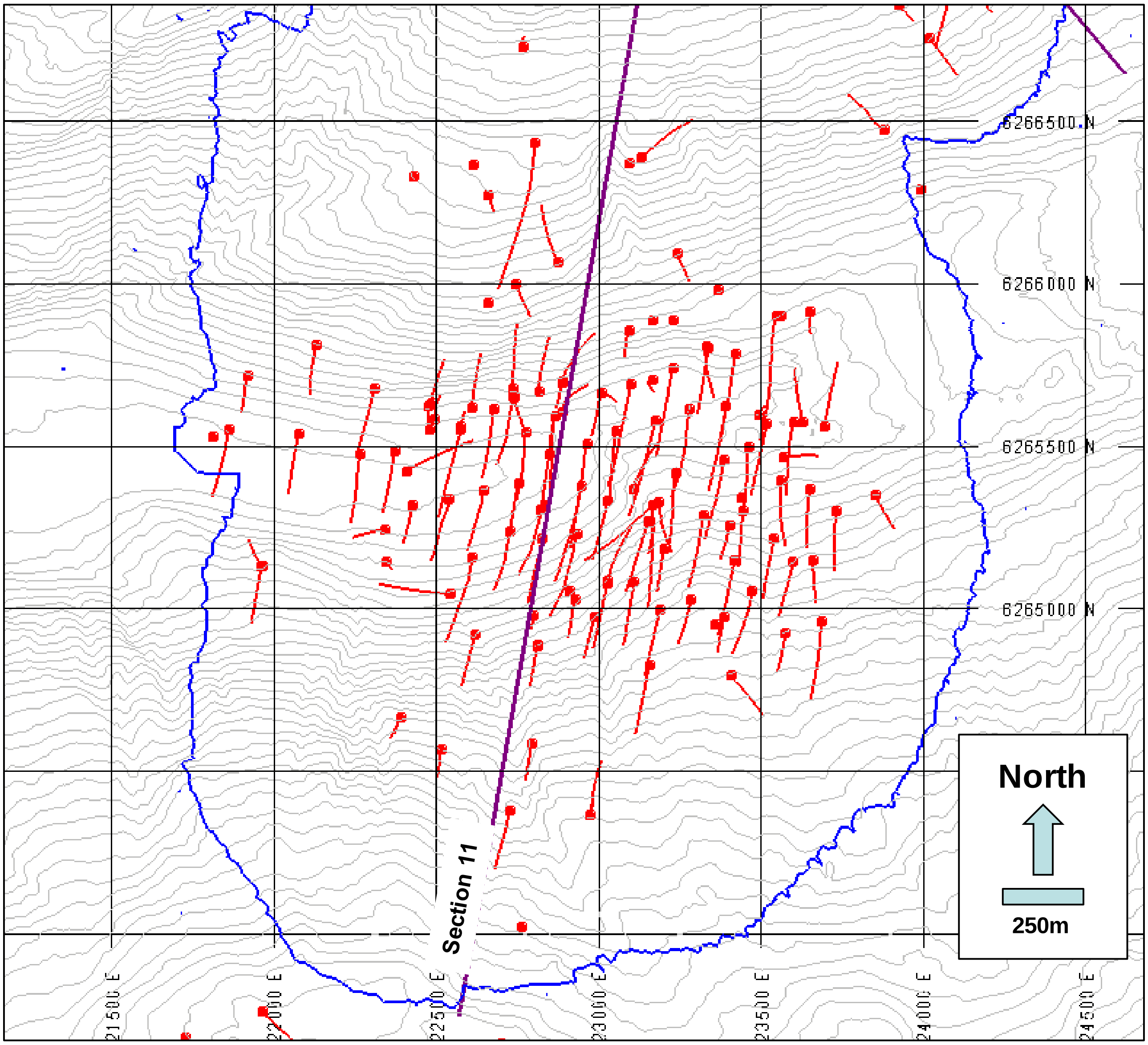
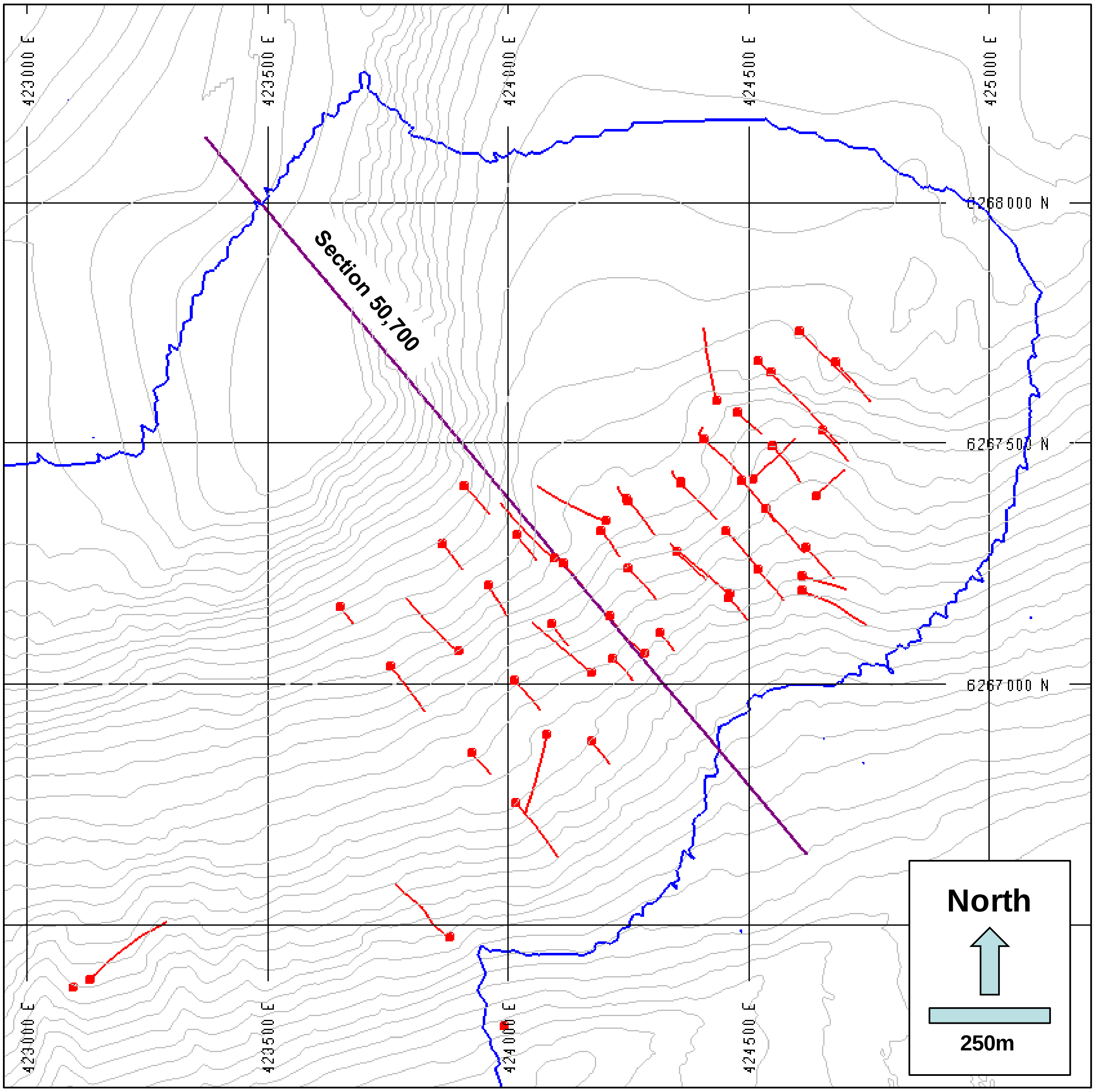


Figure 11-5: Iron Cap Drill Hole Locations



11.3 Drill Core Processing

The following section was taken directly from RMI's April 6, 2007 NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia" and edited to conform with protocol used in 2010:

"Drill core was placed into wooden trays directly upon emptying the core tube at the

drill site. A wooden block marked with the hole depth in meters was placed in the core trays upon the completion of each drill run, which in good conditions was three meters. Core tubes and rods were in metric lengths. The core boxes were covered with a plywood lid which was securely nailed to the core box and placed in a metal basket. The baskets were slung by helicopter to camp, typically after the morning shift change, depending on productivity and weather conditions.

At camp, the core basket was placed near the core logging shack. Each box was layed out in sequence on elevated racks in the core shed. The core was examined for condition, missing core, and depth tag errors. Boxes were labeled with black felt tip pens and embossed steel tags containing the hole number, depth, and box number. The core was then washed with fresh water. Geotechnical data including recovery, RQD, and natural breaks were recorded for each drill run, as marked by the wooden core run blocks. This information was recorded by the geologist or trained logging assistant under direct supervision of a geologist.

The geologist then recorded key geologic information including lithology, alteration, structure, and mineralization using a pre-determined format and coding system that is shown in Table 12-1 through 12-3. The data were recorded on paper logging sheets which were then entered into the digital database at the camp office. The geologist or assistant under the direct supervision of the geologist marked sample intervals on the core at fixed 2-meter-long intervals or at geological contacts so that each sample was approximately 2 meters maximum length. Sample lengths of 2 meters followed Falconbridge Ltd.'s protocol for copper-gold porphyry prospects which is in line with accepted industry practices for this style of mineralization.

The core at the beginning of each sample was marked with a wax pencil, and a Teflon coated paper tag with a unique identification number was stapled to the core box adjacent to the wax marking. Duplicates of the paper tag with the identification number were also placed were placed on the sample bag that was sent to the assay lab. A third copy of the tag, with the identification number, hole number and depth interval was stored. This information was entered into the digital database assay table. The entire hole (excluding any recovered overburden) was sampled. The core was then digitally photographed. All digital photo files are maintained in the company's digital database. Where necessary, a wax pencil was then used to mark a cut line along the top of the drill core to avoid any sampler induced selection bias and to ensure that the same side of the halved core relative to its placement in the box was put into the sample bag that was sent for assay".

11.4 **Relationship Between Drill Hole and Mineralization Orientation**

At Mitchell, most of the holes were drilled at a pre-assigned azimuth and dip of 190° and -60°. Orientation of mineralization has been difficult to determine from surface mapping and sampling as it is finely disseminated and pervasive with no obvious alteration control or relationship to vein density or orientation. It has been assumed that the Mitchell mineralization is likely orientated similar to the intense foliation and sheeted, deformed quartz stockwork veining, which generally dips at -70° along a N10°E azimuth. The assigned drill hole orientation was chosen to cut this orientation as close to perpendicular as practical. At Mitchell, there is sufficient drilling to conclude that the deposit is aligned along this orientation. However in a gross sense the zone has a cylindrical geometry that plunges at about -45° to the northwest. Thus drilled intervals may be slightly oblique to the mineralization trend and may not accurately reflect true thicknesses, although most holes did not completely penetrate the mineralized zone.

At Sulphurets, the historical and current drilling orientation is along an azimuth of 145 inclined at -60°. The general northeasterly strike here appears to reflect a strong stratigraphic control, The strong deformation and schistosity present at Mitchell is not as prevalent at Sulphurets, likely due to the weaker degree and extent of late phyllic alteration, and there is no apparent alienation along the same trend. The plunge direction of -45° to the northwest observed at Mitchell also seems to define the orientation of higher grade zones and breccias within Sulphurets. In general, the drilled intervals of mineralization here are believed to be closer to representing true thicknesses.

Similar to Mitchell, extensive stockwork controlled disseminated mineralization also is found at Kerr along with strong phyllic-argillic alteration. However the associated schistosity dips moderately to the west. The geometry of the deposit is strongly lineated along this trend, which the preferred historical and current drill direction (dipping moderately east) was designed to test. Here the drilled intervals of mineralization are believed to closely indicate the true thickness.

At Iron Cap, mapping, surface sampling, and drilling prior to 2010 had established two dominant structural trends that influenced orientation of mineralization. A regional foliation as observed at other zones is also evident at Iron Cap, which is generally striking from 090 to 120 degrees and dipping moderately to steeply north and north-northwest, and has attenuated pre-existing mineralization. There are several recognized veinlet and fracture orientations. However, the dominant one is a later feature which controls several centimeter to multi-meter scaled quartz-sulfide veins and trends from 020 to 040 degrees and dips steeply to the west. Stratigraphic bedding is obliterated within the Iron Cap zone, but beyond the most intense alteration and within the Iron Cap panel, it generally strikes east-west and dips north. Based on these observations, and knowledge of mineralization orientations at Kerr, Sulphurets, and Mitchell, it was determined that drilling inclined holes at an azimuth of 135 degrees would satisfactorily test all of the dominant structural trends with the least bias. In addition, several holes were drilled at a variety of other azimuths and inclinations to test for possible directional bias.

12.0 SAMPLING METHOD AND APPROACH

Seabridge implemented the same sampling methods in 2009 that were initially developed in 2005-2006. Previous technical reports written by the author discussed sampling methods for prior programs (Lechner 2007, Lechner 2008a, Lechner 2008b, Lechner 2009, and Lechner 2010).

12.1 Sample Length

The 2010 drill core was sawn in half longitudinally into primarily 2-meter-long samples, which were then shipped off site where they were assayed for gold, copper, and other metals. Of the 13,364 samples that were collected, 16 percent were less than 2-meters-long, 76% were exactly 2-meters-long and 8% were longer than 2-meters. In 2010, approximately 98% of drilled meterage was assayed. Approximately 99% of the 2010 drill holes used variations of NQ drilling tools and the remainder with HQ tools. The 74 exploration and geotechnical holes that were drilled in 2010 and used for resource estimation averaged about 359 meters in length. After completing the 2010 drilling campaign, the Kerr deposit has been drilled on roughly 50 to 75m centers over an area which measures about 1700m in the north-south direction and 250m in the east-west direction. The Sulphurets zone has been drilled to about 50m to 100m centers over an area measuring about 1000m (northeast-southwest) by 250m (northwest-southeast). The Mitchell zone has been drilled to roughly 50-meter to 100-meter centers over an area measuring 1400m (east-west) by 900m (north-south). There are areas of wider and closer spaced drilling in each deposit primarily driven by difficulty in constructing drilling platforms in steep terrain. The Iron Cap zone has been drilled on roughly 50 to 100m centers covering an area measuring 1,500 meters by 600 meters.

Based on the style of mineralization, it is the author’s opinion that the 2-meter-long sample lengths are reasonable and appropriate.

12.2 Drilling Conditions

Drilling conditions were generally good. Overburden was not excessive and rock quality was typically high except in isolated fractured or sheared zones where the rock easily broke along foliation planes. Overall average rock quality designation (RQD) for the 2010 drilling campaign was about 73 percent and core recovery averaged about 96%. The frequency of natural breaks averaged about 5 per meter. RQD tended to be poorer for the Kerr and Sulphurets zones where the average RQD's were 67% and 65%, respectively. Core recovery for the 2010 drilling at Kerr, Sulphurets, Mitchell, and Iron Cap zones in 2010 was 91%, 98%, 92%, and 97%, respectively.

12.3 Sample Quality

As a result of strict adherence to the drilling procedures and sampling methods described above, sample quality and representation are considered good to high. Core recovery rates improved in 2010 with only 3%, 1%, 4%, and 3% of the Kerr, Sulphurets, Mitchell, and Iron Cap intervals having recoveries less than 50%, respectively.

12.4 **Geology and Geological Controls**

The following sections were taken directly from a prior RMI report (Lechner, 2007) and are still relevant regarding geologic controls.

"There has been some discussion regarding geology and controls at Mitchell Creek in previous sections. The deposit is considered to be within the spectrum of the gold-enriched copper porphyry environment and metals, chiefly gold and copper (in terms of economic value), are generally at low concentrations. Mineralization is typically finely disseminated, stockwork or sheeted veinlet controlled and pervasively dispersed over dimensions of hundreds of meters. Grades diminish slowly over large distances; sub-economic grades are encountered at distances of several hundreds of meters beyond the interpreted center of the system.

Due to the intensity of hydrothermal alteration, especially at Mitchell Creek, it is difficult or impossible to recognize original protoliths. This is most pronounced in phyllic or quartz-sericite-pyrite altered rocks. In chlorite-sericite (logged as IARG or intermediate argillic) and propylitic altered rocks, a homogeneous, tuffaceous texture is often observed, thus the host is likely intermediate volcanic tuffs or volcaniclastics. Diffuse, ghost-like porphyritic textures may reflect dykes of the Mitchell intrusions. Rare, meter scale aphanitic intermediate dykes are post-alteration and unmineralized.

At Mitchell Creek, there appears to be a spatial association between the highest continuous copper and gold grades with an area of chlorite-magnetite alteration as recognized by Britton, et. al., where the rocks appear to be partially overprinted by phyllic alteration, particularly along the western edge of the intensely phyllic altered exposed bluffs, located at the east side of the zone. Roughly coincident with the area of highest Cu and Au mineralization are lower Mg and Na concentrations as determined by ICP analyses. These may be useful in defining domains for the purposes of resource estimation. There is no clear association with other recorded attributes, including lithology, quartz vein frequency and intensity, or alteration types".

12.5 **Lithological and Alteration Coding**

"In 2006, Seabridge adopted lithological and alteration descriptions from Fowler and Wells (1995), which distinguished rocks above the Sulphurets Thrust fault from those below it. A similar distinction was made with the Mitchell Thrust fault, where the rocks located between the Sulphurets and Mitchell faults were seen to be comprised of similar lithologies as those located above the Sulphurets fault. In 2007, Seabridge simplified the lithologic and alteration coding so that less emphasis was placed on the location of the samples relative to the regional structures and the more emphasis was placed on describing the samples. The lithologic and alteration codes stored in the 2007 drill hole database are summarized in Tables 12-1 and 12-2, respectively. Other key logged

attributes include a numerical alteration intensity from 0 (absent) to 6 (intense), percentage of quartz and pyrite and quartz veinlet frequency".

"At Mitchell Creek, the IARG (intermediate argillic) alteration unit is more likely a transitional unit between propylitic and phyllic assemblages where chlorite has only been partially sericitized. Seabridge will try to verify by ongoing studies".

Table 12-1 Lithologic Codes

Lithologic Code	Lithology
OVBD	Andesite
ANDS	Intermediate Volcanics, Massive Flows/Tuffs
IVOL	Andesite Lapilli Tuff
VALT	Andesite Tuff
VATF	Overburden
QTVN	Quartz vein
PHBX	Hydrothermal Breccia
PSBX	Siliceous Hydrothermal Breccia
DDRT	Diorite/mafic intrusive
GRAN	Granitic porphyry
PPFP	Feldspar Porphyry Intrusions
PQMZ	Quartz Monzonite
PMON	Porphyritic Monzonite
VAAT	Andesite Ash Tuff
VAXT	Andesite Crystal Tuff
VU	Volcanic, unknown protolith (intensely altered)
VUAT	Unknown Ash Tuff
VULT	Unknown Lapilli Tuff
VUTF	Unknown Tuff
VUXT	Unknown Crystal Tuff
SARG	Volcaniclastics/Argillites
SCHT	Schist, unknown protolith (intensely altered)
SEDS	Undifferentiated seds
CCSD	Chert/chemical seds
SSLT	Siltstone
FLTZ	Fault Zone
NREC	No recovery

Table 12-2 Alteration Codes

Alteration Code	Alteration Description
CARB	Carbonate veining, fault related
CL	Chlorite alteration
FEOX	Fe-Oxides due to weathering
HEM	Hematization of intrusives
IARG	Intermediate Argillic - green Ser, Chl, Py
KP	Potassic - K-Fd,Qt,Py,Cp (Porphyry)
PKBX	Potassic - K-Fd,Qt,Ser,Py,Cp (Hydrothermal Breccia)
PR	Propylitic - Chl,Ep,Py,Carb,Mag
PSBX	Silica Flooding - Qt,Ser,Py,Tour,Py (carb) (Hydrothermal Breccia)
QA	Albitic (core area) - Ab,Cb,Chl,Py,Cp,Ser (Porphyry)
QB	Potassic - Bio,Qt,Py,Cp (Chl,Ser,Mo) (Porphyry)
QSP	Phyllic - Qt,Ser,Tour,Py, remnant Ks,Cp,Mo (Hydro. Breccia+porphyhry)
QSPSTW	Phyllic - Qt,Ser,Py (>60% qtz veinlets)
QTVN	Late Quartz Veins
SI	Silica Flooding - Qt,Py,Cp (Tour,Ser) (Porphyry)
SIH	Silicification due to Hornfelsing - Qt, Py
SIL	Pervasive silicification

12.6 **Relevant Sample Composites**

Tables 12-3 through 12-6 show relevant composited drill hole grades for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. The relevant composites reflect continuous down-hole intersections of material above a 0.50 g/t gold equivalent cutoff grade in excess of 50 meters in length. Gold and copper prices of US $650 per ounce and US $2.00 per pound along with gold and copper recoveries of 70% and 85%, respectively were used to determine the gold equivalent cutoff grade. The composited lengths shown in Tables 12-3 through 12-6 are not necessarily "true widths" of mineralization although they represent significant zones of mineralization typical of large scale low-grade deposits.

Table 12-3: Relevant Kerr Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
88-011	51.00	163.85	112.85	0.41	1.31	3.73
88-021	162.10	213.05	50.95	0.54	1.17	3.53
88-001	176.17	228.25	52.08	0.30	1.22	3.43
KS92-135	28.96	92.05	63.09	0.40	1.15	3.35
KS-073	16.80	125.50	108.70	0.32	1.16	3.28
KS-086	26.30	77.40	51.10	0.53	1.05	3.22
KS-087	138.74	195.95	57.21	0.59	0.97	3.07
KS-075	23.20	149.40	126.20	0.29	0.98	2.81
K89-007	70.30	138.10	67.80	0.37	0.91	2.69
KS-082	27.40	87.90	60.50	0.22	0.95	2.65
KS-091	3.00	72.60	69.60	0.56	0.81	2.62
KS-066	76.20	146.00	69.80	0.37	0.87	2.61
KS-071	111.00	177.00	66.00	0.39	0.87	2.61
KS-094	298.50	382.30	83.80	0.35	0.86	2.56
KS92-138	55.78	135.67	79.89	0.38	0.79	2.41
K89-006	57.20	114.00	56.80	0.32	0.92	2.40
KS-124	259.00	331.00	72.00	0.56	0.71	2.39
K89-005	53.85	127.70	73.85	0.32	0.79	2.34
K89-010	101.00	178.05	77.05	0.20	0.80	2.26
KS-067	135.00	185.30	50.30	0.43	0.71	2.26
88-018	3.05	75.85	72.80	0.36	0.72	2.20
KS92-141	59.00	110.30	51.30	0.30	0.72	2.16
KS-123	124.05	238.50	114.45	0.35	0.70	2.13
KS-125	130.15	212.25	82.10	0.33	0.69	2.10
88-015	130.00	196.00	66.00	0.26	0.71	2.09
KS-067	188.70	256.30	67.60	0.39	0.66	2.08
K89-006	120.00	187.22	67.22	0.30	0.66	2.00
T89-011	213.00	319.40	106.40	0.27	0.66	1.97
KS-076	8.60	90.00	81.40	0.16	0.69	1.92
KS-106	57.30	128.20	70.90	0.19	0.66	1.87
KS-128	149.96	297.40	147.44	0.23	0.63	1.85
K89-003	58.00	136.40	78.40	0.28	0.61	1.84
KS-131	141.10	192.00	50.90	0.25	0.61	1.83
K-10-08	137.30	196.80	59.50	0.44	0.54	1.83
K89-002	20.75	101.19	80.44	0.37	0.56	1.82
KS-123	241.15	299.70	58.55	0.27	0.59	1.79
KS-120	38.40	93.57	55.17	0.24	0.60	1.78
T89-013	15.24	90.00	74.76	0.57	0.47	1.77
KS-125	262.40	324.90	62.50	0.31	0.57	1.77
KS-109	69.00	179.00	110.00	0.28	0.57	1.74
K-09-01	218.17	276.00	57.83	0.22	0.59	1.74
KS-089	173.60	255.00	81.40	0.29	0.56	1.72
K-09-01	277.50	344.28	66.78	0.18	0.60	1.72
K87-005	10.30	62.90	52.60	0.41	0.50	1.71
K-10-08	205.00	263.00	58.00	0.21	0.57	1.68

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
88-016	3.05	106.07	103.02	0.28	0.54	1.64
KS-127	158.80	209.40	50.60	0.25	0.52	1.57
KS-089	12.10	70.40	58.30	0.30	0.49	1.56
K-10-06	105.50	160.70	55.20	0.31	0.48	1.54
KS92-143	115.50	186.50	71.00	0.34	0.45	1.51
KS-094	21.90	75.80	53.90	0.26	0.48	1.48
KS-127	33.50	151.80	118.30	0.21	0.49	1.47
KS-077	148.90	256.00	107.10	0.18	0.49	1.44
KS-119	136.00	231.66	95.66	0.20	0.48	1.42
KS92-136	95.10	160.00	64.90	0.20	0.48	1.42
K89-004	94.00	239.88	145.88	0.20	0.48	1.42
KS-123	24.00	108.81	84.81	0.24	0.45	1.40
KS-111	3.05	69.00	65.95	0.20	0.45	1.35
KS-127	212.45	268.90	56.45	0.26	0.42	1.33
T89-008	82.00	175.00	93.00	0.21	0.43	1.32
KS-112	5.18	75.80	70.62	0.18	0.44	1.30
K-09-02	93.00	201.00	108.00	0.25	0.41	1.29
K-10-06	29.90	99.00	69.10	0.18	0.43	1.28
KS-081	70.00	143.30	73.30	0.22	0.41	1.27
KS-108	64.00	134.11	70.11	0.19	0.41	1.25
T89-014	140.00	203.00	63.00	0.28	0.38	1.24
KS-124	79.00	253.00	174.00	0.22	0.40	1.24
KS-067	12.30	87.00	74.70	0.27	0.37	1.22
T89-011	105.00	210.00	105.00	0.21	0.38	1.18
KS-116	24.38	87.00	62.62	0.20	0.37	1.16
KS-117	3.70	78.33	74.63	0.21	0.37	1.15
K89-019	159.00	361.49	202.49	0.13	0.40	1.14
KS-116	144.00	218.00	74.00	0.14	0.39	1.13
KS-105	8.15	94.49	86.34	0.19	0.35	1.10
KS-126	80.60	141.80	61.20	0.20	0.35	1.09
T89-008	4.57	76.00	71.43	0.15	0.36	1.06
KS-131	43.00	105.00	62.00	0.19	0.34	1.06
KS-130	28.04	110.64	82.60	0.18	0.34	1.05
KS-115	159.90	215.00	55.10	0.15	0.34	1.02
KS92-139	3.66	54.56	50.90	0.19	0.32	1.01
KS-121	89.70	162.46	72.76	0.18	0.32	1.01
KS-104	36.30	87.50	51.20	0.15	0.33	1.00
88-022	2.74	55.00	52.26	0.17	0.32	1.00
KS-107	57.91	114.40	56.49	0.16	0.32	0.99
K89-019	105.00	156.00	51.00	0.20	0.29	0.95
KS-121	165.50	218.10	52.60	0.14	0.31	0.94
KS-116	239.30	302.05	62.75	0.15	0.28	0.88
KS-088	102.80	169.60	66.80	0.15	0.29	0.88
KS-122	197.00	251.00	54.00	0.18	0.27	0.86
Average	**109.40**	**183.27**	**73.87**	**0.33**	**0.77**	**2.30**

Table 12-4: Relevant Sulphurets Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
SG92-02	84.00	166.60	82.60	1.32	0.86	3.14
S-09-10	400.42	494.60	94.18	0.58	0.71	2.41
S91-389	71.10	166.70	95.60	0.70	0.64	2.34
S-10-17	290.00	359.00	69.00	1.04	0.49	2.25
S-10-21	287.00	351.70	64.70	0.70	0.60	2.19
S-06-04	188.00	310.00	122.00	0.80	0.52	2.14
S-09-11	183.00	354.00	171.00	0.73	0.55	2.13
MW-09-07A	125.85	182.50	56.65	0.68	0.55	2.09
S-08-08	274.00	344.20	70.20	0.89	0.46	2.06
S-09-10	326.00	399.55	73.55	0.73	0.52	2.05
SG92-07	232.00	291.69	59.69	0.69	0.51	2.00
S-09-14	131.40	263.50	132.10	0.76	0.48	1.98
S-10-22	171.00	249.00	78.00	0.60	0.54	1.98
SG92-15	116.13	190.40	74.27	1.57	0.15	1.95
S-09-13	75.00	140.00	65.00	0.75	0.47	1.94
S-09-15	263.00	351.00	88.00	0.65	0.48	1.89
SG92-19	14.00	93.80	79.80	1.69	0.05	1.83
S91-388	52.90	104.30	51.40	0.55	0.50	1.77
S81-23	4.80	62.08	57.28	1.37	0.14	1.72
S-10-23	304.00	406.50	102.50	0.74	0.39	1.72

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
SG92-12	131.00	261.00	130.00	0.58	0.44	1.70
S80-12	35.00	166.24	131.24	1.29	0.15	1.65
SG92-10	218.00	292.60	74.60	0.57	0.41	1.62
S-09-15	107.00	204.05	97.05	0.55	0.42	1.61
SG92-23	159.70	224.27	64.57	0.54	0.40	1.56
S-10-18	29.20	113.00	83.80	0.64	0.33	1.47
SG92-04	19.00	87.00	68.00	0.51	0.36	1.43
S91-398	12.10	69.00	56.90	0.35	0.45	1.38
SG92-13	25.00	144.82	119.82	0.57	0.30	1.33
S-10-28	90.40	164.00	73.60	0.85	0.18	1.30
S81-24	3.00	60.40	57.40	1.02	0.09	1.25
S-10-17	158.00	218.00	60.00	0.46	0.28	1.16
S81-39	93.00	150.00	57.00	0.98	0.05	1.12
S91-391	97.90	182.40	84.50	0.67	0.11	0.95
MQ-05-01	172.00	222.00	50.00	0.23	0.27	0.90
S-10-30	113.00	168.00	55.00	0.30	0.23	0.87
S68-1	156.67	214.27	57.60	0.38	0.18	0.84
S-10-20	53.00	103.00	50.00	0.47	0.14	0.83
MC-05-02	126.00	180.00	54.00	0.13	0.27	0.82
Average	**179.68**	**264.06**	**84.38**	**0.86**	**0.49**	**2.09**

Table 12-5: Relevant Mitchell Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
M-08-086	200.00	306.00	106.00	0.22	1.22	2.83
M-07-058	297.65	432.00	134.35	0.99	0.35	1.90
M-06-017	17.00	80.60	63.60	1.14	0.29	1.89
M-08-067	384.00	469.00	85.00	0.48	0.62	1.86
MW-09-06A	3.95	87.40	83.45	1.03	0.30	1.81
M-07-048	6.40	58.80	52.40	0.92	0.34	1.79
M-06-009	4.00	296.00	292.00	0.98	0.31	1.78
M-07-029	53.30	163.80	110.50	1.21	0.26	1.76
M-06-007	4.40	287.90	283.50	0.98	0.29	1.72
M-08-065	4.00	380.10	376.10	0.96	0.29	1.69
M-07-051	28.50	146.25	117.75	0.92	0.30	1.68
M-07-035	146.00	484.00	338.00	1.03	0.25	1.68
M-07-059	2.50	152.25	149.75	0.94	0.29	1.67
M-08-090	2.50	169.90	167.40	0.96	0.28	1.67
M-06-013	4.85	105.10	100.25	0.94	0.27	1.63
M-07-058	176.00	288.00	112.00	0.80	0.32	1.63
M-07-054	3.50	446.00	442.50	0.92	0.27	1.62
M-08-076	9.58	238.78	229.20	0.98	0.25	1.61
M-07-049	0.00	396.85	396.85	1.12	0.22	1.59
M-06-017	99.00	164.60	65.60	0.89	0.27	1.58
M-08-077	15.00	133.40	118.40	0.91	0.26	1.57
M-07-055	121.60	177.65	56.05	0.88	0.26	1.56
M-06-011	3.70	297.00	293.30	0.85	0.27	1.54
M-07-045	300.90	630.00	329.10	1.05	0.21	1.53
M-07-025	9.00	465.00	456.00	0.84	0.27	1.52
M-08-086	336.00	544.14	208.14	0.89	0.28	1.51
M-07-035	516.00	574.30	58.30	0.79	0.28	1.50
M-07-055	6.10	101.57	95.47	0.94	0.22	1.50
M-06-013	107.60	248.00	140.40	0.79	0.27	1.48
M-07-024E	358.80	597.25	238.45	0.83	0.25	1.48
M-07-058	4.50	146.00	141.50	0.81	0.26	1.47
M-08-069	1.20	79.00	77.80	0.85	0.24	1.45
S91-395	0.00	114.10	114.10	0.74	0.28	1.45
M-08-067	471.00	714.00	243.00	0.69	0.30	1.45
S91-387	0.00	60.30	60.30	0.91	0.19	1.41
M-07-050	3.05	123.45	120.40	0.96	0.17	1.40
M-09-095	110.53	205.65	95.12	0.84	0.22	1.39
M-08-094	2.50	195.00	192.50	0.80	0.23	1.39
M-06-014	269.00	453.00	184.00	0.92	0.18	1.38
M-06-006	6.00	221.70	215.70	0.91	0.21	1.37
M-08-079	313.00	399.00	86.00	0.19	0.45	1.34
M-08-069	81.00	586.00	505.00	0.78	0.22	1.34
M-06-008	34.00	346.00	312.00	0.83	0.20	1.33
M-06-002	102.00	426.00	324.00	0.85	0.19	1.33
M-07-047	198.00	410.15	212.15	0.61	0.27	1.30
M-07-026	24.00	376.20	352.20	0.82	0.19	1.30
M-07-056	4.57	257.50	252.93	0.88	0.16	1.30
M-08-073	91.00	374.00	283.00	0.78	0.20	1.29
M-08-070	172.00	225.00	53.00	0.68	0.24	1.28
M-06-003	5.00	208.00	203.00	0.85	0.16	1.27
WM-05-01	81.50	282.89	201.39	0.80	0.19	1.27
M-07-051	147.40	259.70	112.30	0.66	0.23	1.26
M-07-059	155.60	285.00	129.40	0.61	0.25	1.26
M-07-034	42.00	126.79	84.79	0.70	0.22	1.26
M-08-064	23.00	345.00	322.00	0.84	0.17	1.26
M-07-035	72.00	144.00	72.00	0.80	0.18	1.25
M-08-092	316.00	418.00	102.00	0.73	0.20	1.25
M-07-058	448.00	565.00	117.00	0.63	0.24	1.25
M-07-039	41.00	116.00	75.00	0.73	0.20	1.24
M-07-052	13.70	96.05	82.35	0.82	0.16	1.24
M-07-028	74.37	237.00	162.63	0.79	0.17	1.23
M-08-067	78.00	372.00	294.00	0.64	0.26	1.23
M-07-031	76.00	214.00	138.00	0.69	0.21	1.21
M-07-053	124.00	442.00	318.00	0.75	0.17	1.20
M-07-045	128.00	300.00	172.00	0.78	0.16	1.20
M-07-037	143.85	309.00	165.15	0.85	0.13	1.19
M-07-048	164.00	342.40	178.40	0.72	0.18	1.18
S91-386	0.00	153.70	153.70	0.73	0.18	1.18
M-08-086	545.12	718.00	172.88	0.60	0.23	1.17
M-06-024	110.00	356.80	246.80	0.66	0.20	1.16
M-07-026	377.90	472.72	94.82	0.71	0.18	1.16
M-07-052	98.30	210.31	112.01	0.74	0.17	1.16
M-06-001	5.70	306.00	300.30	0.81	0.13	1.15
M-08-086	43.74	120.00	76.26	0.71	0.17	1.14
M-08-092	53.00	314.00	261.00	0.68	0.18	1.14
M-08-090	173.24	597.00	423.76	0.55	0.23	1.14
M-08-077	135.05	271.00	135.95	0.65	0.19	1.13
S91-395	116.50	190.50	74.00	0.60	0.21	1.13
M-08-093	119.00	645.00	526.00	0.65	0.19	1.13
M-06-017	166.10	223.00	56.90	0.63	0.20	1.12
M-07-027	179.85	231.00	51.15	0.67	0.18	1.12
M-08-062	313.00	571.13	258.13	0.54	0.22	1.10
M-09-109	2.10	201.00	198.90	0.71	0.15	1.08
M-08-065	488.00	592.60	104.60	0.59	0.19	1.07
M-08-091	124.00	408.00	284.00	0.67	0.16	1.06
M-07-054	600.00	670.45	70.45	0.58	0.19	1.06
M-08-062	572.02	745.00	172.98	0.55	0.20	1.06
M-07-058	567.00	720.00	153.00	0.49	0.22	1.05
M-09-099	259.00	337.50	78.50	0.66	0.15	1.05
M-08-061	273.00	599.30	326.30	0.71	0.13	1.04
M-09-096	3.50	191.00	187.50	0.80	0.09	1.04
M-08-065	384.55	482.00	97.45	0.54	0.19	1.04
M-06-010	53.00	198.00	145.00	0.66	0.14	1.03
M-06-002	3.00	100.00	97.00	0.70	0.12	1.02
M-07-057	61.20	171.00	109.80	0.59	0.17	1.02
M-09-107	74.00	148.00	74.00	0.22	0.31	1.01
M-07-047	9.75	89.00	79.25	0.68	0.13	1.01
M-07-037	6.00	143.25	137.25	0.73	0.11	1.01
M-08-062	32.00	100.75	68.75	0.65	0.14	1.00
M-07-054	470.00	544.28	74.28	0.55	0.18	1.00
M-06-003	210.00	310.00	100.00	0.62	0.14	0.99
M-09-099	510.50	599.50	89.00	0.52	0.18	0.99
M-08-094	197.00	339.00	142.00	0.56	0.17	0.99
M-08-063	290.00	569.00	279.00	0.66	0.13	0.99
M-08-071	99.00	154.00	55.00	0.66	0.13	0.98
M-06-012	185.00	265.00	80.00	0.60	0.14	0.96
M-07-050	151.25	237.00	85.75	0.62	0.13	0.94
M-06-005	5.60	108.00	102.40	0.54	0.15	0.94
M-08-066	94.00	435.00	341.00	0.66	0.11	0.93
M-07-034	248.00	298.00	50.00	0.52	0.15	0.91
M-08-072	80.50	139.00	58.50	0.63	0.11	0.91
M-07-060	285.00	338.00	53.00	0.46	0.17	0.89
M-06-015	2.90	206.00	203.10	0.63	0.10	0.89
M-07-034	300.00	368.00	68.00	0.49	0.15	0.89
M-09-107	175.00	233.00	58.00	0.53	0.13	0.87
M-08-073	390.00	442.00	52.00	0.47	0.16	0.87
M-07-046	66.00	123.00	57.00	0.48	0.15	0.86
S91-387	61.40	123.90	62.50	0.51	0.14	0.86
M-06-014	83.00	137.00	54.00	0.64	0.09	0.86
M-10-117	5.20	122.90	117.70	0.54	0.12	0.86
M-07-048	343.80	394.39	50.59	0.47	0.15	0.86
M-07-044	466.00	553.00	87.00	0.57	0.11	0.86
M-07-057	257.20	308.00	50.80	0.46	0.15	0.85
M-08-076	320.09	408.00	87.91	0.49	0.14	0.85
M-07-057	173.00	250.75	77.75	0.51	0.13	0.85
M-09-099	177.00	257.00	80.00	0.52	0.13	0.84
M-07-034	190.00	246.00	56.00	0.48	0.14	0.84
M-09-106	222.00	274.00	52.00	0.50	0.13	0.83
M-08-070	251.00	320.00	69.00	0.39	0.17	0.83
M-10-119	241.00	344.00	103.00	0.63	0.08	0.83
M-10-116	265.00	323.00	58.00	0.66	0.07	0.83
M-07-044	336.00	402.00	66.00	0.67	0.06	0.83
M-07-060	175.00	263.00	88.00	0.48	0.14	0.83
M-07-043	126.00	232.00	106.00	0.50	0.11	0.78
M-08-061	640.00	690.00	50.00	0.44	0.11	0.72
M-09-108	223.00	273.00	50.00	0.41	0.10	0.67
Average	**106.51**	**291.92**	**185.41**	**0.84**	**0.25**	**1.46**

Table 12-6: Relevant Iron Cap Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
IC-10-033	136.00	199.00	63.00	2.64	0.32	2.31
IC-10-011	326.00	377.00	51.00	1.71	0.23	2.30
IC-10-030	2.60	82.00	79.40	0.63	0.59	2.12
IC-10-009	366.00	418.00	52.00	1.39	0.21	1.93
IC-10-029	152.00	251.00	99.00	1.01	0.37	1.91
IC-10-006	2.92	76.20	73.28	0.83	0.44	1.89
IC-10-040	4.20	165.50	161.30	0.89	0.37	1.85
IC-10-028	467.00	584.00	117.00	1.03	0.29	1.77
IC-10-032	18.50	111.00	92.50	0.71	0.43	1.76
IC-10-025	100.00	196.00	96.00	1.04	0.23	1.62
IC-10-010	193.00	300.00	107.00	0.79	0.32	1.60
IC-05-01	3.30	91.30	88.00	0.88	0.26	1.48
IC-10-017	390.70	491.40	100.70	0.79	0.26	1.45
IC-10-008	107.00	215.00	108.00	0.78	0.26	1.43
IC-10-011	2.90	53.00	50.10	0.44	0.39	1.43
IC-10-033	3.40	67.00	63.60	0.31	0.43	1.41
IC-10-029	1.50	90.30	88.80	0.42	0.39	1.39
IC-10-027	189.00	260.50	71.50	0.55	0.33	1.39
IC-10-037	8.50	111.90	103.40	0.90	0.19	1.39
IC-10-035	2.60	103.00	100.40	0.70	0.26	1.36
IC-10-033	75.00	132.00	57.00	0.41	0.37	1.35
IC-10-024	143.00	194.00	51.00	0.35	0.41	1.35
IC-10-034	11.07	75.50	64.43	0.57	0.30	1.35
IC-10-031	117.00	275.00	158.00	0.84	0.20	1.34
IC-10-015	254.00	339.00	85.00	0.31	0.43	1.34
IC-10-007	15.50	66.00	50.50	0.72	0.24	1.33
IC-10-035	105.00	351.00	246.00	0.74	0.25	1.31
IC-10-016	224.40	278.20	53.80	0.74	0.22	1.29
IC-10-045	126.00	184.00	58.00	0.26	0.41	1.27
IC-10-025	1.35	63.00	61.65	0.41	0.33	1.24
IC-10-017	173.00	224.40	51.40	0.23	0.39	1.22
IC-10-039	277.70	450.00	172.30	0.31	0.35	1.19
IC-05-02	146.90	238.00	91.10	0.62	0.21	1.15
IC-05-03	1.50	54.60	53.10	0.40	0.29	1.13
IC-10-029	287.00	354.00	67.00	0.67	0.18	1.12
IC-10-032	159.00	224.00	65.00	0.40	0.28	1.11
IC-10-009	203.00	275.00	72.00	0.73	0.15	1.11
IC-10-037	228.50	280.00	51.50	0.53	0.23	1.11
IC-10-008	38.00	97.80	59.80	0.35	0.29	1.09
IC-10-031	345.80	426.00	80.20	0.45	0.24	1.06
IC-05-04	113.00	232.00	119.00	0.41	0.25	1.02
IC-10-013	326.00	378.00	52.00	0.23	0.29	0.96
IC-05-01	157.30	215.30	58.00	0.39	0.22	0.95
IC-05-02	74.90	138.90	64.00	0.40	0.21	0.93
IC-10-023	8.15	72.00	63.85	0.37	0.21	0.91
IC-10-016	50.00	106.00	56.00	0.36	0.21	0.90
S80-14	84.00	138.00	54.00	0.35	0.22	0.89
IC-10-019	2.80	54.00	51.20	0.34	0.21	0.87
IC-10-015	407.00	471.30	64.30	0.08	0.30	0.85
IC-10-026	46.00	106.00	60.00	0.27	0.22	0.82
IC-10-023	188.00	250.00	62.00	0.17	0.25	0.82
IC-05-01	105.30	155.30	50.00	0.33	0.16	0.75
Average	**134.08**	**214.26**	**80.18**	**0.65**	**0.29**	**1.36**

The relevant composited data shown in Tables 12-3 through 12-6 were sorted by decreasing gold equivalent grade (AuEQV). The average depths, average continuous mineralized lengths, and average gold/copper/gold equivalent grades are shown at the bottom of the table (right hand side).

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section describes Seabridge's sample security, sample preparation, and analytical methods that were used in 2010 for their KSM project. These are essentially the same methods that have been described in previous RMI NI 43-101 reports dealing with the KSM project (Lechner, 2007, Lechner 2008, Lechner 2009, and Lechner 2010).

13.1 Statement on Sample Preparation Personnel

All initial sample preparation (sawing and bagging) was conducted by laborers contracted from Tahltan Native Development Corporation, trained by and under the direct supervision of geologists employed by Seabridge Gold. Drill core and quality control samples were shipped to Eco Tech's prep facility located in Stewart, B.C. and then shipped by Eco Tech to their assay laboratory located in Kamloops, B.C. where the prepped samples were analyzed.

13.2 Sample Preparation and Dispatch

Upon completion of logging and sample demarcation, the core boxes were moved to the core cutting facilities in camp, usually the following day. The core cutting building is a 14'x16' plywood platform, covered with a poly tarp on aluminum poles. The walls were left open to facilitate air circulation and prevent dust contamination. Three gasoline engine powered saws with 14" diamond impregnated blades designed for rock cutting were utilized, on day shifts only. The saws were mounted on secure wooden stands at waist height. The saw blades were cooled, cleaned, and lubricated with fresh, non-recirculated water during cutting. The saw operator placed uncut core boxes on tables adjacent to the saws, and cut each piece of core sequentially within each marked sample interval. The assay half of the sample was placed in a heavy duty polythene bag and the other half was returned to the core box. Once a sample interval was completely sawn, the corresponding sample tag number was stapled to the inside at the top of the bag, and the bag was secured with staples. The sample number was also written on the bag with a permanent felt tip marker.

The bags were placed sequentially in rows on pallets or on the floor. Upon completion of a batch of 33 (see below), the samples were placed into large polyweave (rice) shipping bags, six per bag (three for the larger HQ core). The polyweave bag was labeled with the project number, sample numbers, shipment number, and lab address, and then secured with plastic tie straps. In addition, for security purposes the polyweave bag was also secured with a uniquely numbered tie strap, and the number recorded on the retained copy of the sample transmittal form. The other copy of the sample transmittal form was placed in the last shipping bag of each batch. The bags were stored adjacent to the core cutting building or helicopter pad until a complete shipment was ready, which usually included several batches. During normal production and good weather, shipments were sent out at least every two days.

The sample shipment was placed inside the project chartered helicopter and flown directly to the Granduc Road staging area and unloaded by the pilot. At the staging area the shipment was either stored and locked inside a metal bulk shipping container or transferred directly to a waiting truck. Trucking was contracted to Granmac Services Ltd. of Stewart. The shipment was driven to Stewart where the samples were unloaded at the sample preparation facilities by Eco Tech Laboratories personnel. Occasionally the samples were taken directly to Stewart via helicopter and transferred to the prep lab by truck contracted by Granmac. The prep lab took an inventory of the shipment and confirmed that the numbered tie strap was not broken or tampered with. Eco Tech then sent notification of the receipt of shipment with tie strap and sample numbers to Seabridge personnel at camp who confirmed the sample shipment.

13.3 **Analytical Procedures**

At the Eco Tech facilities in Stewart, samples were sorted and dried (if necessary), crushed through a jaw crusher and cone or roll crusher to –10 mesh, then split through a Jones riffle until a –250 gram sub sample was achieved. The sub sample was pulverized in a ring and puck pulverizer so that 95% of the material passed a -140 mesh screen, then rolled to homogenize. The resulting pulp sample was placed in a numbered paper envelope and securely packed in cardboard boxes. These boxes were shipped via Greyhound freight services to the Eco Tech Laboratory facilities located in Kamloops, B.C.

At the Eco Tech's lab in Kamloops, a 30 gram sample size was split out from the pulp envelope and then fire assayed using appropriate fluxes. The resultant doré bead was parted and then digested with aqua regia followed by an atomic absorption (AA) finish using a Perkin Elmer AA instrument. The lower limit of detection for gold is 0.03 g/t or 0.001 oz/t. For other metals, a multi-element ICP analysis was completed. For this procedure, a 0.5 gram sample was digested with 3 ml of a 3:1:2 (HCl: HN03:H20) which contains beryllium, which acts as an internal standard for 90 minutes in a water bath at 95°C. The sample was then diluted with 10 ml of water and analyzed on a Jarrell Ash ICP unit. Eco Tech's ICP detection limits (lower and upper) are summarized in Table 13-1.

Assay results were then collated by computer and were printed along with accompanying internal quality control data (repeats and standards). Results were printed on a laser printer and were faxed and/or mailed to appropriate Seabridge personnel. Appropriate standards and repeat samples were included on the data sheet.

Table 13-1: ICP Detection Limits

Element	Lower	Upper	Element	Lower	Upper
Ag	0.2 ppm	0.0 ppm	Mo	1 ppm	10,000 ppm
Al	0.01%	10.00%	Na	0.01%	10.00%
As	5 ppm	10,000 ppm	Ni	1 ppm	10,000 ppm
Ba	5 ppm	10,000 ppm	P	10 ppm	10,000 ppm
Bi	5 ppm	10,000 ppm	Pb	2 ppm	10,000 ppm
Ca	0.01%	10.00%	Sb	5 ppm	10,000 ppm
Cd	1 ppm	10,000 ppm	Sn	20 ppm	10,000 ppm
Co	1 ppm	10,000 ppm	Sr	1 ppm	10,000 ppm
Cr	1 ppm	10,000 ppm	Ti	0.01%	10.00%
Cu	1 ppm	10,000 ppm	U	10 ppm	10,000 ppm
Fe	0.01%	10.00%	V	1 ppm	10,000 ppm
La	10 ppm	10,000 ppm	Y	1 ppm	10,000 ppm
Mg	0.01%	10.00%	Zn	1 ppm	10,000 ppm
Mn	1 ppm	10,000 ppm			

13.4 **Quality Control Measures**

Seabridge implemented the same quality control procedures that they used for their previous KSM programs. Various standard reference material (SRM) sources have been used since 2006. They included blanks of material obtained from commercial landscaping materials (crushed marble and granite) and "barren" river gravels collected near Stewart B.C., along with different commercially certified standards of prepackaged pulps. Assay quality control measures included the insertion of a sample blank and pulp standard within each laboratory batch of approximately 35 samples. Thus a complete batch contained a minimum of one blank and one pulp standard, with the remainder being core samples. The blank and pulp standard were numbered using the same number sequence that was used for the core samples and inserted into each batch shipment randomly by the geologist during the logging process.

Three different blanks were used in 2010. Blank 3 is one that was used for the past several years and was prepared from local barren river gravels which were screened to remove fines and oversize to produce nominal 1" diameter pieces. Blank 4 consisted of crushed white limestone/marble used for landscaping. Blank 5 was also landscaping material consisting of crushed gray quartzite. Blanks 4 and 5 were purchased in 20 kg bags from a home and garden retailer located in Terrace, B.C. Blanks were submitted into the 2010 sample stream at a frequency of about 1 blank for every 33 samples. Approximately 378 barren samples or "blanks" were submitted to Eco Tech. Figures 13-1 and 13-2 chart the performance of the gold and copper blanks for the 2010 drilling campaign.

Figure 13-1: 2010 Au Blank Performance

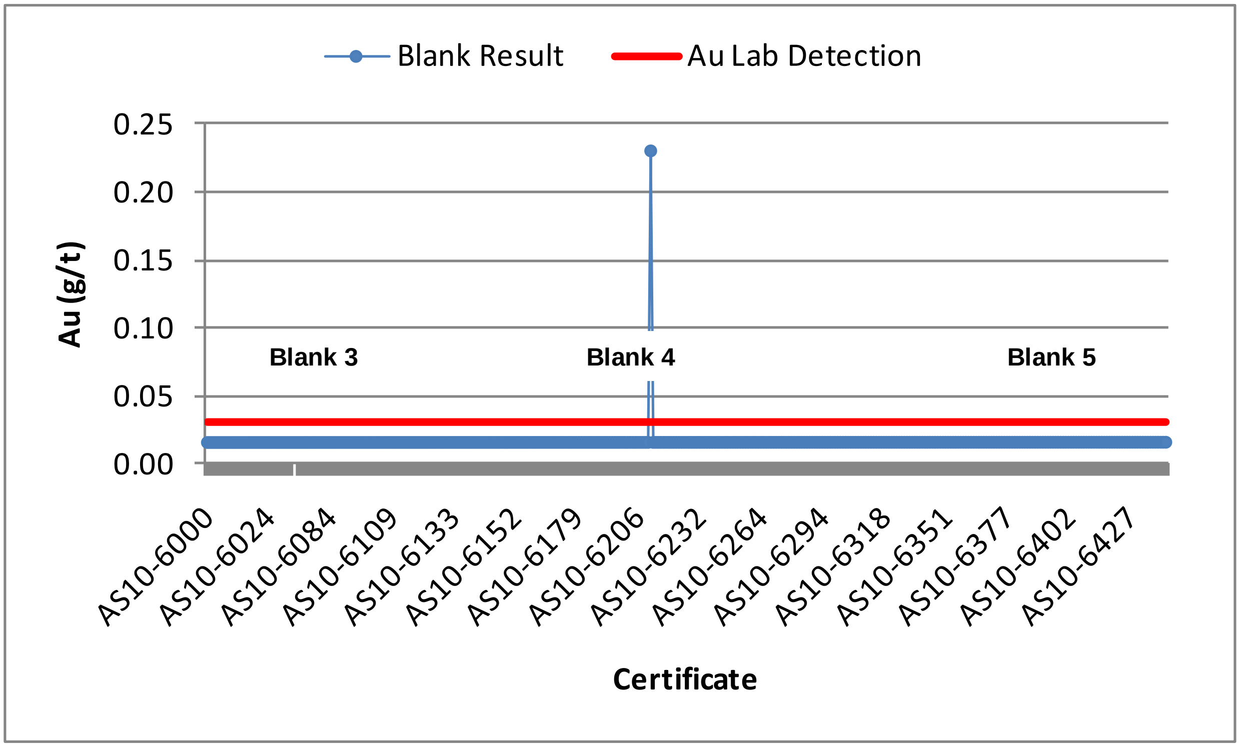
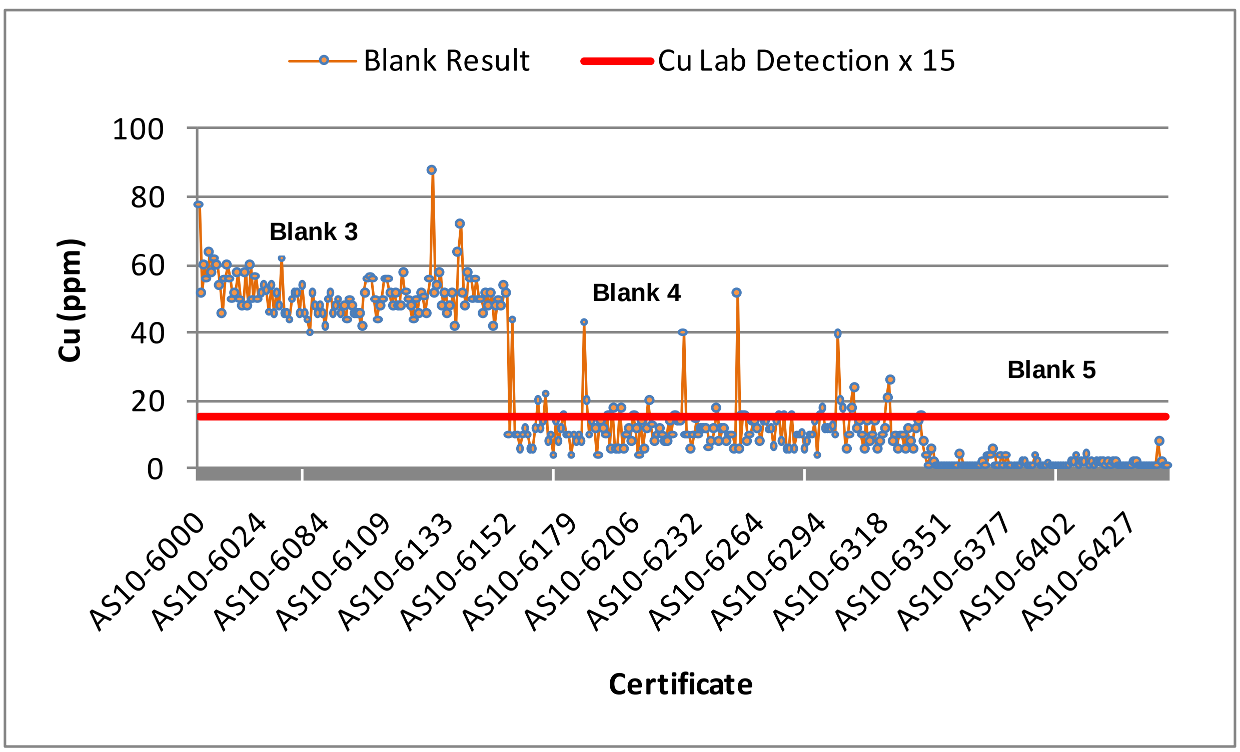


Figure 13-2: 2010 Cu Blank Performance



There was one notable failure with the gold blank as can be seen in Figure 13-1. The "background" copper values for the three blanks are readily apparent from the distinct populations of copper grades for Blank 3, Blank 4, and Blank 5 as shown in Figure 13-2. It appears that there were five instances where Blank 4 may have been mislabeled by logging personnel and those samples were actual Blank 3 material.

Five of the six pulp standards that were used by Seabridge for their 2010 drilling/sampling campaign were purchased from CDN Resource Laboratories Ltd. (CDN) out of Delta, B.C. Six CDN standards (CDN-CGS-19, CDN-CM-4, CDN-CM-5, CDN-CM-6,and CDN-CM-7) were prepared from material that was collected from the Casino copper-gold-molybdenum porphyry property located in Yukon Territory. The sixth standard (SEA-1) was prepared from a bulk sample of core collected from the Mitchell zone that had been used for crushing tests. This standard was prepared by CDN Resource Laboratories Ltd., and analyzed in round robin fashion by seven laboratories. The Certificate of Analysis was prepared by Smee and Associates Consulting Ltd. of North Vancouver in May, 2010. These standards have certified gold and copper values that are definitely relative to the type and tenor of mineralization that has been identified at the Mitchell deposit. A total of 377 SRM's were inserted into the 2010 sample stream or a frequency of about one SRM for every 33 samples or 3% of the total assay samples. Table 13-2 summarizes the SRM's that were used by Seabridge for their 2010 drilling campaign. The table shows the number of SRM's that were submitted, their expected values along with ±2 standard deviation units.

Table 13-2: 2010 KSM Standard Reference Materials

Standard	Number Submitted	Gold Values (g/t)			Copper Values (%)			Molybdenum Values (%)		
		Expected	-2 Std Dev	+2 Std Dev	Expected	-2 Std Dev	+2 Std Dev	Expected	-2 Std Dev	+2 Std Dev
CGS-19	64	0.74	0.67	0.81	0.132	0.122	0.142	n/a	n/a	n/a
CM-4	47	1.18	1.06	1.30	0.508	0.483	0.533	0.032	0.028	0.036
CM-5	53	0.29	0.25	0.34	0.319	0.299	0.339	0.050	0.045	0.055
CM-6	44	1.43	1.34	1.52	0.737	0.698	0.776	0.083	0.075	0.091
CM-7	45	0.43	0.39	0.47	0.445	0.418	0.472	0.027	0.025	0.029
SEA-1	124	0.77	0.71	0.84	0.204	0.194	0.214	0.007	0.006	0.008
Total	377	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a

The performance of the various gold, copper, and molybdenum standards are graphed as a function of time (certificate number) in Figures 13-3 through 13-19.

Figure 13-3: 2010 Au Standard CGS-19 Performance

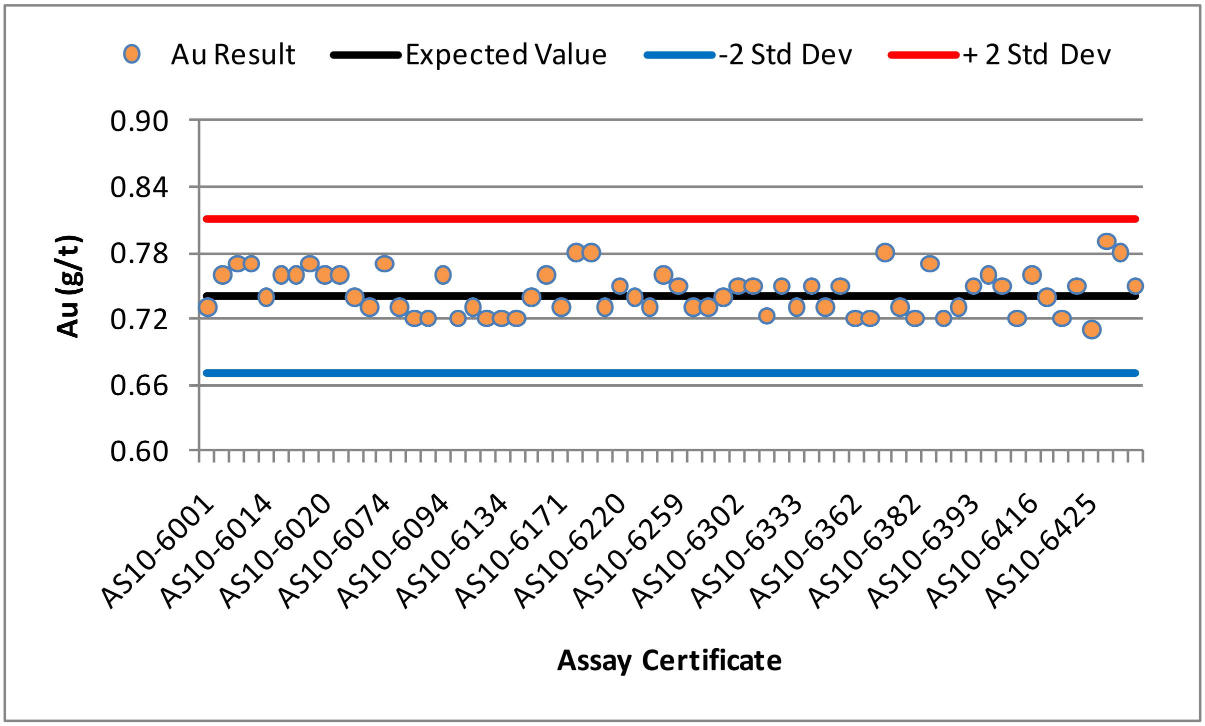


Figure 13-4: 2010 Cu Standard CGS-19 Performance

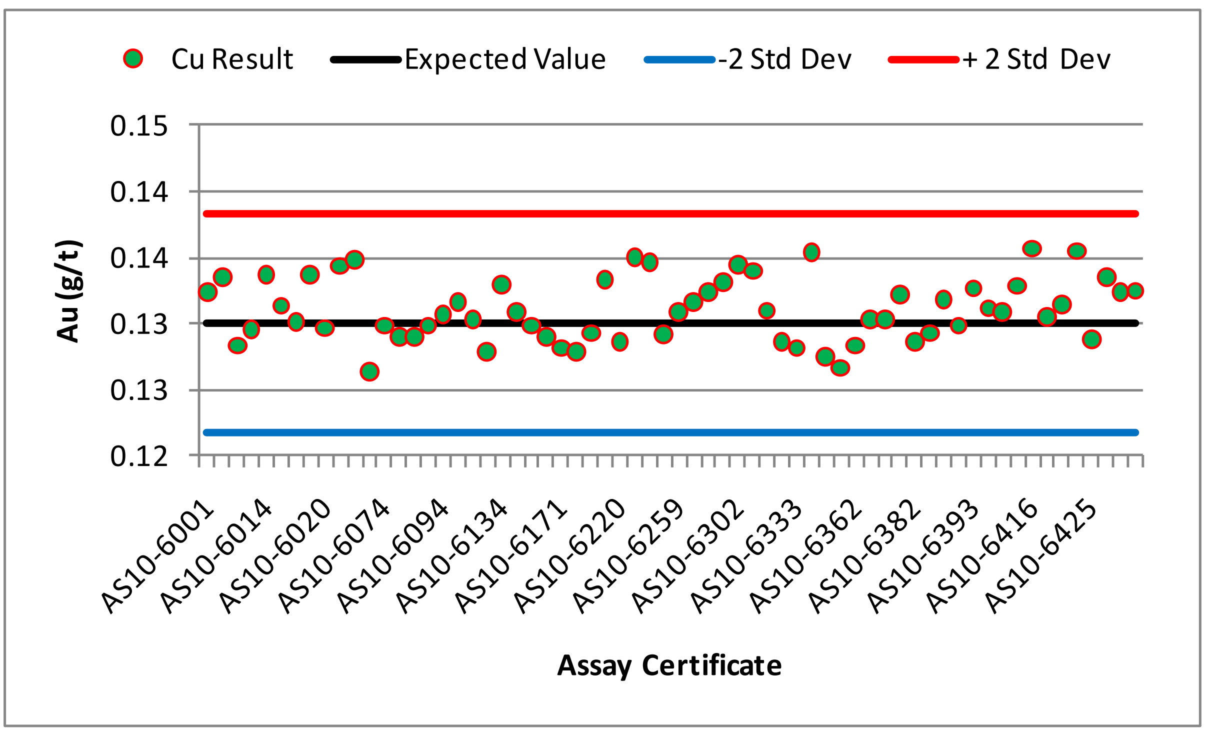


Figure 13-5: 2010 Au Standard CM-4 Performance

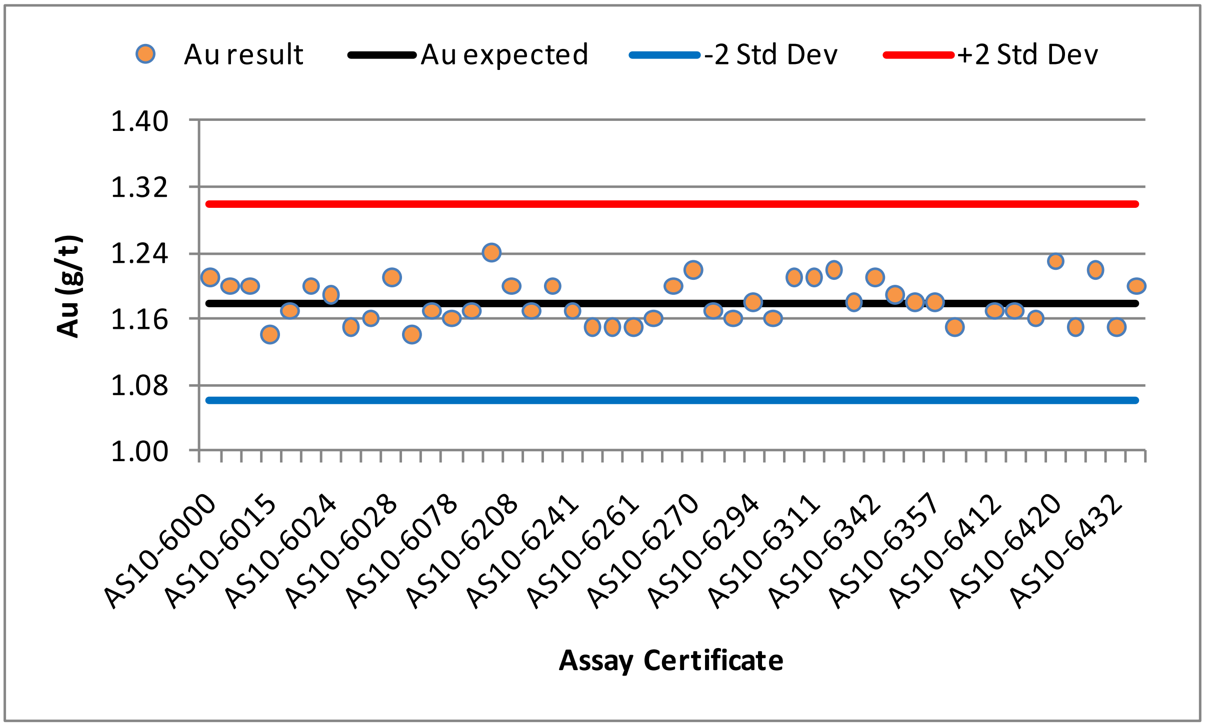


Figure 13-6: 2010 Cu Standard CM-4 Performance

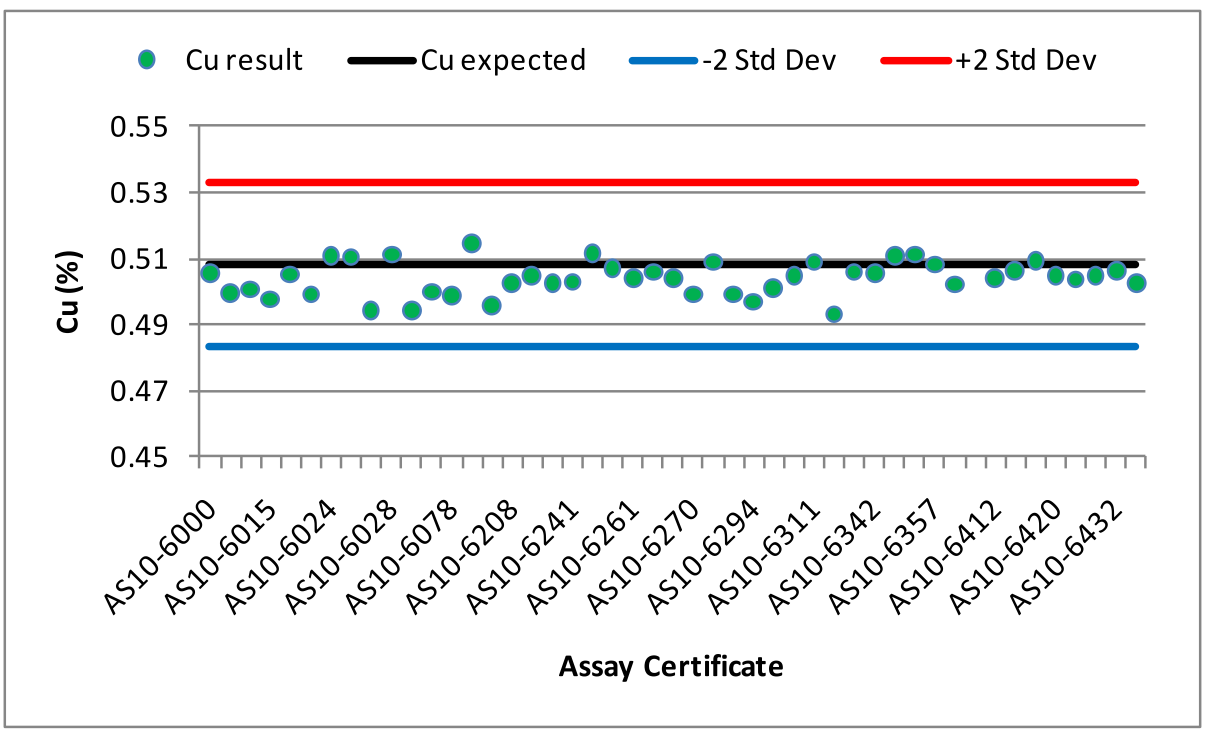


Figure 13-7: 2010 Mo Standard CM-4 Performance

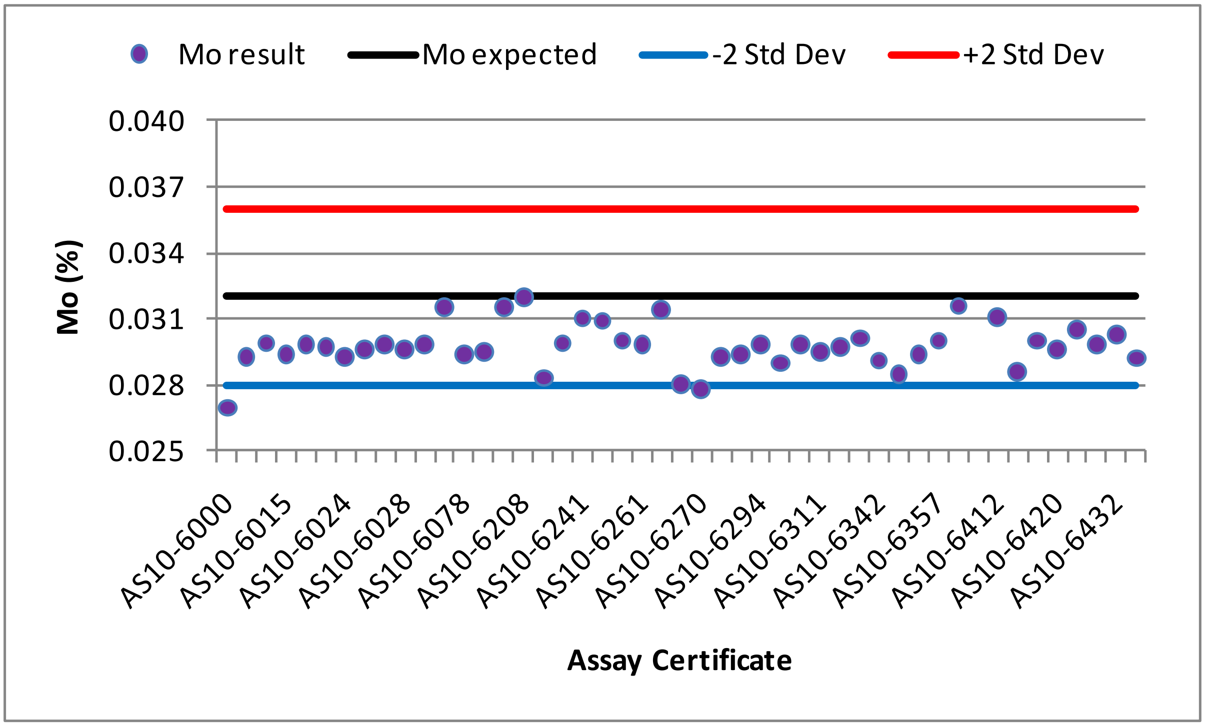


Figure 13-8: 2010 Au Standard CM-5 Performance

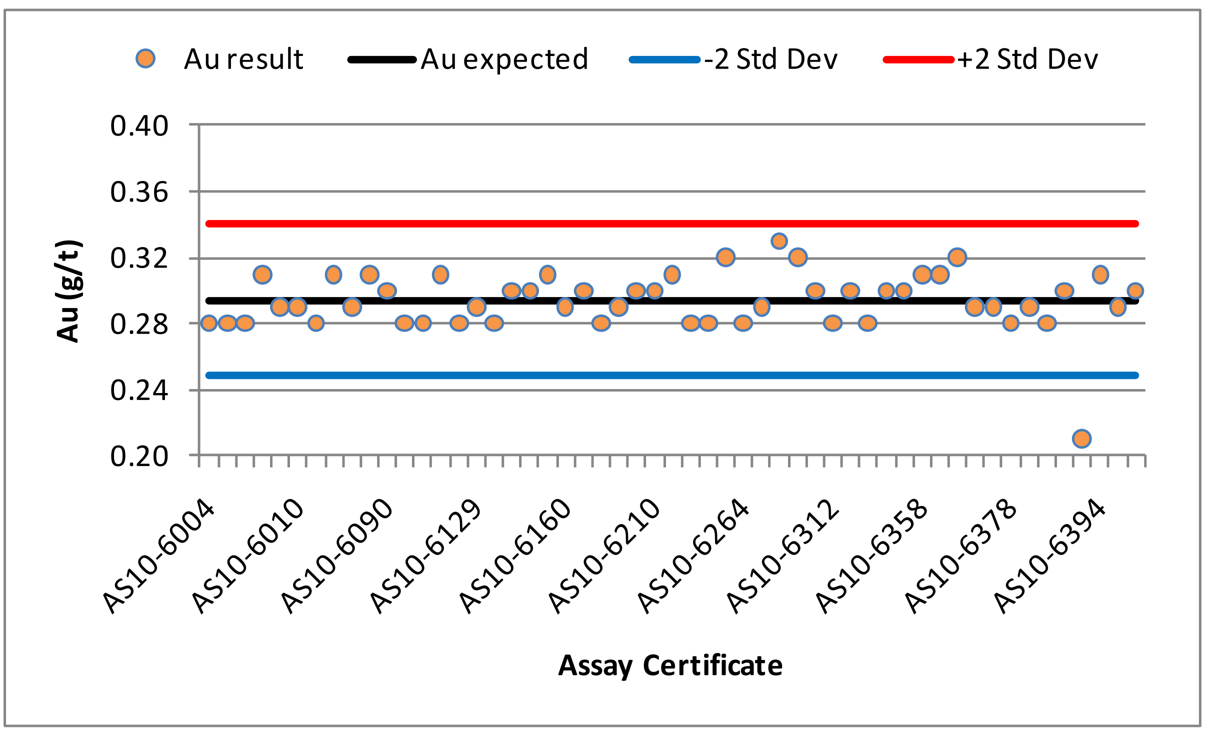


Figure 13-9: 2010 Cu Standard CM-5 Performance

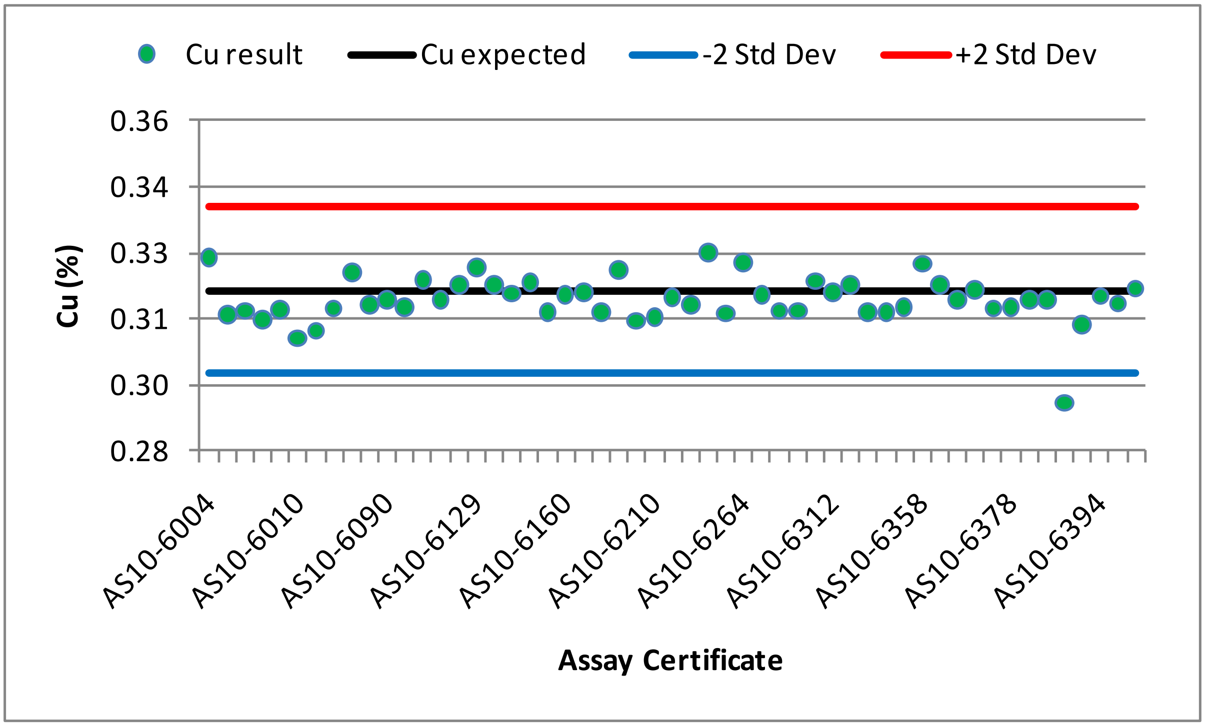


Figure 13-10: 2010 Mo Standard CM-5 Performance

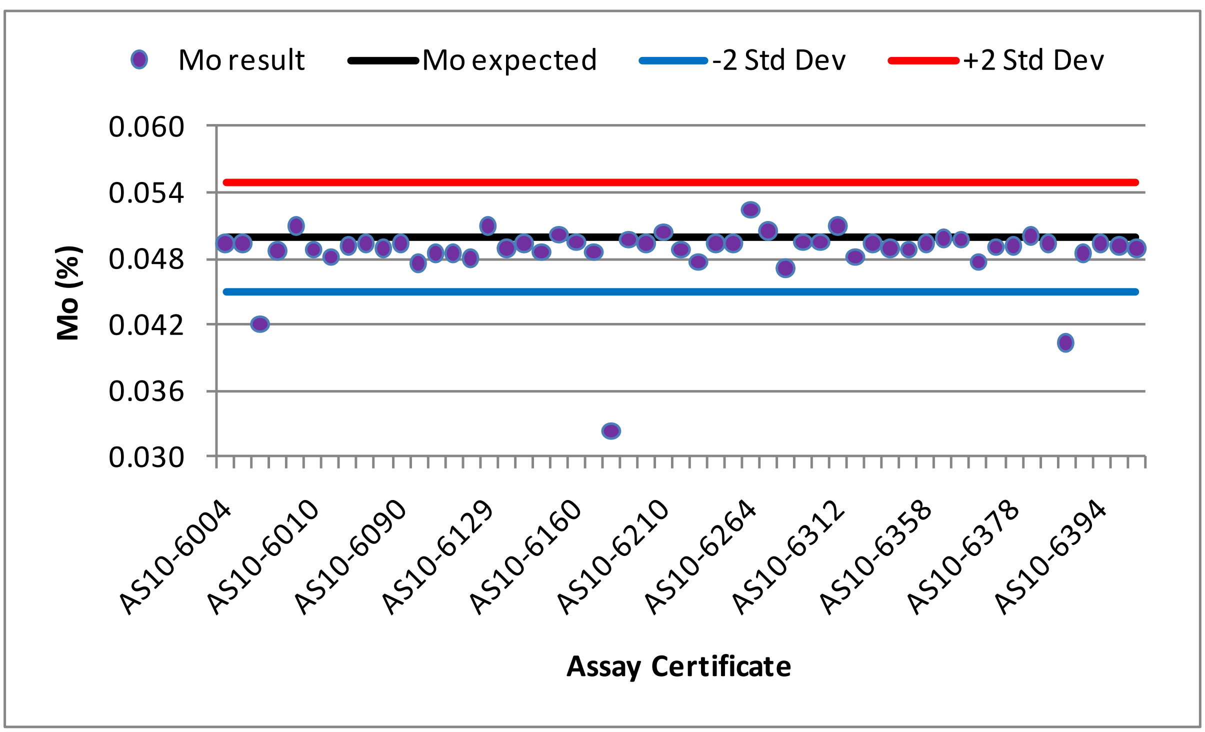


Figure 13-11: 2010 Au Standard CM-6 Performance

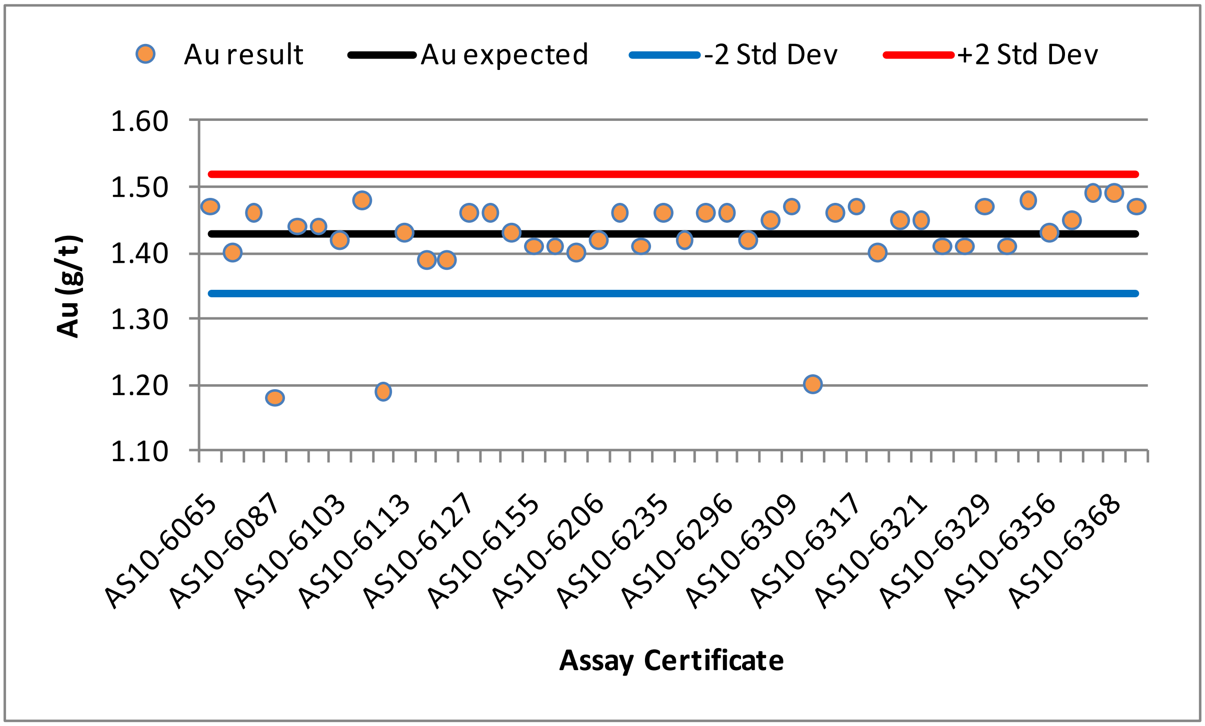


Figure 13-12: 2010 Cu Standard CM-6 Performance

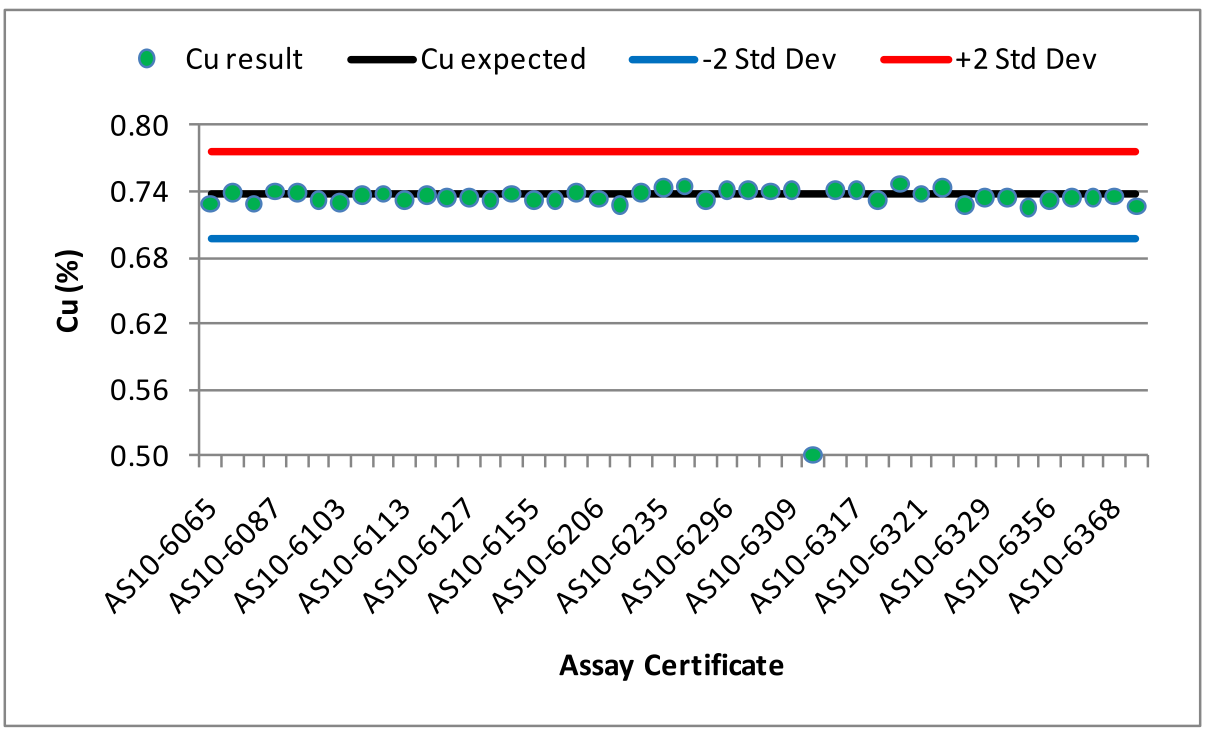


Figure 13-13: 2010 Mo Standard CM-6 Performance

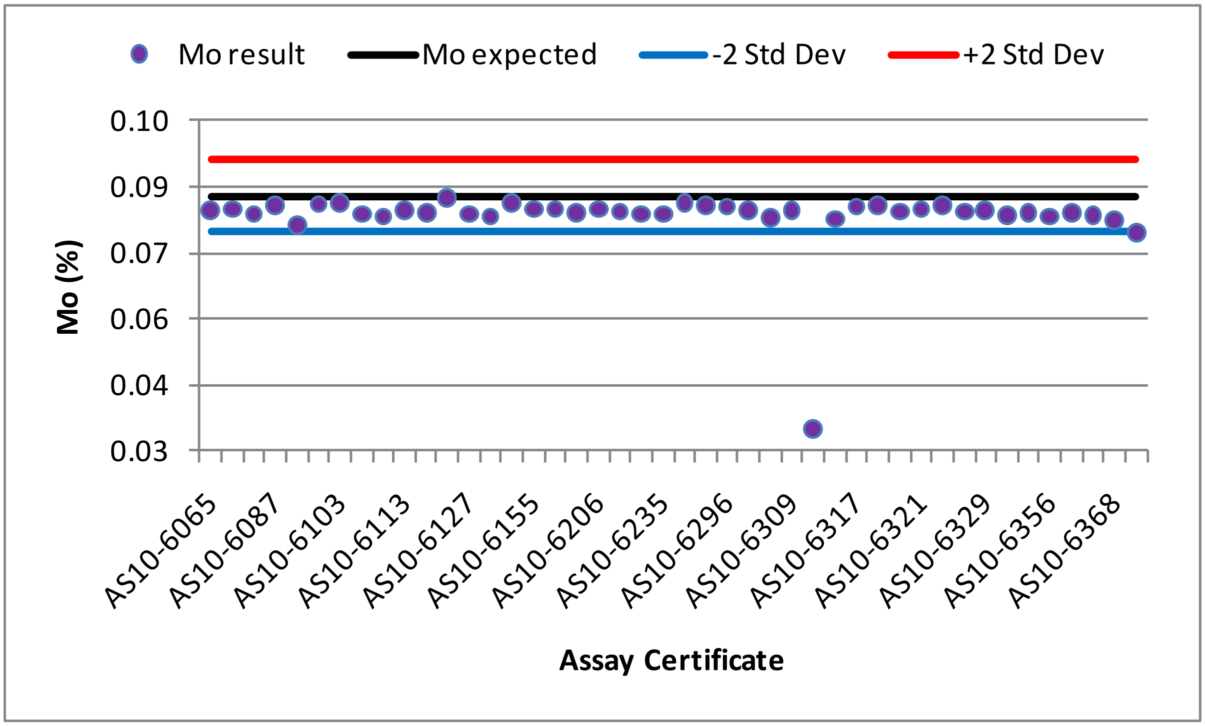


Figure 13-14: 2010 Au Standard CM-7 Performance

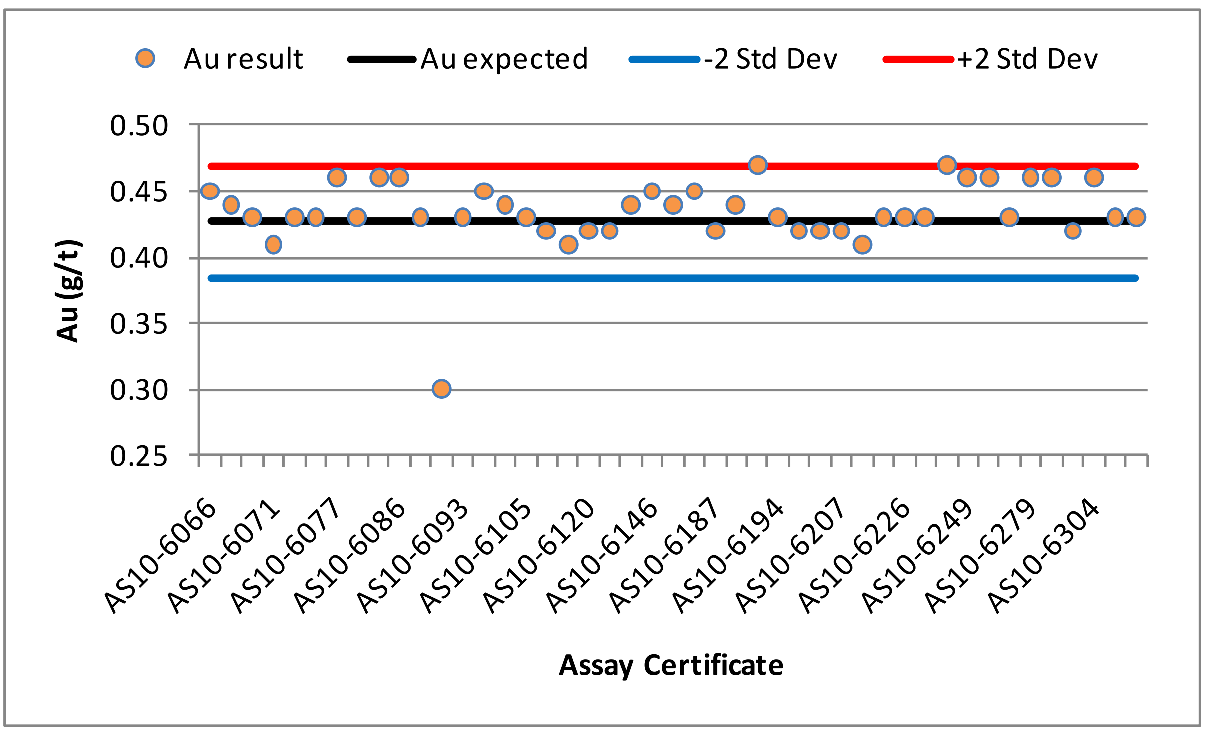


Figure 13-15: 2010 Cu Standard CM-7 Performance

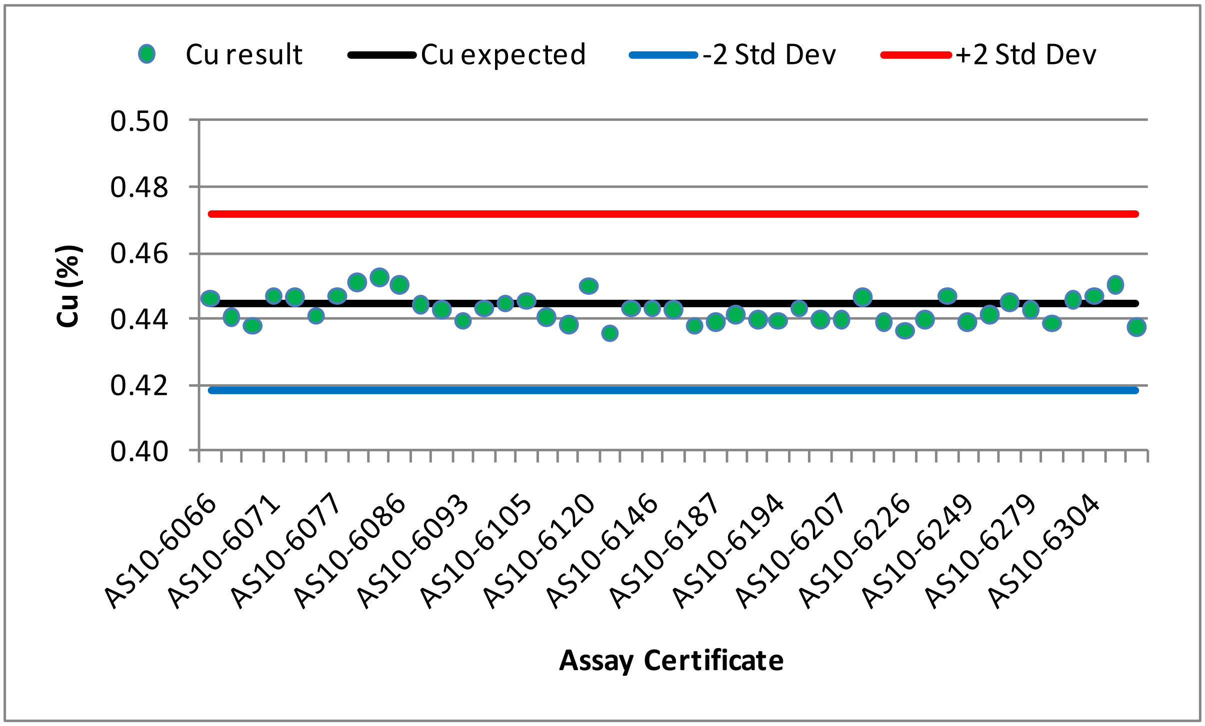


Figure 13-16: 2010 Mo Standard CM-7 Performance

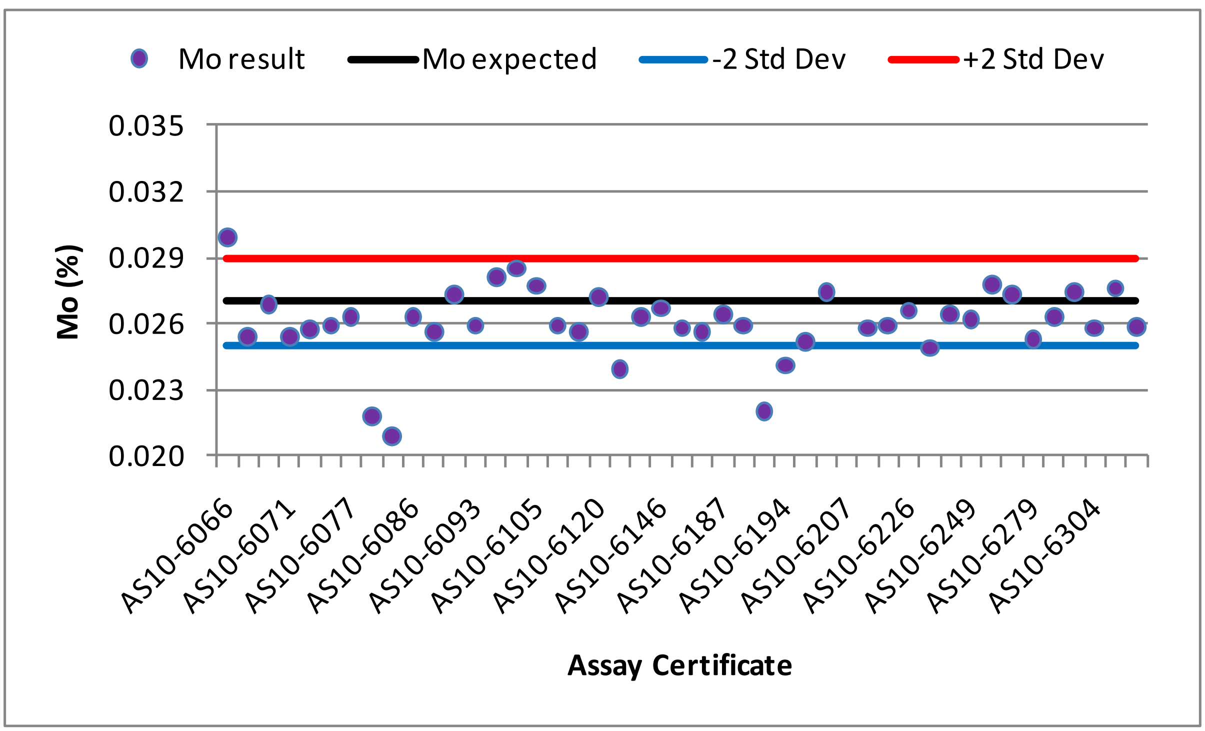


Figure 13-17: 2010 Au Standard SEA-1 Performance

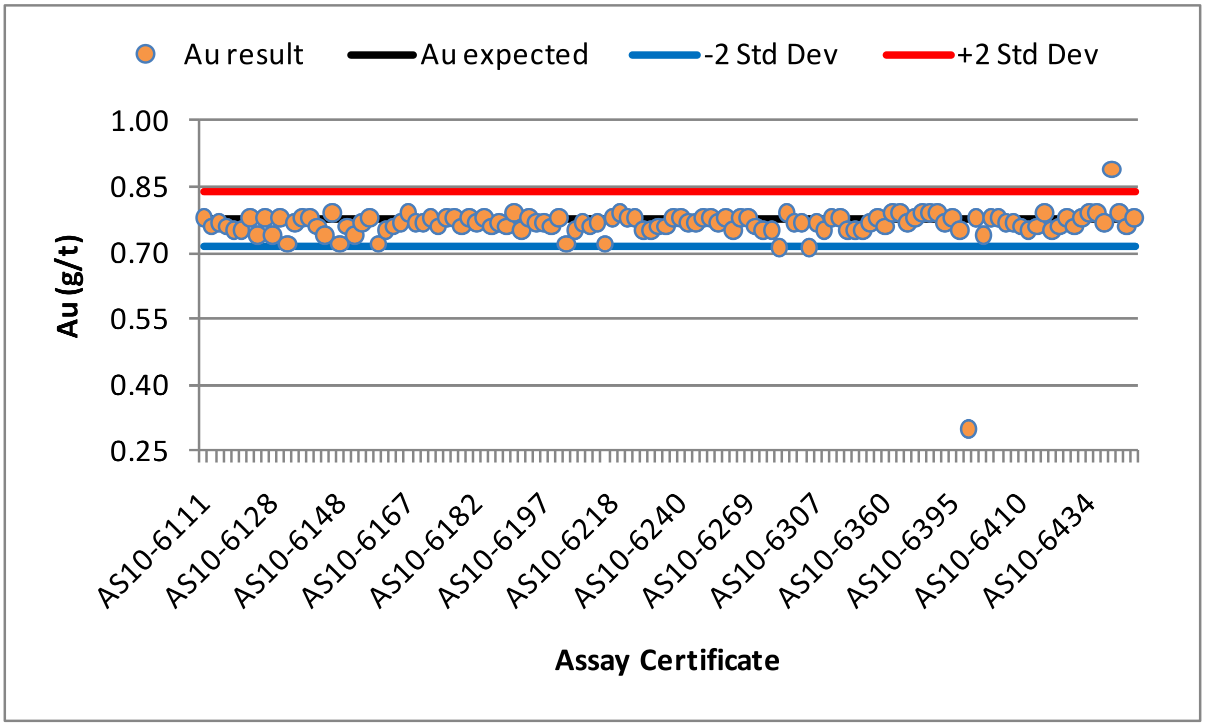


Figure 13-18: 2010 Cu Standard SEA-1 Performance

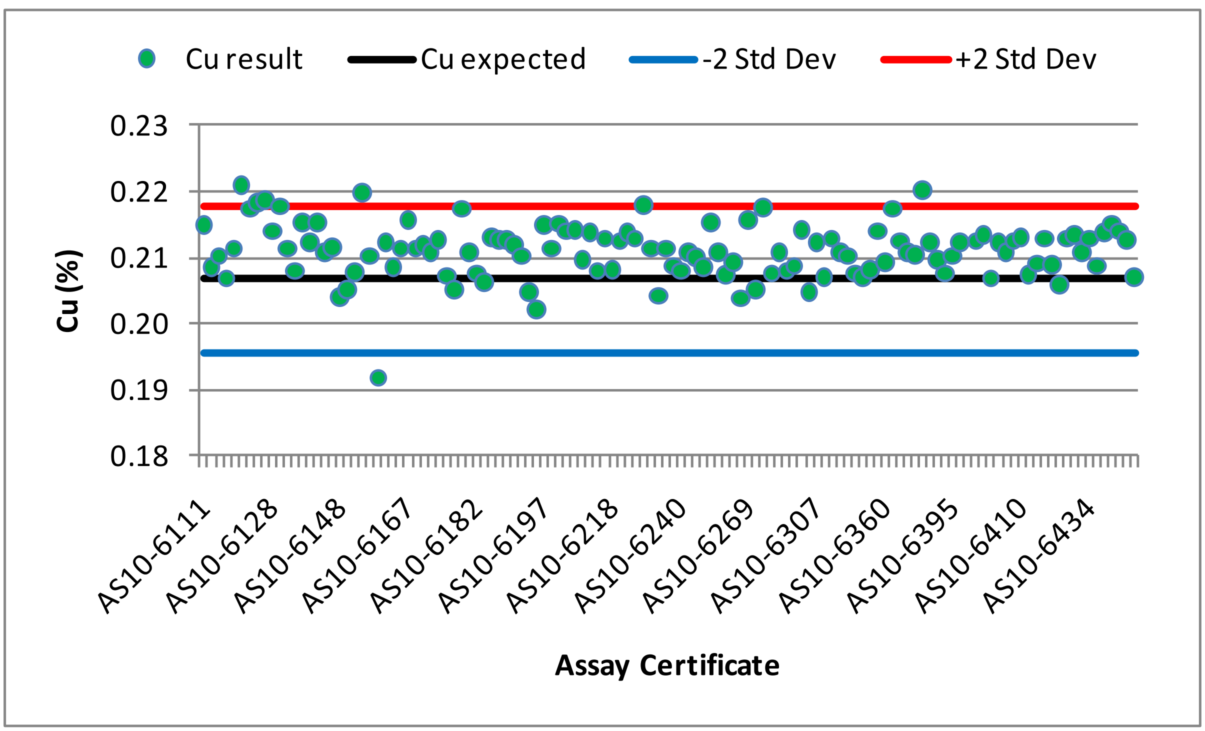
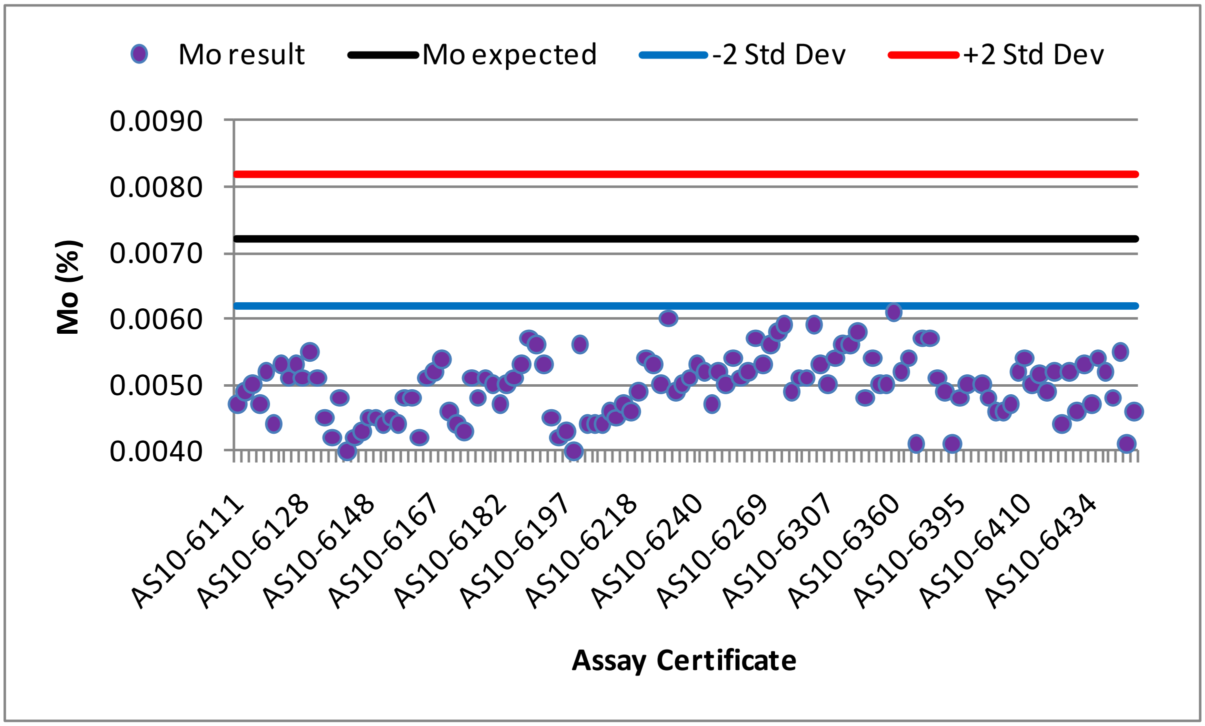


Figure 13-19: 2010 Mo Standard SEA-1 Performance



In general, most of the SRM results track well within ± 1 standard deviation of the expected value. One exception is low grade molybdenum standards (Figures 13-7 and 13-19) which routinely came back lower than the expected value. This is particularly evident in Figure 13-19 shown above. Higher grade molybdenum standards (Figures 13-10, 13-13, and 13-16) assayed by Eco Tech were within reasonable tolerances. In RMI's opinion the poor performance of the lower grade molybdenum standards is not a material issue.

In addition to the insertion of control samples with each batch, Seabridge also submitted duplicate core samples in every second batch by sawing one half of the drill core into two ¼ core splits that were submitted as individual samples to Eco Tech. 179 core duplicates or about 1.3% of the total samples were submitted to Eco Tech in 2010. Table 13-3 summarizes the basic descriptive statistic for the "original" and "duplicate" ¼ core samples for Au, Ag, Cu, and Mo.

Table 13-3: Summary of 2010 ¼ Core Assay Results

Parameter	Au (g/t)		Ag (g/t)		Cu (%)		Mo (%)	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
Count	179	179	179	179	179	179	179	179
Min	0.015	0.015	0.100	0.100	0.0020	0.0022	0.000	0.000
Max	5.100	7.400	72.500	122.000	1.0600	1.4800	0.072	0.062
Mean	0.363	0.380	3.157	3.472	0.1508	0.1485	0.003	0.003
Median	0.200	0.200	1.500	1.300	0.1066	0.1132	0.001	0.001
1st Quartile	0.115	0.110	0.547	0.505	0.0476	0.0454	0.000	0.000
3rd Quartile	0.435	0.420	3.000	2.950	0.2019	0.1932	0.002	0.002
Std Dev	0.539	0.668	7.514	10.974	0.1535	0.1620	0.007	0.008
CV	1.486	1.757	2.380	3.161	1.0182	1.0910	2.421	2.356

As can be seen in Table 13-3, there is a relatively close comparison in the distribution of original and duplicate ¼ core grades. RMI notes that the duplicate Au and Ag sample grades are about 5% and 10% higher than the original ¼ core sample, respectively while the Cu duplicate is about 1.5% lower than the original. The ¼ core original (X-axis) and duplicate (Y-axis) sample grades are compared as quantile-quantile plots in Figures 13-20 through 13-23 for gold, copper, silver, and molybdenum, respectively.

Figure 13-20: 2010 ¼ Core Au QQ Plot

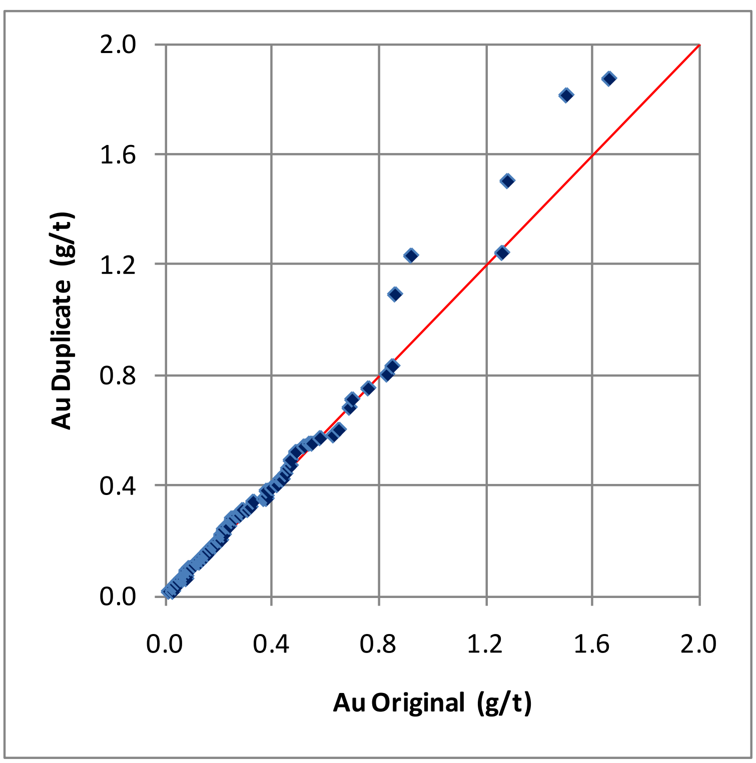


Figure 13-21 2010 ¼ Core Cu QQ Plot

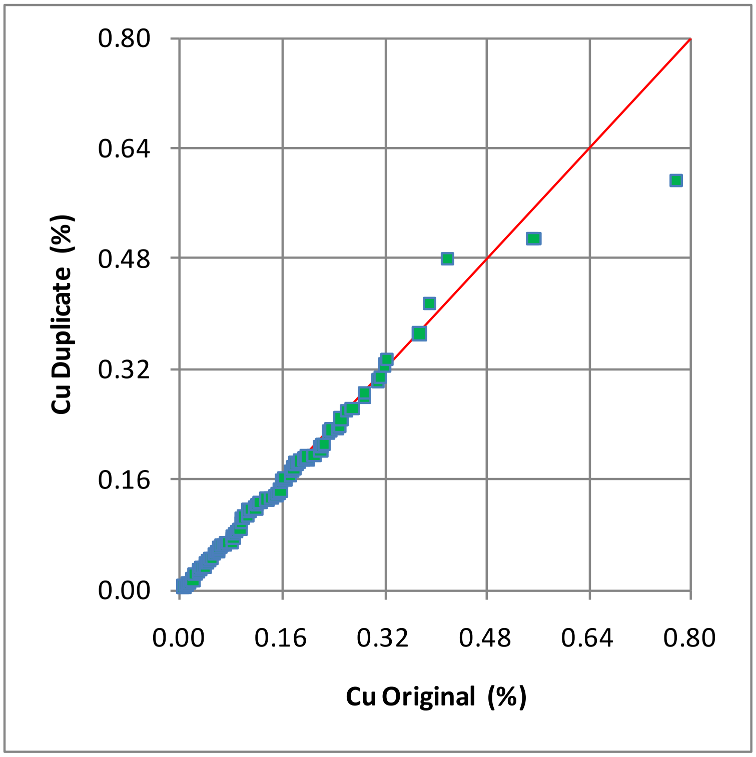


Figure 13-22 2010 ¼ Core Ag QQ Plot

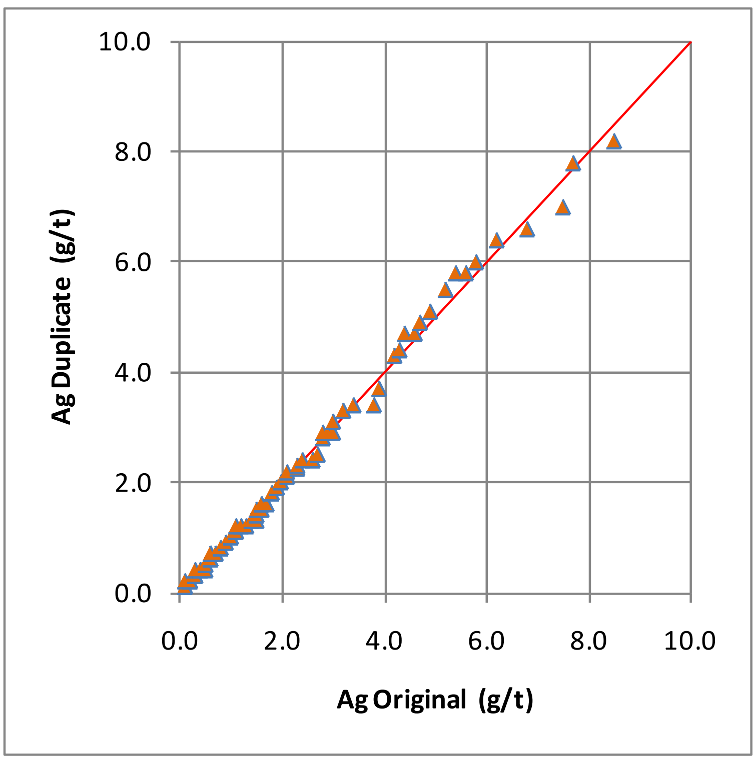
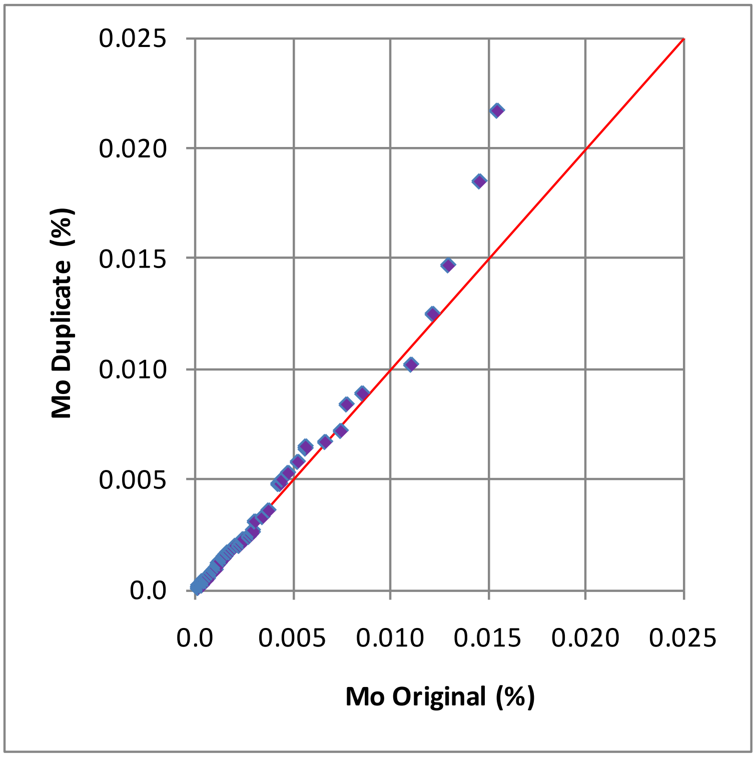


Figure 13-23 2010 ¼ Core Mo QQ Plot



About 11% of the 2010 samples (1,484 samples) that were prepared and assayed by Eco Tech were re-assayed as same pulp "cross-checks" by ALS Chemex of North Vancouver, B.C. Table 13-4 summarizes basic descriptive statistics for the original pulp assay (Eco Tech) and the check assay (Chemex) for gold, silver, copper, and molybdenum. Quantile-quantile plots compare the same pulp gold, copper, silver, and molybdenum results in Figures 13-24 and 13-27, respectively.

Table 13-4: Summary of 2010 Same Pulp Check Assay Results

Parameter	Au (g/t)		Ag (g/t)		Cu (%)		Mo (%)	
	EcoTech	Chemex	EcoTech	Chemex	EcoTech	Chemex	EcoTech	Chemex
Count	1,484	1,484	1,484	1,484	1,484	1,484	1,484	1,484
Min	0.015	0.005	0.100	0.005	0.0001	0.0010	0.000	0.001
Max	15.800	15.250	204.000	205.000	2.0500	2.0400	0.267	0.239
Mean	0.372	0.366	3.578	3.798	0.1548	0.1581	0.009	0.009
Median	0.210	0.200	1.400	2.000	0.1084	0.1130	0.001	0.002
1st Quartile	0.100	0.100	0.600	1.000	0.0436	0.0460	0.000	0.001
3rd Quartile	0.430	0.420	3.400	4.000	0.2065	0.2153	0.004	0.005
Std Dev	0.721	0.707	8.625	8.532	0.1668	0.1680	0.026	0.025
CV	1.941	1.931	2.410	2.246	1.0776	1.0627	2.984	2.707

Figure 13-24: 2010 Eco Tech vs. Chemex Au Check Assays

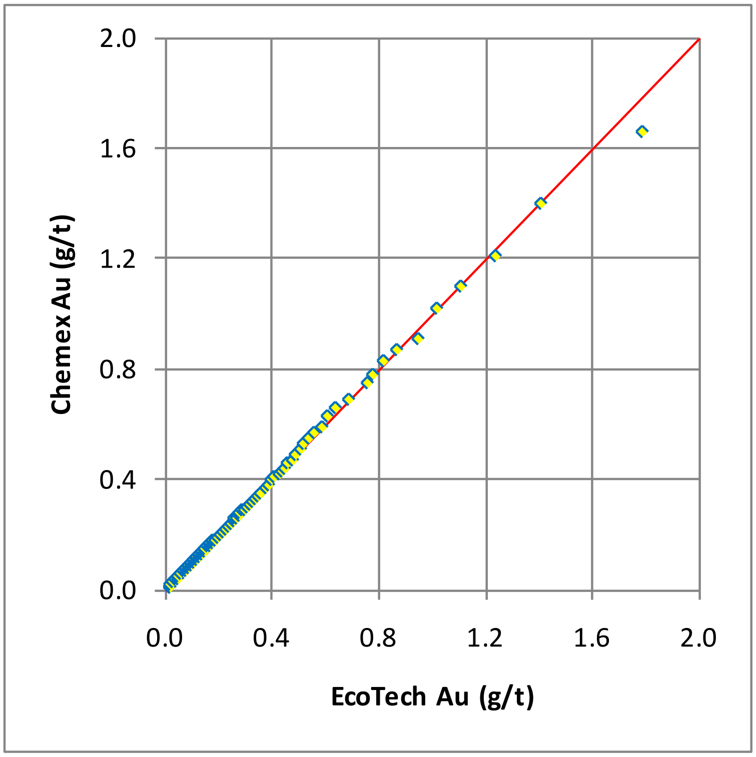


Figure 13-25: 2010 Eco Tech vs. Chemex Cu Check Assays

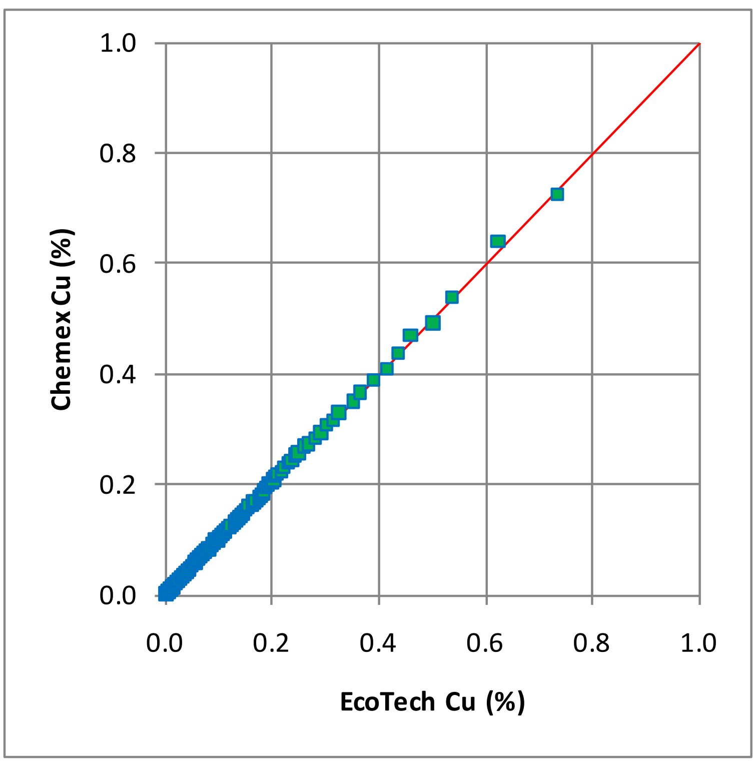


Figure 13-26: 2010 Eco Tech vs. Chemex Ag Check Assays

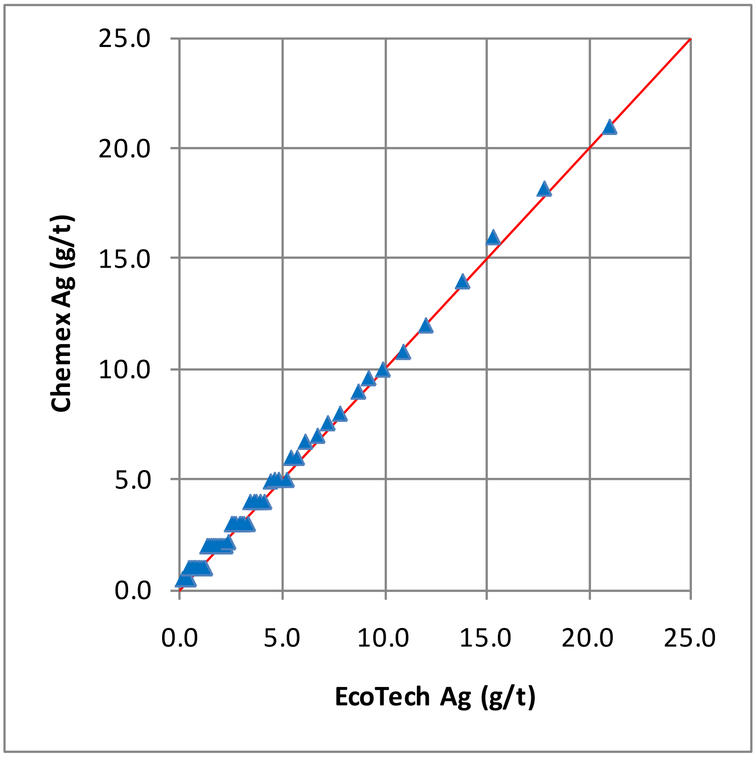
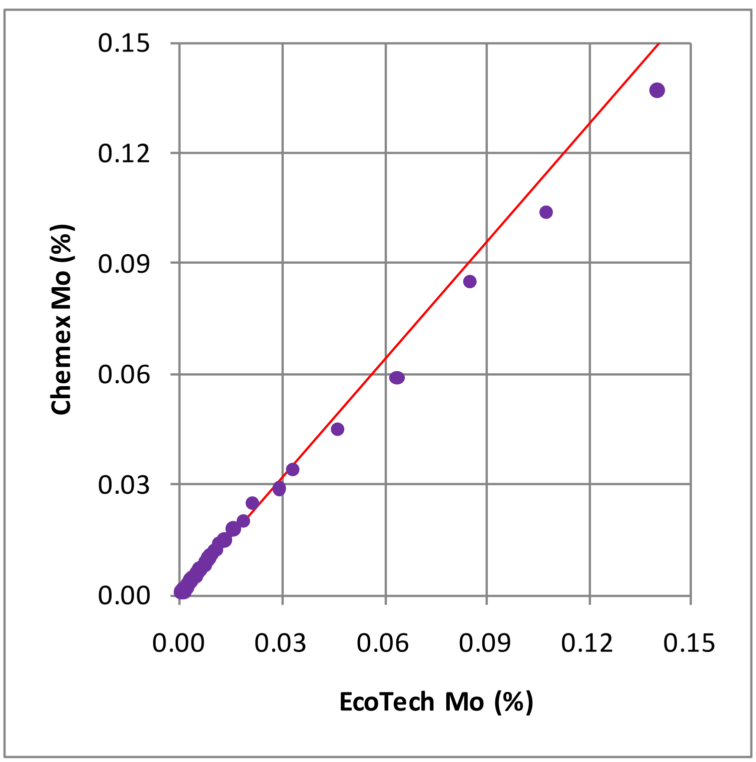


Figure 13-27: 2010 Eco Tech vs. Chemex Mo Check Assays



Both Eco Tech and Chemex employed the same assay measurement techniques for gold. For other metals, the cross-checks compared Eco Tech ICP analyses with ALS ore grade, AAS finish analyses. Both methods utilized a triple acid digestion. For finely disseminated, low grade base metal mineralization similar to that which occurs at the Mitchell deposit, the ICP analyses are generally considered to be as reliable, or more reliable, than ore grade, AAS finish analyses.

13.5 Corrective Action

During the course of the 2010 assaying program there was one blank "failure". Eleven samples associated with this "failure" were re-analyzed. There were ten standard reference material "failures" during the 2010 assaying program. In some cases, the entire batch of samples associated with the "failure" was re-run or in other cases, a partial list of samples associated with the "failure" was re-run by Eco Tech. Four of the ten standard reference "failures" were cases where the wrong standard number was recorded by Seabridge. The Seabridge QA/QC program worked properly in identifying these common errors and appropriate corrective action was taken.

13.6 Author's Opinion

In the opinion of the author, the security, sample preparation, analytical procedures, and QA/QC protocols/results were adequate and that the subsequent assays are suitable to be used to estimate Mineral Resources.

14.0 DATA VERIFICATION

Previous RMI 43-101 Technical Reports have discussed various data verification measures that were undertaken by the author for the Kerr, Sulphurets, and Mitchell properties. This section describes the procedures and results of the author's database verification procedures used for Seabridge's 2010 data.

14.1 Electronic Database Verification

The author performed an audit of the 2010 KSM drill hole database by comparing Eco Tech Laboratory Ltd. certified gold and copper assay results with values stored in Seabridge's electronic database. The author manually checked gold and copper assays from nine of Seabridge's 2010 drill holes for verification. The data that were verified are summarized in Table 14-1 by drill hole and mineral zone. The data shown in Table 14-1 represent about 9.5% of the 2010 Seabridge assay data.

Table 14-1: **2010 Database Verification**

Drill Hole	Zone	Number Checked	Meters Checked	Au Errors	Cu Errors	Ag Errors	Mo Errors
IC-10-011	Iron Cap	264	522	0	0	0	0
IC-10-025	Iron Cap	228	449	0	0	0	0
IC-10-042	Iron Cap	153	293	0	0	0	0
K-10-06	Kerr	139	298	0	0	0	0
M-10-119	Mitchell	200	383	0	0	0	0
S-10-23	Sulphurets	266	522	0	0	0	0
Grand Total	n/a	1,250	2,466	0	0	0	0

RMI notes that no real errors were discovered but in two cases over limit ICP copper analyses were re-run and erroneously posted under a "zinc" column on the Eco Tech paper copy certificate (AS 2010-6304). The Seabridge electronic database had the correct over limit copper values as those data were electronically shipped from Eco Tech to Seabridge and the records were imported into the Seabridge AcQuire® database. RMI also notes that two copper values reported by Eco Tech as 1291.5 and 3676.5 ppm were rounded up by Eco Tech to 1292 and 3677 ppm, respectively. The AcQuire® database export that RMI used as an import into MineSight® rounded those values down to 1291 and 3676 ppm, respectively. RMI does not consider these discrepancies as material or as errors.

It is the author's opinion that the KSM electronic database that was used to estimate Mineral Resources that are the subject of this report is accurate. This is based on the RMI's own independent comparison of certified assays and the database.

14.2 **QA/QC Verification**

Seabridge purchased certified standard reference materials (SRM's) from CDN Resource Laboratories Ltd. (CDN). The SRM's were prepared and certified by CDN from various gold-copper porphyry deposits located in British Columbia and the Yukon. Specific information regarding the composition and round-robin assay results for each of the commercial SRM's that were used by Seabridge can be obtained from CDN's website (www.cdnlabs.com).

Seabridge also had CDN Resource Laboratories Ltd. prepare a custom standard from KSM drill core that has been collected from various Seabridge drilling campaigns. The core was dried and then mechanically ground in a rod mill and then screened through a 270 mesh sieve. The +270 fraction was retained but not used. The -270 fraction (< 53 micron) was mechanically mixed for three days in a V-Blender rotating at approximately 20 rpm. Seventy (70) 100 gram samples where split out and sent for round-robin analysis with 10 samples going to 7 commercial labs. Those labs include ALS Chemex (Vancouver), Acme (Vancouver), Assayers Canada (Vancouver), Actlabs (Ancaster, Ontario), Actlabs (Thunder Bay, Ontario), and TSL Laboratories (Saskatoon). The results from the various labs were returned to Smee & Associates Consulting Ltd. For tabulation and certification. The standards were packaged in lots of 75 grams in tin-tie kraft bags.

Approximately 377 SRM's were submitted to Seabridge's primary laboratory (Eco Tech Laboratories) as a part of their QA/QC program. About 378 blanks were submitted to Eco Tech along with 179 ¼ core duplicate samples. 1,484 Eco Tech pulps were shipped to ALS Chemex in Vancouver for check assay purposes. A more thorough discussion of Seabridge's 2010 quality assurance/quality control procedures was presented in Section 13.

The author personally reviewed the assay results from the certified standards, blanks, duplicate assays, and same pulp check assays and prepared the charts (Figures 13-1 through 13-27).

14.3 **Topographic Contour Data**

In 2008, McElhanney Consulting Services Ltd. of Vancouver, B.C. was contracted to perform an aerial survey and to provide Seabridge with an updated accurate topographic base map of the three deposits and surrounding area. The data were obtained from a helicopter borne LiDAR survey undertaken by McElhanney. LiDAR (Light Detection and Ranging) is an optical remote sensing technology that measures properties of scattered light to find range and other information of a distant target. McElhanney's system uses the Leica ALS50-II Airborne Laser Scanner. This uses a Multiple Pulse in Air (MPiA) system, which is a light-based measuring system which emits photons by laser. LiDAR collects topographical data using laser range and return signal intensity data recorded in-flight. The Leica ALS50 system can yield details under tree cover and orthorectify imagery using specialized software. The product provided included gridded bare earth data to 2 metre spacing and contours at 1 metre intervals in digital formats.

The new topographic map of the district was provided to Seabridge in the UTM NAD83 coordinate system, which is the standard system for all B.C. government and industry mapping applications. Seabridge contracted Aero Geometrics Ltd. of Vancouver to translate the KSM drill hole collar locations from NAD27 to NAD83 datum. Geometrics used MAPS3D software to perform the transformation of all collar coordinates. This software, a product of Sierra Systems, uses the Canadian National Transformation Version 1.1 and 2.0 for the transformation.

RMI and Seabridge noted some discrepancies in the GPS surveyed collar locations and the new LIDAR topographic surface. These differences are thought to be based on 1) no transform of the Z-coordinate was considered by the Canadian National Transformation software 2) the inaccuracy of the initial GPS elevation 3) many of the holes were surveyed immediately below the drill deck and not ground level or "stick-up" and 4) differences magnified by steep terrain. During the 2009 drilling campaign, Seabridge contracted McGladrey & Associates to survey a number of recently completed drill hole collars for quality control purposes. McGladrey & Associates located the drill hole collars (predominantly the tops of the drill hole anchors that remained in the ground and not the actual ground pierce point of the drill hole) using high precision GPS methods. The primary data were then post-processed using the CSRS PPP service, an online global database that provides more precise locations. Table 14-2 shows the difference in collar easting, northing, and elevation between McGladrey & Associates and Seabridge's surveys for twelve 2009 drill holes. Negative values mean that the Seabridge coordinate is less than the McGladrey & Associate value.

Table 14-2: **Drill Hole Collar Survey Checks**

Drill Hole	Zone	Easting (m)	Northing (m)	Elevation (m)
K-09-01	Kerr	1.27	-0.52	-2.25
K-09-02		-0.62	0.76	0.22
MW-09-09		-0.85	0.47	-1.17
MW-09-13A		-0.92	0.71	-0.49
S-09-14	Sulphurets	-0.19	0.05	-0.37
S-09-09		-0.38	-0.51	-0.61
S-09-10		-1.27	0.04	1.15
M-09-097	Mitchell	-0.73	0.71	-0.38
M-09-100		-1.43	0.00	-0.27
M-09-103		-1.21	0.51	-0.59
M-09-106		-0.52	0.65	-0.09
MW-09-10		-1.03	1.02	1.22
Average	n/a	-0.66	0.32	-0.30

The differences in collar locations shown Table 14-2 are not thought to be material given the block size of the resource models and provide some assurances that in general, most of the KSM drill hole collar locations are reasonable. RMI has recommended that all future drill hole collars should be surveyed at ground level to minimize "collar stick-up".

After RMI received an updated topographic surface from Moose Mountain Technical Services, the drill hole collars were compared against the surface. Six drill holes out of the 440 total KSM drill holes were found to be ± 4.0 meters higher (5 holes) or lower (1 hole) than the surface. Hole M-10-117 was found to be a definite survey bust. After conferring with Seabridge personnel, RMI adjusted the six drill holes to match the topographic surface. Table 14-3 summarizes the six holes in which the collar elevation was adjusted.

Table 14-3: Drill Hole Collar Adjustment

Drill Hole	Zone	Surveyed Elevation	Elevation to Topo	Stickup (m)	Adjusted Elevation
S-09-09	Sulphurets	1588.70	1580.58	2.60	1583.18
S-09-13	Sulphurets	1360.70	1356.04	1.50	1357.54
S-09-11	Sulphurets	1527.40	1523.04	1.70	1524.74
M-09-095	Mitchell	969.50	964.40	0.40	964.80
M-09-107	Mitchell	1124.94	1135.43	1.25	1136.68
M-10-117	Mitchell	1188.00	1012.08	1.50	1013.58

14.4 **Specific Gravity Data**

For the Kerr deposit, Placer Dome performed 1,366 bulk density determinations by weighing selected pieces of drill core in air and water using a triple beam balance from which the density calculation was made (i.e. weight in air / weight in air - weight in water). RMI does not know if the samples were completely dried or whether the samples were waxed prior to submersion in water. RMI examined these determinations by lithology, alteration, copper/gold grades, and depth. There was very little difference in the mean density value of 2.84 g/cm^3 by those attributes. Seabridge has since collected 26 bulk density determinations from their 2009/2010 drilling programs using the same methods as described for the Placer Dome determinations. RMI notes that the average bulk density for the Seabridge data was 2.84.

A total of 440 bulk density determinations have been collected for the Sulphurets zone. The majority of those determinations (337) were collected by Placer Dome in 1992. Seabridge collected an additional 85 determinations from their 2009/2010 drilling programs using the weight in air, weight in water method.

A total of 843 bulk density determinations have been performed by Seabridge from their 2006-2010 drilling campaigns using the weight in air, weight in water method.

Seabridge completed 154 bulk density determinations from their 2010 Iron Cap drilling program. Again, the weight in air, weight in water methods was used.

Table 14-4 summarizes the bulk density values used to tabulate resource tonnage by mineral zone.

Table 14-4: KSM Bulk Density Values

Kerr Zone	Bulk Density
Overburden	2.00
CL-PR Alteration	2.81
QSP Alteration	2.88
Weak CLQSP Alteration	2.87
Pre-min Dyke	2.87
Hornblende Dyke	2.85
Default	2.84

Sulphurets Zone	Bulk Density
Overburden	2.00
Hazelton Volcanics	2.77
Diorite	2.85
Monzonite	2.71
Au Bx & Au Leach Bx	2.77
Raewyn Copper	2.77
Default	2.71

Mitchell Zone	Bulk Density
Overburden	2.00
Glacial Ice	0.90
Default Upper Plate (above MTF)	2.71
Default Lower Plate (below MTF)	2.77
Hazelton Volcanics	2.77
Monzonite	2.71
CL-PR Alteration	2.74
QSP/IARG	2.80
Default	2.77

Iron Cap Zone	Bulk Density
Overburden	2.00
Glacial Ice	0.90
Default	2.74

15.0 ADJACENT PROPERTIES

In 2010, Pretium Resources Ltd. purchased the Snowfield and Brucejack mineral resource properties from Silver Standard Resources Inc. (SSRI). Pretium recently announced an updated estimate of Mineral Resources for their Snowfields project, which is located immediately east of Seabridge's Mitchell deposit. Table 15-1 summarizes the publicly disclosed resources of the Snowfield project, which were tabulated using a 0.30 g/t gold equivalent cutoff grade (Pretium, 2001b).

Table 15-1:
Pretium Snowfield Mineral Resources Using a 0.30 g/t AuEq Cutoff

Resource Category	Tonnes (millions)	Au (g/t)	Ag (g/t)	Cu (%)	Mo (ppm)	Re (ppm)	Au Ozs (000)	Ag Ozs (000)	Cu Lbs (billions)	Mo Lbs (millions)	Re Ozs (millions)
Measured	189.8	0.82	1.69	0.09	97.4	0.57	4,983	10,332	0.38	40.8	3.5
Indicated	1,180.3	0.55	1.73	0.1	83.6	0.50	20,934	65,444	2.60	217.5	19.0
Measured + Indicated	1,370.1	0.59	1.72	0.1	85.5	0.51	25,917	75,776	2.98	258.3	22.5
Inferred	833.2	0.34	1.90	0.06	69.5	0.43	9,029	50,964	1.10	127.7	11.5

In addition to disclosing updated resources for their Snowfield deposit, Pretium also disclosed a new resource estimate for their Brucejack property, which is located east of Seabridge's KSM property. The Brucejack deposit consists of nine discrete zones of mineralization. Table 15-2 summarizes the publicly disclosed resources for the combined mineralized zones which makeup the Brucejack project, which were tabulated using a 0.30 g/t gold equivalent cutoff grade (Pretium, 2011a).

Table 15-2:
Pretium Combined Brucejack Mineral Resources Using a 0.30 g/t AuEq Cutoff

Resource Category	Tonnes (millions)	Au (g/t)	Ag (g/t)	Au Ozs (000)	Ag Ozs (000)
Measured	11.7	2.25	75.56	846	28,423
Indicated	285.3	0.80	9.57	7,338	87,782
Measured + Indicated	297.0	0.86	12.17	8,184	116,205
Inferred	542.5	0.72	8.67	12,558	151,220

Pretium has initiated Preliminary Economic Assessment studies for their Brucejack project and concurrent engineering studies for the Snowfield project. The qualified person for this technical report has not verified the resources disclosed by Pretium for their Snowfield and Brucejack deposits. While there appears to be similarities between the Mitchell and Snowfield deposits, the Brucejack mineralization reported by Pretium is not necessarily indicative of mineralization found at the nearby Kerr, Sulphurets, Mitchell, and Iron Cap zones.

RMI has not verified the information shown in Tables 15-1 and 15-2. It is RMI's opinion that a portion of the mineralization shown in Table 15-1 is similar to mineralization associated with the Mitchell zone because the Mitchell and Snowfield zones are located immediately adjacent to each other. However, RMI notes that there are distinct differences between the upper portion of the Snowfield mineralized system and the Mitchell zone.

The mineralization shown in Table 15-2 is unlikely to be indicative of the mineralization currently recognized at the KSM property.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The KSM Project includes four major mineralized zones, identified as the Mitchell, Kerr, Sulphurets and Iron Cap deposits. The deposits contain significant gold, copper, silver and molybdenum mineralization.

Several metallurgical test programs have been carried out to assess the metallurgical response of the mineral materials, especially the samples from the Mitchell deposit. The latest test programs were performed from 2007 through early 2011. The metallurgical testing programs, including historical testing programs, are listed in Table 16-1. The following sections will summarize the test work.

Table 16-1: Metallurgical Test Work Programs

Year	Program ID	Laboratory	Mineralogy	Flotation/ Cyanide Leach	Grindability	Others
2011	KM 2897	G&T		√		
2010/2011	KM 2748	G&T	√	√	√	√
2010	KM 2755	G&T	√	√		√
2010	KM 2670	G&T	√	√		
2009/2010	KM 2535	G&T		√	√	
2009/2010		SGS		√	√	√
2009/2010		Köeppern -UBC			√	
2009	KM 2344	G&T	√	√	√	√
2009		Pocock				√
2008	KM 2153	G&T	√	√	√	√
2008		Hazen			√	
2007	KM 1909	G&T	√	√	√	√
1991		Placer Dome RC		√	√	√
1990		Placer Dome RC	√	√	√	√
1989		Brenda Mines Met Lab		√	√	
1989		Coastech		√		

<u>Abbreviations:</u>
G&T = G&T Metallurgical Services Ltd.
SGS = SGS Mineral Services
Köeppern = Köeppern Machinery Australia Pty Ltd.
UBC = University of British Columbia
Pocock = Pocock Industrial Inc.
Hazen = Hazen Research Inc.
Placer Dome RC = Placer Dome Research Centre
Brenda Mines Met Lab = Brenda Mines Ltd. Metallurgical Laboratory
Coastech = Coastech Research Inc.

16.1 Historical Test Work

Wardrop received several historical test work reports from Seabridge. The test work includes preliminary investigations into mineralogy, material hardness, and metallurgical responses to flotation. Most of the early test work was conducted on the samples from the Kerr zone.

These testing programs used a comparative ball mill work index method to determinate the mineralization hardness and concluded that mineralization was moderately soft for ball mill grinding.

Most of the flotation test work conducted before 1997 was of a preliminary exploratory nature. The test work conducted by Placer Dome Research Centre in 1990 showed that total recoveries for rougher and scavenger flotation ranged from 89 to 96% for copper; from 67 to 94% for gold and from 81 to 95% for silver. The samples indicated poor copper upgrading. Gold recovery into the third cleaner concentrate was approximately 50% on average.

The 1991 additional test work by Placer Dome Research Centre indicated that copper and gold recoveries to the rougher flotation concentrates increased with an increase in primary grinding fineness. At the grind size of 80% passing 99 μm, 87% of the copper was recovered into the rougher concentrate for the Rubble zone sample and 97% for the Crackle Breccia sample. The copper grades were much improved in the final cleaner concentrates to 28% for the Rubble Zone sample and 33% for the Crackle Breccia sample.

16.2 2007-2011 Test Work

Since 2007 twelve main testing programs were carried out to investigate the mineralogical characteristics, ore hardness, metallurgical performance of various mineral samples, and to determine process related parameters, such as unit thickening rates and filtration rates. The metallurgical performance investigations included flotation recoveries of copper, gold, silver and molybdenum minerals, gravity concentration of gold and silver minerals, and cyanide extraction of gold and silver. The flotation test work included open cycle batch tests, locked cycle tests and pilot plant tests. Although most test work was conducted primarily on samples from the Mitchell deposit, the testing programs also investigated the metallurgical performance of samples from the Sulphurets, Kerr and Iron Cap deposits.

In general, the mineralization from the four different deposits responded similarly to a flotation and sulphide concentrate cyanidation process with respect to copper, gold, silver and molybdenum recoveries. The Mitchell samples gave the most consistent results throughout the testing program.

16.2.1 Mitchell Mineralization

Test Samples

All the testing samples for the various testing programs were collected from diamond drill cores produced from various drilling programs.

The 2007 testing program used three composite samples. Table 16-2 shows the chemical assays and key mineral distribution of the composite samples.

Table 16-2: Test Samples – Mitchell, 2007 (G&T)

	Units	Composite			Average
		A	B	C	
Element		**Assay**			
Copper	%	0.2	0.2	0.2	0.2
Gold	g/t	0.9	0.9	0.9	0.9
Silver	g/t	3.0	4.0	4.0	4.0
Sulphur	%	4.6	3.6	1.8	3.3
Mineral		**Distribution**			
Chalcopyrite	%	0.6	0.6	0.6	0.6
Pyrite	%	10.0	9.4	4.2	7.9
Gangue	%	89.5	90.0	95.2	94.9

The later test work used the samples collected from 2008 and 2009 drilling programs. The 2008 testing program used a total of approximately 5,720 kg of drill core samples for the testing. Most of the samples were collected from the Mitchell Zone. The variability testing samples are listed in Table 16-3.

Table 16-3: Head Assay on Variability Test Samples – Mitchell, 2008 (G&T)

Sample ID	Metal Content (% or g/t)*					Sample ID	Metal Content (% or g/t)*				
	Cu	Au	Ag	Mo	As		Cu	Au	Ag	Mo	As
MET 2	0.25	0.82	4	0.003	0.003	MET 19	0.30	0.67	4	0.002	0.001
MET 3	0.24	0.65	8	0.004	0.020	MET 20	0.17	0.54	4	0.005	0.004
MET 4	0.26	0.83	3	0.004	0.001	MET 21	0.21	0.83	2	0.004	0.003
MET 5	0.20	0.66	2	0.004	0.001	MET 22	0.20	0.85	3	0.011	0.002
MET 6	0.21	0.74	2	0.010	0.001	MET 23	0.11	0.32	3	0.025	0.010
MET 7	0.28	1.49	3	0.001	0.002	MET 24	0.24	0.86	3	0.001	0.053
MET 8	0.21	0.57	2	0.003	0.002	MET 25	0.14	0.43	2	0.007	0.005
MET 9	0.13	0.48	2	0.002	0.002	MET 26	0.13	0.68	2	0.002	0.004
MET 10	0.07	0.39	3	0.010	0.004	MET 27	0.15	0.82	2	0.003	0.002
MET 11	0.19	0.64	3	0.003	0.003	MET 28	0.16	0.86	3	0.012	0.001
MET 12	0.20	0.79	3	0.002	0.001	MET 29	0.19	0.79	5	0.018	0.006
MET 13	0.30	1.24	4	0.002	0.003	MET 30	0.14	0.22	3	0.003	0.005
MET 14	0.31	1.31	18	0.001	0.004	MET 32	0.22	1.18	2	0.002	0.006
MET 15	0.28	0.87	3	0.003	0.003	MET 33	0.33	0.96	7	0.002	0.008
MET 16	0.44	1.24	5	0.001	0.001	MET 34	0.28	0.85	3	0.004	0.002
MET 17	0.27	0.74	3	0.003	0.003	MET 35	0.12	0.30	1	0.003	0.008
MET 18	0.28	1.34	5	0.001	0.004	MET 36	0.52	0.81	1	0.023	0.005

* g/t for Au and Ag.

A total of ten additional composites were generated from the MET samples, including nine composite samples representing the major Mitchell Zone mineralization types that were projected to be mined during the different mining periods laid out in the mine plan generated from the Preliminary Assessment study. The feed grades for the composites are shown in Table 16-4.

Table 16-4: Head Assay on Composites – Mitchell, 2008 (G&T)

Sample ID	Metal Content				
	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	As (%)
QSP 0-10	0.24	0.94	4	0.001	0.004
QSP 10-30	0.23	1.08	8	<0.001	0.004
QSP 0-30	0.24	0.95	4	0.004	0.002
QSP 0-10 LG	0.17	0.86	4	0.004	0.007
Hi Qtz 0-10	0.21	1.08	4	0.004	0.004
Hi Qtz 10-30	0.27	0.90	4	<0.001	0.004
Hi Qtz 0-30	0.25	1.02	4	0.004	0.001
Prop 10-30	0.26	1.00	3	<0.001	0.001
IARG 0-10	0.10	0.60	4	0.006	0.006
Master Comp 1	0.19	0.84	4	0.003	0.003

The 2009-2010 testing programs used a total of 12.1 tonnes of core samples from 3,218 different drill core intervals from the Mitchell and Sulphurets deposits. Eleven composites were generated from the Mitchell deposit according to mineralization types. The metal contents in the composite samples from the Mitchell deposit are shown in Table16-5.

Table 16-5: Metal Contents of Composites – Mitchell, 2009 (G&T)

Composite	Mineralization Type*	Metal Content						
		Cu(T) (%)	Cu(ox) (%)	Cu(CN) (%)	Au(T) (g/t)	Au(CN) (g/t)	Mo (%)	Ag (g/t)
Comp 40	CL-PR	0.20	0.006	0.008	0.67	0.013	0.004	3.6
Comp 41	BBRX	0.71	0.006	0.008	0.35	0.007	0.010	8.9
Comp 42	QSP	0.28	0.006	0.011	1.02	0.009	0.002	4.1
Comp 43	CL-PR	0.22	0.004	0.011	0.70	0.004	0.004	3.1
Comp 44	Hi Qtz	0.27	0.008	0.019	0.92	0.006	0.010	4.2
Comp 45	IARG	0.13	0.002	0.004	0.57	0.013	0.010	3.5
Comp 46	CL-PR	0.15	0.003	0.004	0.67	0.012	0.011	2.0
Comp 47	QSP	0.16	0.004	0.006	0.73	0.015	0.013	2.3
Comp 48	QSP	0.10	0.003	0.002	0.61	0.013	0.015	2.2
Comp PP1	Blend	0.24			0.76		0.004	
Comp PP2	Blend	0.18			0.64		0.010	

*** Notes:**
QSP: Quartz, sericite, pyrite altered rocks
IARG: Intermediate argillic altered rocks (quartz, sericite, chlorite, pyrite, ±clays)
CL-PR: Chlorite-propylitic altered rocks (quartz, chlorite, pyrite, ±magnetite, ±epidote, ±calcite)
Hi Qtz: Altered rocks with >60% quartz veining by volume, higher than average pyrite (7-15%)
BBRX: Bornite breccia (breccia w/bornite, chalcopyrite, pyrite in matrix of quartz, clay, anhydrite)
Blend: Blend from various mineralization types for pilot plant testing
Cu(T): Total copper; **Cu(OX):** oxide copper; **Cu(CN):** cyanide soluble copper
Au(T): Total gold; **Au(CN):** cyanide soluble gold.

The assay data indicated that the copper mineral oxidation level was low, only 3% or less of the copper is present in oxide forms.

The Composite PP1 sample was constructed from CL-PR, QSP and Hi Qtz mineralization, the three dominant mineralization types of the Mitchell deposit. Composite PP2 was selectively prepared with higher molybdenum core intervals.

In 2010, three additional Mitchell zone composites were generated using the drill core interval samples from the 2009-2010 drilling program. The sample details are shown below and in Table 16-6.

- PP Composite 3: the crushed materials generated from high pressure grinding rolls (HPGR) tests (approximately 12 tonnes) for bench tests and pilot plant tests.

- PP Hi-Mo Composite: halved drill cores (approximately 6.3 tonnes)

- BS Hi-Mo Composite: high molybdenum content drill cores selected from halved drill cores for PP Hi-Mo composite.

Table 16-6: Metal Contents of Composites – Mitchell, 2010 (G&T)

Sample	Cu (%)	Mo (%)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)
PP Composite 3	0.20	0.006	4.29	3.66	0.79	3.2
BS Hi-Mo Composite	0.12	0.013	3.95	3.27	0.57	2.4
PP Hi-Mo Composite	0.16	0.012	4.02	3.67	0.60	-

Mineralogy

The mineralogical composition study of the 2008 testing program shows that the sulphide mineral content in all three samples (QSP 0-30, Hi Qtz 0-30, and Master Composite 1) is dominated by pyrite which is present as approximately 6 to 8 % of the sample weight. The study also indicated that the copper was present in the form of chalcopyrite. Detailed analysis data are presented in Table 16-7.

Table 16-7: Mineral Composition Data – Mitchell, 2008 (G&T)

Sample	Mineral Composition (%)		
	Chalcopyrite	Pyrite	Gangue
QSP 0-30	0.66	6.6	92.7
Hi Qtz 0-30	0.67	8.2	91.2
Master Comp	0.54	8.1	91.4

The pyrite to chalcopyrite ratios are relatively high in the three composite samples. The average ratio is 12:1 while the highest ratio reaches 15:1. There does not appear to be close pyrite-chalcopyrite interlocking. Figure 16-1 illustrates the typical relationship among the main minerals in the samples.

Figure 16-1: Mineral Relationship – Master Composite, Mitchell

Particle Fractions <75 µm >32 µm: Particle Fractions <150 µm >75 µm:

Note: Cp = Chalcopyrite, Py = Pyrite, Ma = Magnetite, He = Hematite, Gn = Gangue.

The degree of chalcopyrite liberation ranged from 46 to 56% across the samples tested at a primary grind size of 80% passing 116 µm to 136 µm. The Hi Qtz sample showed a higher two-dimensional chalcopyrite liberation than the QSP sample. A primary grind size of 80% passing 125 µm was recommended for the Mitchell Zone.

Mineralization Hardness

Various grindability tests have been conducted in a number of test programs including SMC testing, crushing characteristics to HPGR, and standard Bond ball mill work index determination.

Grindability/Crushability Determination - Bond Ball Mill Work Index

Both G&T and SGS carried out standard Bond ball mill work index tests on the Mitchell mineralization. As summarized in Table 16-8, the Bond work indices determined from different testing programs range from12.5 kWh/t to 15.5 kWh/t, averaging 14.4 kWh/t. The data suggests that the Mitchell samples are of moderate hardness. The Bond abrasion index (Ai) of Composite PP1 was measured at 0.293 g by SGS.

Table 16-8: Bond Ball Mill Wi Test Results – Mitchell, 2008

Samples	Wi (kWh/t)	Ai (g)	Samples	Wi (kWh/t)	Samples	Wi (kWh/t)
2009 G&T			2008 G&T		2007 G&T	
Composite 40	15.5		High Quartz 0-10	15.2	A	14.7
Composite 41	14.8		High Quartz 10-30	15.3	B	14.8
Composite 42	15.2		IARG 0-10	13.9	C	14.8
Composite 43	14.6		QSP 0-10	14.5		
Composite 44	13.4		QSP 10-30	15.2		
Composite 45	14.1					
Composite 46	12.8					
Composite 47	13.3					
Composite 48	12.5					
Sub Average	14.0					
2009/2010 SGS						
Composite PP1	13.8	0.293				
Total Average	**13.9**			**14.8**		**14.8**

Grindability/Crushability Determination - SMC Tests

The SMC grindability tests were conducted by Hazen in 2008. The samples used for the grindability tests were identified as QSP, IARG, CL-RICH, QSP STW/QTVN, and H FELDS. The SMC test results are shown in Table 16-9.

Table 16-9: SMC Test Results – Mitchell, 2008

Parameter Sample	Value				
	QSP	IARG	CL-RICH	QSP STW/QTVN	H FELDS
SG	2.81	2.42	2.78	2.69	2.71
A (maximum breakage)	70.7	75	68.1	82.6	81.6
B (relation between energy & impact breakage)	0.71	0.4	0.57	0.6	0.44
Axb (overall Au-SAG hardness)	50.2	30	38.8	49.6	35.9
Drop Weight Index (DWi)	5.5	7.9	7.1	5.4	7.5
Coarse ore work index, Mia (kWh/t)	16.1	24.8	19.9	16.3	21.2
Ta (estimated abrasion parameter)	0.47	0.33	0.37	0.49	0.35

The DWi and Axb data indicate that on average, the materials were moderately hard to SAG mill grinding in comparison to the JK Tech database. Contract Support Services conducted a few of process simulations to develop the primary grinding circuit arrangement and to estimate equipment sizing.

Grindability/Crushability Determination - HPGR

In 2009 and 2010, two separate HPGR comminution characteristic testing programs were performed – bench scale testing at SGS and pilot plant scale tests at Köeppern's HPGR pilot plant at UBC.

The bench scale LABWAL tests by SGS were conducted on the Mitchell and Sulphurets composite samples. The tests included batch tests and locked cycle tests (LCT). The test results indicate that the Sulphurets mineralization is harder with respect to HPGR crushing than the Mitchell mineralization. On average, the net specific energy requirement is 2.33 kWh/t for the Mitchell sample and 3.08 kWh/t for the Sulphurets sample. The locked cycle test results, including specific grinding force (N/mm^2) and specific throughput rate (ts/hm^3-(m_c)), are summarized in Table 16-10.

Table 16-10: HPGR Average Test Results – LCT, Mitchell, 2009-2010

Parameter	Unit	Mitchell	Sulphurets
Operation			
Pressure of Operation	bar	65	66
Moisture	% H_2O	1.8	1.7
Dry Net Throughput	t/h	1.9	1.6
Circulating Load	%	34.7	47.1
Net Power	kW	4.4	5.1
Gross Specific Energy Requirement	kWh/t	2.96	3.8
Net Specific Energy Requirement	kWh/t	2.33	3.08
HPGR Product Analysis			
50% Passing	µm	694	1,046
80% Passing	µm	1,988	2,220
Percent Passing 100 mesh		25.3	17.7
Percent Passing 6 mesh		100	100
Flake Thickness	mm	6	5.8
Performance Indicators			
Specific Grinding Force	N/mm^2	3.24	3.31
Specific Throughput	ts/hm^3-(mf)	226	213
Specific Throughput Rate	ts/hm^3-(mc)	195	187
Ratio mj/mf		0.86	0.88
Specific Power	kWs/m^3	528	657
New minus 100 Mesh Produced	%	19.6	11.9
New minus 6 Mesh Produced	%	73.5	60.6

Based on the test results, SGS also conducted related simulations to size the HPGR.

Köeppern conducted a pilot plant tests in 2010 at its HPGR pilot plant at UBC using approximately 5.5 t of drill core samples collected from the Mitchell deposit. The pilot plant HPGR rollers are 0.75 m in diameter and 0.22 m in width. A lower net specific energy consumption of approximately 1.94 kWh/t was recorded for the closed circuit tests, in comparison with 1.99 kWh/t obtained from the single pass tests.

The HPGR test work program showed that the Mitchell material is very amenable to the HPGR crushing process.

Grindability/Crushability Determination – Tower Mill

As a part of the 2009 testing program, Metso Minerals Industries Inc. investigated the specific energy consumption for secondary grinding using tower mills. The mill feed particle size was 80% passing 173 µm and the product particle size was 125 µm. The test results indicate that the specific energy requirement for the grinding by a jar mill was 1.36 kWh/t for the Mitchell composite sample. As projected by Metso, the specific energy requirement, by a stirred tower mill, would be approximately 0.88 kWh/t for a similar particle size reduction.

Grindability/Crushability Determination – Regrinding/IsaMill™

SGS used the IsaMill™ procedure to determine the specific energy requirement for regrinding the gold bearing pyrite rougher concentrate which was produced from the Mitchell samples. The tests indicated that the specific energy requirement to regrind the concentrate from 80% passing 66 µm to 80% passing 16 µm was 24.2 kWh/t. The grinding media consumption, 2 mm Keramax MT1 grind beads, was 6 g/kWh.

<u>Process Flowsheet and Parameter Development</u>

Substantial test work was conducted to develop the process flowsheet and to optimize the process conditions through various testing programs. A flotation-cyanidation combination process was developed for this mineralization. The process consists of:

- copper-gold-molybdenum bulk rougher flotation followed by gold-bearing pyrite flotation
- regrinding the resulting bulk rougher concentrate followed by three stages of cleaner flotation to produce a copper-gold-molybdenum bulk cleaner flotation concentrate
- molybdenum separation of the bulk cleaner flotation concentrate to produce

a molybdenum concentrate and a copper/gold concentrate containing associated silver

- cyanide leaching of the gold-bearing pyrite flotation concentrate and the scavenger cleaner tailing product.

The development of the flotation and cyanidation test conditions is summarized below:

Flotation Tests

Flotation Parameter Development Tests

The tested process parameters for copper-gold-molybdenum bulk concentrate and gold-bearing pyrite concentrate include primary grind size, regrind size, slurry pH, and reagent regimes. After various tests, the following flotation conditions were developed for the locked cycle tests in the most recent testing programs:

- primary grind size: 80% passing approximately 125 μm
- rougher flotation pH: 10
- bulk concentrate regrind size: 80% passing approximately 20 μm
- cleaner flotation pH: 11.5
- flotation reagent
  - bulk flotation: 3418A (dithiophosphinates) + A208 (dithiophospate) + fuel oil
  - gold-bearing pyrite flotation: A208 + potassium amyl xanthate (PAX).

The open circuit batch tests showed that the mineralization responded well to these flotation conditions.

Variability Tests

In the 2008 testing program, a total of 34 samples were used for variability tests, including two samples (Met 35 and Met 36) from Sulphurets Zone. Primary grind sizes ranged from 80% passing 115 to 171 µm, averaging 149 µm. The rougher concentrate from the copper circuit was reground to approximately 80% passing18 µm prior to cleaner flotation.

It appeared that the copper recoveries reporting to the third cleaner concentrates in the open circuit tests increased with copper feed grade. As shown in Figure 16-2, G&T

established the relationship between copper recovery and copper feed grade at a fixed concentrate grade of 25% Cu. The variation in the copper metallurgical performance of various mineral samples is shown in Figure 16-3.

Figure 16-2: Copper Recovery vs. Copper Feed Grade – Mitchell, 2008 (G&T)

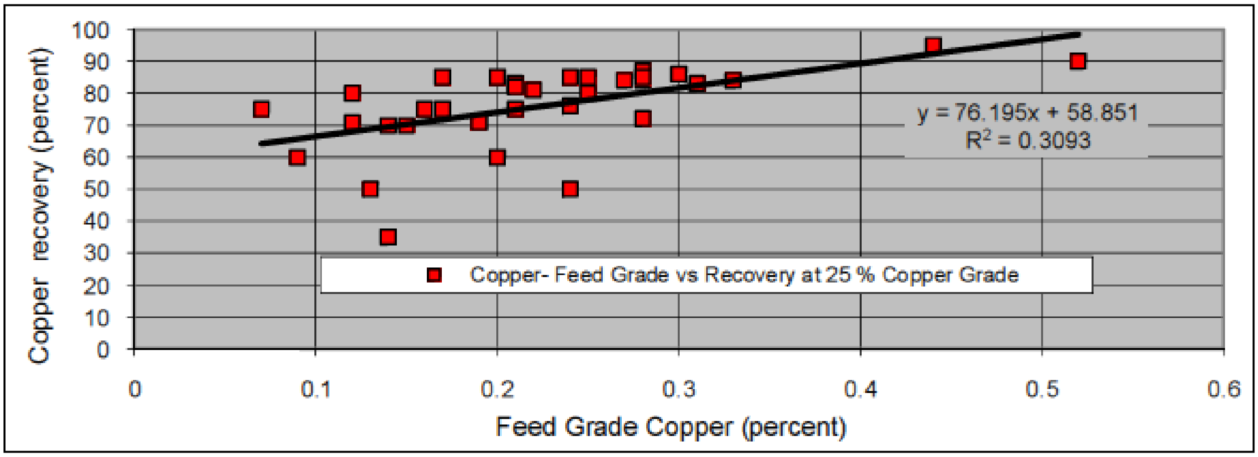
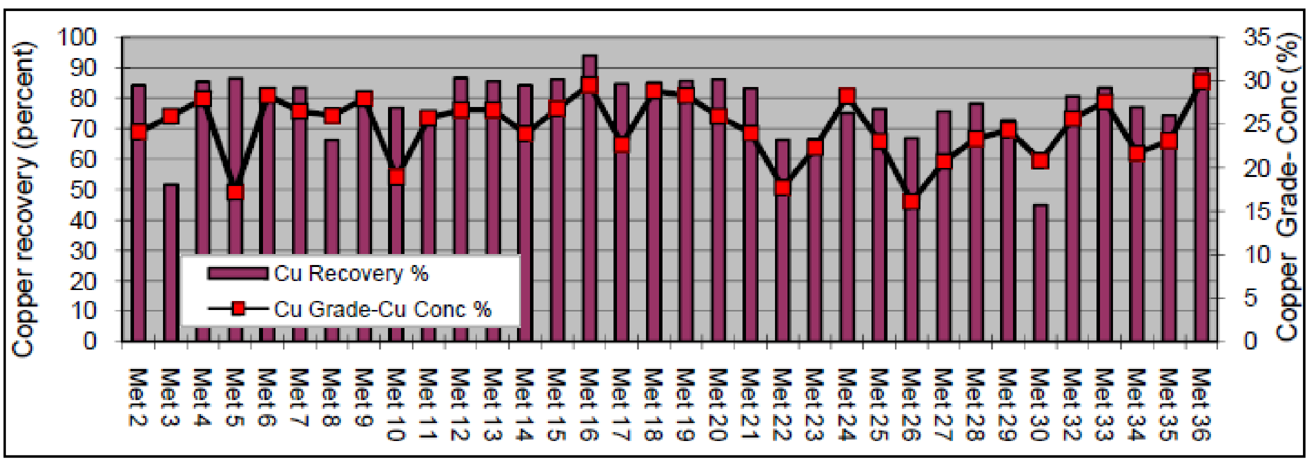
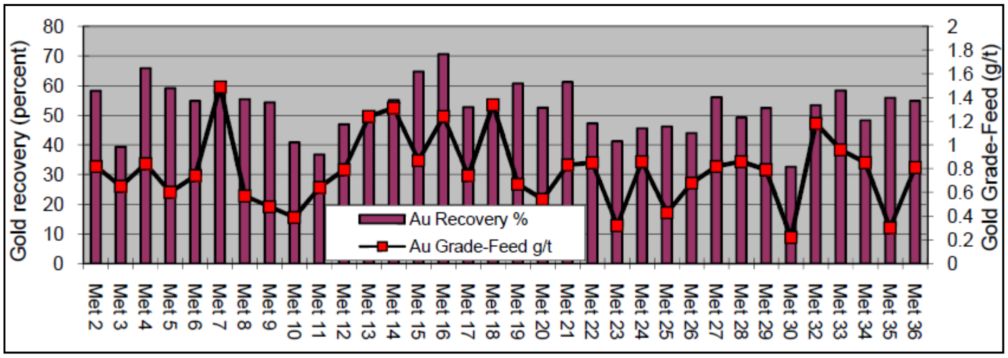


Figure 16-3: Copper Recovery & Concentrate Grade – Individual Samples, Mitchell, 2008 (G&T)



The gold recovery to the copper concentrate fluctuated from 30% to 70%. The tests seemed to show that gold recovery to copper concentrate increased as a function of head gold content; however, the correlation was not strong. The gold metallurgical performance is plotted in Figure 16-4.

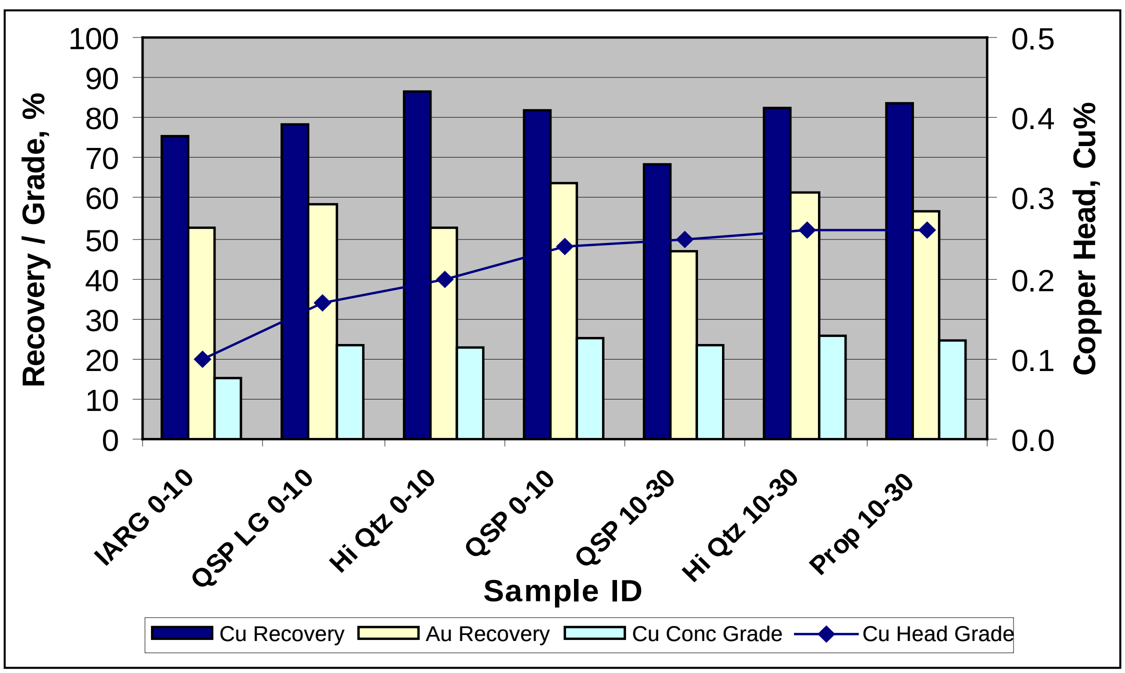
Figure 16-4: Gold Recovery & Feed Grade – Individual Samples, Mitchell, 2008 (G&T)



Gold recoveries to the gold bearing pyrite concentrate from the pyrite flotation circuit varied from 4 to 29%, averaging approximately 16%. Combined gold recoveries from both the copper flotation circuit and gold bearing pyrite flotation circuit ranged from 73 to 96%, averaging approximately 86%.

Further tests were conducted on seven composites representing the major Mitchell zone mineralization types projected to be mined during various operating periods. The test results are shown in Figure 16-5. At primary grind sizes ranging from 130 to 168 µm, the third cleaner concentrates from the open batch flotation tests produced between 69% and 86% copper recovery and between 47 and 64% gold recovery.

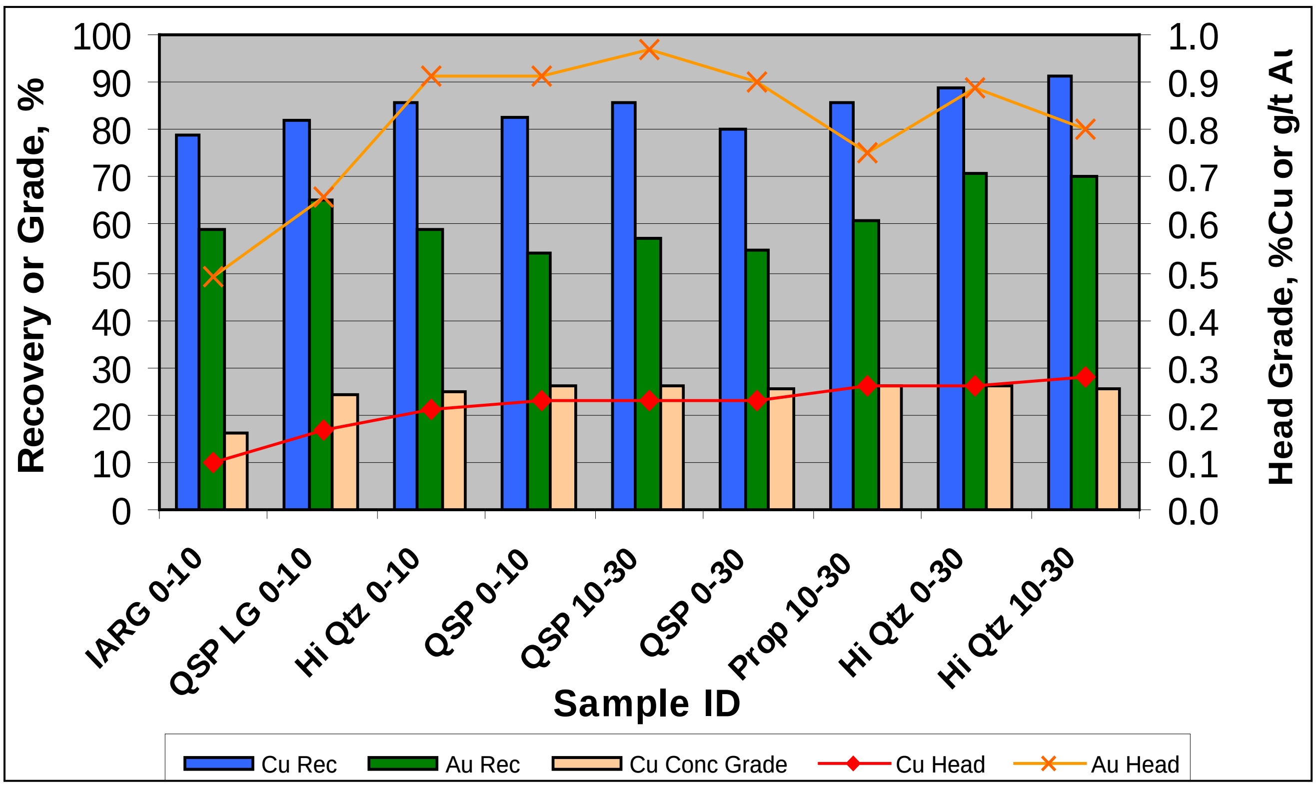
Figure 16-5: Metallurgical Performance – Composites, Mitchell, 2008 (G&T)



Similar to the MET sample variability tests, the total average gold recovery from the copper-gold rougher and scavenger flotation was approximately 86% from the composite samples.

Open circuit tests with two stages of cleaner flotation at a pH of 11.5 were performed on the nine composite samples. Primary grind sizes ranged from 80% passing 87 µm to 137 µm, averaging 119 µm. Regrind sizes varied from 80% passing 12 µm to 22 µm, averaging 18 µm. The results are shown in Figure 16-6.

Figure 16-6: Metallurgical Performance – Open Circuit Tests, Mitchell, 2008 (G&T)



The second cleaner concentrate recovered between 79 to 91% of the copper and 54 to 71% of the gold from all nine composites. On average, the metal recovery was 84.6% for copper and 61.2% for gold.

The results appeared to indicate that copper recovery increased with an increase in copper head grade. The test results also showed that gold recovery to the copper concentrate did not appear to correlate with gold head grade or copper head grade.

The average test results from the 2009-2010 flotation test work are summarized in Figure 16-7 and Figure 16-8. The results show that there is a significant variation in the metallurgical performance between the different ore samples. The BBRX mineralization

(Composite 41) showed the best metallurgical response to the flowsheet. This was most likely due to the much higher feed grade in this composite. Compared to the 2008 Hi Qtz mineralization test results, the Hi Qtz mineralization (Composite 44) produced a slightly lower level of metallurgical performance.

Figure 16-7: Copper Metallurgical Performance –Mitchell, 2009 (G&T)

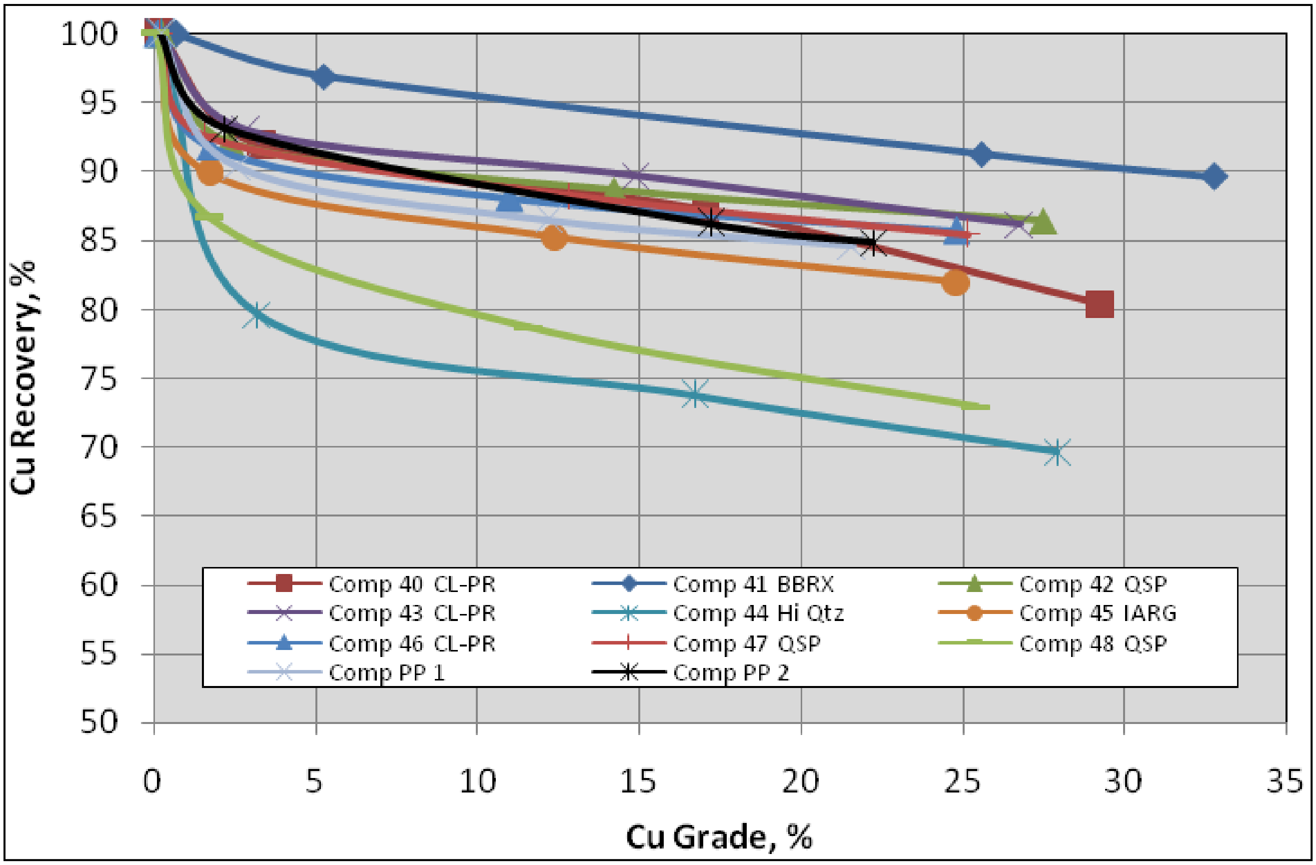
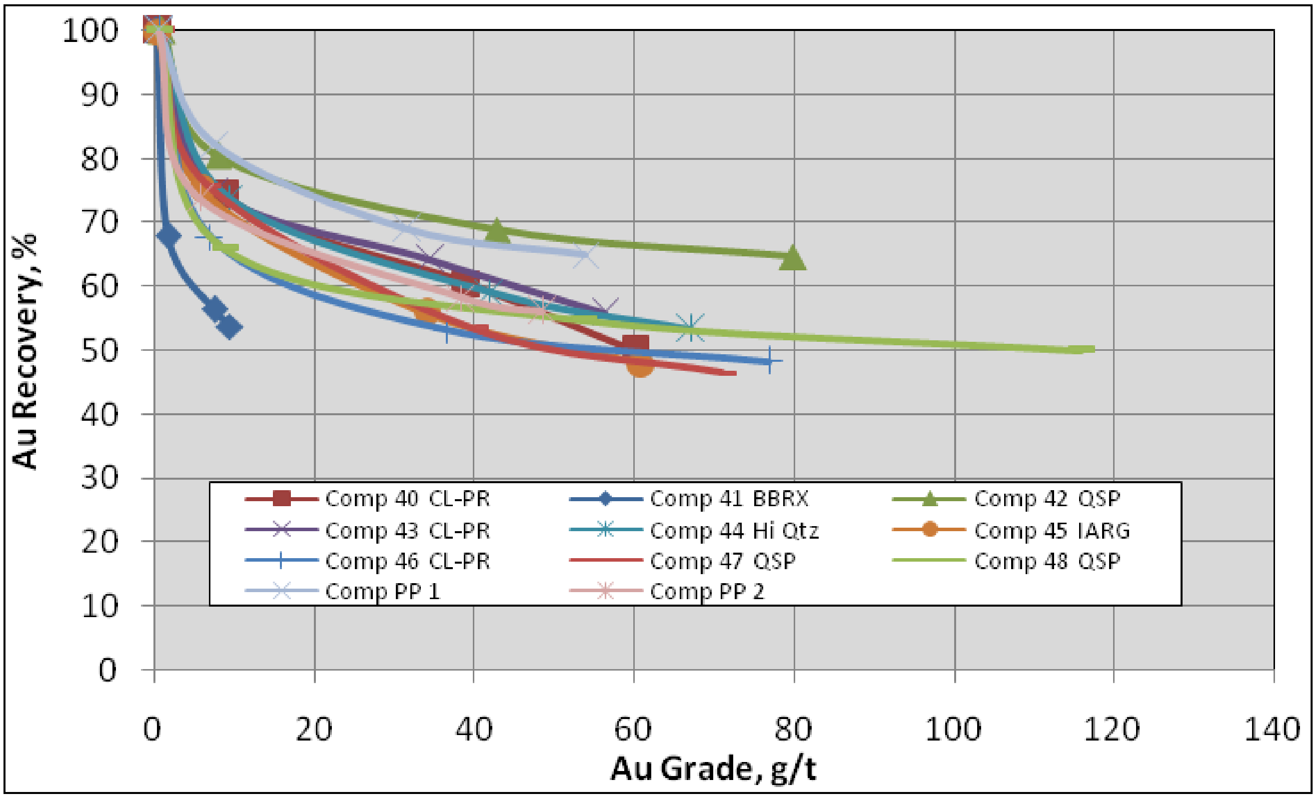


Figure 16-8: Gold Metallurgical Performance –Mitchell, 2009 (G&T)



The results also show that most of the cleaner concentrate grades of the individual composites were greater than or close to 25% Cu, averaging 28% Cu. However, the Composites PP 1 and PP 2 produced lower grade concentrates containing 22% Cu. The average copper recovery was 83%. The average gold recovery to the final copper concentrates was 55%.

Locked Cycle Tests

Fourteen locked cycle tests have been conducted on various composite samples. The test results are summarized in Table 16-11 for the Mitchell mineralization and Table 16-12 for the samples blended from the Mitchell mineralization and the other mineralization.

Table 16-11: Locked Cycle Test Results – Mitchell

Test Program	Comp	Product	Grind Size (P_{80} µm)	Mass (%)	Grade (% or g/t)				Flotation Recovery (%)			
					Cu	Au	Ag	Mo	Cu	Au	Ag	Mo
G&T	Master	Head		100.0	0.21	0.89	4.2		100.0	100.0	100.0	
2153/141		Cu/Mo Concentrate	119/16	0.9	20.2	62.8	273		87.8	63.0	58.5	
		Bulk Cleaner Tailings		7.0	0.10	1.66			3.3	13.0		
		Au-Pyrite Concentrate		5.6	0.10	2.02			2.6	12.7		
G&T	Master	Head		100.0	0.21	0.92	3.7		100.0	100.0	100.0	
2153/142		Cu/Mo Concentrate	119/17	0.8	22.0	64.7	242		87.0	58.5	52.5	
		Bulk Cleaner Tailings		6.9	0.14	2.08			4.5	15.7		
		Au-Pyrite Concentrate		6.0	0.11	2.25			3.0	14.6		
G&T	PP Comp 1	Head		100.0	0.24	0.81			100.0	100.0		
2344/73		Cu/Mo Concentrate	103/14	1.0	22.3	55.7			89.3	66.2		
		Bulk Cleaner Tailings		6.8	0.13	1.70			3.7	14.0		
		Au-Pyrite Concentrate		2.5	0.13	1.80			1.4	5.5		
G&T	PP Comp 1	Head		100.0	0.23	0.84	4.0		100.0	100.0	100.0	
2535/18		Cu/Mo Concentrate	103/16	0.7	28.0	77.8	260		87.2	67.4	47.0	
		Bulk Cleaner Tailings		7.4	0.19	1.62	17.6		6.0	14.2	32.0	
		Au-Pyrite Concentrate		2.5	0.19	1.37	7.1		2.0	4.1	4.4	
G&T	PP Comp 1	Head		100.0	0.24	0.82	3.9		100.0	100	100.0	
2535/20		Cu/Mo Concentrate	137/17	0.9	23.8	62.0	248		88.1	66.2	55.6	
		Bulk Cleaner Tailings		7.4	0.10	1.61	11.3		2.9	14.4	21.2	
		Au-Pyrite Concentrate		2.8	0.21	1.69	7.2		2.4	5.6	5.1	
G&T	PP Comp 3	Head		100.0	0.20	0.74	3.2	0.006	100.0	100.0	100.0	100.0
2670/12		Cu/Mo Concentrate	147/15	0.6	30.1	77.7	264	0.386	84.2	58.0	52.6	35.7
		Bulk Cleaner Tailings		6.2	0.19	1.49		0.036	6.0	12.5		37.9
		Au-Pyrite Concentrate		4.9	0.12	2.04		0.014	3.1	13.6		11.6
G&T	PP Comp 3	Head		100.0	0.20	0.79	3.2	0.006	100.0	100.0	100.0	100.0
2670/18		Cu/Mo Concentrate	147/22	0.6	27.4	70.5	272	0.462	86.1	56.5	53.0	49.7
		Bulk Cleaner Tailings		6.0	0.13	1.98	9.3	0.016	3.9	15.1	17.4	15.8
		Au-Pyrite Concentrate		4.4	0.15	2.26	6.4	0.016	3.4	12.7	8.8	11.7
G&T	PP Hi Mo	Head		100.0	0.16	0.60	3.3	0.014	100.0	100.0	100.0	100.0
2670/22		Cu/Mo Concentrate	143/21	0.6	22.4	61.7	243	1.200	78.9	56.9	43.8	47.9
		Bulk Cleaner Tailings		6.6	0.17	1.87	10.0	0.042	7.3	20.6	19.8	19.9
		Au-Pyrite Concentrate		5.6	0.16	1.39	6.9	0.026	5.7	12.9	11.6	10.2
G&T	BS Hi Mo	Head		100.0	0.12	0.55	2.4	0.010	100.0	100.0	100.0	100.0
2670/26		Cu/Mo Concentrate	143/17	0.3	24.9	70.3	185	1.258	71.5	43.2	26.0	42.2
		Bulk Cleaner Tailings		5.8	0.27	1.58	9.7	0.049	13.3	16.6	23.4	28.1
		Au-Pyrite Concentrate		5.7	0.13	1.79	5.5	0.026	6.0	18.3	13.1	14.5
G&T	Comp 46	Head		100.0	0.15	0.65	2.3	0.012	100.0	100.0	100.0	100.0
2897/01	of	Cu/Mo Concentrate	120/16	0.6	22.6	80.5	226	1.759	89.1	73.5	58.6	86.3
	KM2344	Bulk Cleaner Tailings		7.6	0.04	1.01	4.6	0.008	2.1	11.8	15.3	5.1
		Au-Pyrite Concentrate		5.6	0.09	1.16	2.9	0.003	3.3	10.0	7.2	1.4
SGS	PP Comp 1	Head		100.0	0.21	0.72		0.005	100.0	100.0		100.0
		Cu/Mo Concentrate	129/28	0.8	23.1	53.7		0.410	89.0	59.6		65.0
		Bulk Cleaner Tailings		9.2	0.06	1.54		0.009	2.62	19.8		13.2
		Au-Pyrite Concentrate		5.8	0.09	0.81		0.013	2.60	6.6		12.0

***** Primary Grind Size/Regrind Size

Table 16-12: Locked Cycle Test Results – Blended Samples

Test Program	Comp	Product	Grind Size* (P_{80} µm)	Mass (%)	Grade (% or g/t)				Flotation Recovery (%)			
					Cu	Au	Ag	Mo	Cu	Au	Ag	Mo
G&T 2535/19	Mitchell (PP Comp1)/Kerr (52/53 Blend)	Head	127/20	100.0	0.31	0.70	3.5		100.0	100.0	100.0	
		Cu/Mo Concentrate		1.1	25.3	40.0	168		87.4	60.4	51.4	
		Bulk Cleaner Tailings		8.0	0.12	1.36	8.2		3.2	15.5	18.9	
		Au-Pyrite Concentrate		4.2	0.24	0.94	5.9		3.3	5.7	7.1	
G&T 2670/62	Mitchell / Sulphurets Blend	Head	141/22	100.0	0.22	0.67	2.8	0.007	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.8	24.2	52.0	178	0.664	85.9	59.8	50.9	72.4
		Bulk Cleaner Tailings		8.6	0.09	1.40	5.6	0.008	3.6	18.1	17.2	9.7
		Au-Pyrite Concentrate		3.9	0.19	1.47	4.9	0.010	3.5	8.6	6.8	5.5
G&T 2748/18	Mitchell (1/3 PP Comp 1)/ Iron Cap (1/3 C1+1/3 C2)	Head	135/15	100.0	0.24	0.79		0.004	100.0	100.0		100.0
		Cu/Mo Concentrate		0.8	27.6	59.6		0.250	87.8	58.2		51.5
		Bulk Cleaner Tailings		8.2	0.09	1.52		0.010	2.9	15.7		20.7
		Au-Pyrite Concentrate		7.4	0.13	1.85		0.003	4.0	17.4		5.4

* Primary Grind Size/Regrind Size

The test results showed a substantial variation in the concentrate grade, ranging from 20% Cu to 30% Cu. On average, the final copper concentrate contained 24.3% Cu. The average recoveries to the concentrate were 86% for copper, 61% for gold, 50% for silver and 55% for molybdenum. Approximately 26% of the gold and 29% of the silver in the feed reported to the gold bearing products which will be further extracted by cyanide leaching. The test results showed that better metallurgical performance was achieved in the more recent testing programs.

As shown in Table 16-12, the metallurgical performances achieved from the samples blended from the Mitchell mineralization and the other mineralization were comparable to the performance attained from the Mitchell samples.

Pilot Plant Tests

In the 2009 testing program, G&T carried out initial pilot plant tests using approximately 5 tonnes of coarsely crushed drill core. Compared to the bench locked cycle tests, the pilot plant tests produced lower metal recoveries and concentrate grades.

The average copper recovery to the copper-gold concentrate with 18% Cu was 72% for the Composite PP1 sample. Test P2 produced a 23.9% Cu concentrate. G&T indicated that the low copper recovery might have resulted from pilot plant control or circuit stability issues. This in turn caused copper losses into the pyrite circuit and the 1st cleaner tailings. These initial pilot plant results are summarized in Table 16-13.

Table 16-13: Pilot Plant Test Results – Mitchell, 2009 (G&T)

Test	Grind Size (P_{80} µm*)	Grade			Recovery (%)			
		Cu (%)	Au (g/t)	Mo (%)	Mass	Cu	Au	Mo
Composite PP1 (Head Assay: 0.24% Cu, 0.76 g/t Au, 0.004% Mo)								
P1	144	17.1	33.6	0.15	1.0	65.4	46.1	31.5
P2	96	23.9	59.6	0.17	0.7	65.2	51.9	23.6
P3	104	16.3	35.7	0.14	1.3	80.2	58.6	40.8
P4	103	15.5	29.8	0.03	1.2	74.3	50.7	8.8
P5	97	18.4	41.4	0.12	0.9	76.0	52.3	26.4
Average	**109**	**18.2**	**40.0**	**0.12**	**1.0**	**72.2**	**51.9**	**26.2**
Composite PP2 (Head Assay: 0.18% Cu, 0.61 g/t Au, 0.010% Mo)								
P6	84	16.7	33.0	0.70	1.0	79.7	50.3	54.8
P7	91	17.7	42.5	0.95	0.9	81.7	60.5	72.3
P8	88	18.0	36.9	0.81	0.9	79.1	47.4	65.8
Average	**88**	**17.4**	**37.5**	**0.80**	**0.9**	**80.2**	**52.7**	**64.3**

* Primary Grind Size

In the 2010 testing program, G&T further conducted two pilot plant runs on the PP Composite 3 and the PP Hi-Mo Composite samples. Compared to the 2009 pilot plant tests, the 2010 testing program produced much better metallurgical performance. The pilot test results are presented in Table 16-14.

Table 16-14: Pilot Plant Test Results – Mitchell, 2010 (G&T)

Test	Grind Size (P_{80} µm*)	Grade				Recovery (%)				
		Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Mass	Cu	Au	Ag	Mo
Composite PP3 (Head Assay: 0.20% Cu, 0.79 g/t Au, 3.2 g/t Ag, 0.006% Mo)										
P1	115/16	26.4	62.0	482	0.43	0.7	83.0	50.2	53.1	43.2
		25.2	62.5	382	0.26	0.6	79.2	50.9	54.8	29.0
P2	153/22	25.7	58.7	278	0.32	0.6	74.6	44.6	45.6	27.7
		26.6	69.8	295	0.45	0.5	71.2	45.9	43.9	31.4
		27.2	80.2	316	0.59	0.4	61.2	44.2	39.8	31.5
		26.9	72.3	262	0.26	0.5	69.8	43.5	40.0	22.1
P3	152/23	25.4	64.6	239	0.35	0.6	71.3	54.4	39.1	29.9
		24.3	62.4	240	0.24	0.7	79.2	52.1	49.6	28.5
		25.3	56.2	182	0.27	0.6	81.6	51.4	42.6	29.1
		25.5	58.8	220	0.32	0.6	79.3	52.9	47.2	37.1
P4	143/22	24.8	58.7	268	0.32	0.6	72.6	47.0	45.4	32.3
		26.4	63.8	280	0.33	0.7	80.3	50.8	50.1	32.8
		24.5	64.6	236	0.51	0.8	84.1	65.3	51.7	47.3
		23.6	64.7	215	0.41	0.6	81.8	56.7	44.8	41.4
Average		**25.6**	**64.2**	**278**	**0.36**	**0.6**	**76.4**	**50.7**	**46.3**	**33.1**
PP Hi-Mo Composite (Head Assay: 0.16% Cu, 0.6 g/t Au, 3.2 g/t Ag, 0.012% Mo)										
P5	163/28	22.0	52.1	244	0.31	0.7	77.8	47.3	52.5	33.1
		25.1	67.7	248	0.31	0.4	71.8	45.8	38.5	20.6
		19.3	61.8	276	0.71	0.7	81.5	66.6	59.6	41.6
		20.3	47.2	253	1.20	0.7	78.6	48.5	52.4	63.6
P6	146/21	18.9	56.7	239	0.91	0.6	78.0	54.8	49.9	43.2
		18.2	58.2	247	1.27	0.7	80.5	60.3	54.3	60.9
		20.5	57.8	246	1.21	0.6	80.1	58.3	50.3	60.6
		20.7	57.8	236	1.28	0.6	82.2	58.5	50.6	59.7
P7	143/22	19.7	67.9	259	1.27	0.6	78.9	59.7	51.5	66.8
		20.0	55.4	260	1.38	0.7	80.6	58.5	51.3	70.4
Average		**20.5**	**58.3**	**251**	**0.99**	**0.6**	**79.0**	**55.8**	**51.1**	**52.1**

* Primary Grind Size/Regrind Size

For the PP Composite 3, the pilot tests did not produce, on average, similar metallurgical performance as the locked cycle tests. On average, the copper recoveries from the PP Composite 3 ranged from 61% to 84%. The concentrate produced averaged approximately 25.6% Cu.

For the PP Hi-Mo Composite, the copper recovery reporting to the final bulk concentrate containing between 18.2% Cu to 25.1% Cu ranged from72% to 82%. The data are similar to the data generated from the locked cycle tests.

The metallurgical performance observed from the best pilot plant results was close to the results achieved in the locked cycle testing for both composites.

On average, approximately 50% of the feed silver was recovered to the copper concentrate for both composites. The average silver concentration in the concentrate was approximately 250 g/tonne.

The molybdenum recovery into the final bulk concentrate was 52% for Hi-Mo Composite and 33% for Composite PP3.

Copper-Gold and Molybdenum Separation Tests

In the 2009/2010 testing program, preliminary flotation tests were performed in an effort to produce molybdenum concentrate from copper-gold-molybdenum bulk concentrates.

The flotation separation tests were performed on the bulk concentrate produced from pilot plant tests and from bench scale open circuit tests.

The 2009 testing showed that molybdenum concentrates produced from the bulk flotation concentrate from the 2009 pilot plant tests were less than 30% Mo. G&T indicated that aging of the bulk concentrates prior to the molybdenum flotation testing was one of the potential reasons for producing the low grade molybdenum concentrates. A follow-up 20-kg bench scale test on the freshly ground Composite PP2 sample produced a 48% Mo concentrate containing 1.8% Cu.

In 2010, further copper/molybdenum separation tests were conducted on the concentrates produced from the 2010 pilot plant tests. The open circuit test achieving the best overall separation metallurgical performance produced a 51% Mo concentrate with a molybdenum recovery from the copper concentrate of 72%.

The molybdenum-copper separation locked cycle test recovered 88% of the molybdenum from the copper concentrate and produced a 41% Mo concentrate.

G&T also conducted preliminary leaching tests on the molybdenum concentrates using both the Brend-Leach procedure and hydrochloric acid leaching. The test results indicated that the copper and lead contents would be reduced respectively from 2.06% Cu to 0.26% Cu and from 0.14% Pb to 0.03% Pb. The hydrochloric acid leaching alone on a molybdenum concentrate with 1.5% Cu only reduced copper content to 0.81%.

The assay on the final molybdenum concentrates indicated that the concentrates contained approximately 2200 g/t rhenium (Re).

Cyanide Leach Tests

Most of the testing programs conducted cyanide leaching tests on the first cleaner tailings and gold-bearing pyrite concentrate or the blend of the two flotation products.

Cyanidation Tests – Products from Flotation Open Circuit Tests

A total of 30 cyanide leach tests were carried out on the gold bearing products from the flotation variability tests. Prior to the leaching, the combined first cleaner tailings and the gold-pyrite concentrate was reground to a particle size of 80% passing 9 µm to 16 µm and aerated with air for 16 hours.

The test results are summarized in Table 16-15. The average gold extraction was approximately 79%. Increasing leach retention time from 24 to 48 hours did not appear to improve gold extraction.

Table 16-15: Cyanidation Test Results – Individual Samples, Mitchell, 2008

Sample ID	Regrind Size (P_{80} µm)	Feed (g/t Au)	Extraction (% Au)	Sample ID	Regrind Size (P_{80} µm)	Feed (g/t Au)	Extraction (% Au)
48 Hour Leach Retention Time				**24 Hour Leach Retention Time**			
MET 2	11	1.7	60	MET 3	12	1.4	65
MET 5	9	1.6	79	MET 4	13	1.6	78
MET 8	9	2.2	74	MET 6	9	2.4	84
MET 11	10	6.3	94	MET 7	11	3.4	78
MET 14	15	2.7	81	MET 9	9	1.3	74
MET 17	13	1.9	87	MET 10	11	2.7	91
MET 20	11	1.1	58	MET 12	10	3.3	87
MET 23	15	1.3	82	MET 13	10	8.9	90
MET 26	13	2.7	85	MET 15	14	2.0	85
MET 29	10	4.1	83	MET 16	13	3.2	82
MET 33	16	1.9	88	MET 18	11	1.4	63
				MET 19	12	2.0	82
				MET 21	9	2.2	69
				MET 22	12	2.7	63
				MET 24	10	4.1	87
				MET 25	9	1.7	78
				MET 27	13	2.2	81
				MET 30	11	1.6	76
				MET 32	7	3.4	91
Average	**12**	**2.5**	**79**	**Average**	**11**	**2.7**	**79**

Similar tests were conducted on the products generated from the open circuit flotation tests of various composite samples. The leach retention time was 24 hours. As shown in Table 16-16, the gold extractions from the leach feeds ranged from 65 to 89% for the samples from the 2008 testing program and from 69% to 89% for the 2009 testing program. The average gold extraction was approximately 78% from the 2008 test work and 81% from the 2009 test work.

The 2009 test results also indicated that cyanide leaching kinetics was rapid. Approximately 69% of the gold was extracted within 6 hour leach retention time.

Table 16-16: Cyanidation Test Results – Composites, Mitchell, 2008/2009

Sample ID	Feed (g/t Au)	Extraction (% Au)	Sample ID	Feed (g/t Au)	Extraction (% Au)	
					6 h*	24 h*
2008 Testing Program			**2009 Testing Program**			
QSP 0-10	2.2	82	Comp 40 CL-PR	2.0	80	85
IARG 0-10	1.3	80	Comp 41 BBRX	0.4	54	86
Hi Qtz 0-10	2.3	74	Comp 42 QSP	2.1	69	78
QSP LG 0-10	1.7	74	Comp 43 CL-PR	1.5	81	89
QSP 10-30	2.3	89	Comp 44 Hi Qtz	2.1	65	77
Prop 10-30	1.6	82	Comp 45 IARG	1.7	80	81
Hi Qtz 10-30	2.0	66	Comp 46 CL-PR	1.8	73	81
QSP 0-30	2.2	78	Comp 47 QSP	1.9	48	69
Hi Qtz 0-30	1.6	65	Comp 48 QSP	2.0	71	80
Average	**1.9**	**78**	**Average**	**1.7**	**69**	**81**

* leach retention time

Cyanidation Tests – Products from Flotation Locked Cycle Tests

The first cleaner tailings and the gold-pyrite concentrate from the various locked cycle tests were cyanide leached to investigate the responses of the gold bearing products to the leaching process. The test results are summarized in Table 16-17. On average, the leach feed samples contained approximately 1.7 g/t Au and 9.6 g/t Ag. The leach tests showed that 69% of the gold and 56% of the silver were extracted from the gold bearing products. Average cyanide consumption was 2.9 kg/t.

Table 16-17: Cyanidation Test Results on LCT Test Results - Mitchell

Testing Program	Sample	Regrind Size (P_{80} μm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
G&T-2153	Master	15	1.8	67.6	9.1	62.1
G&T-2153	Master	15	2.2	73.2	10.1	64.4
G&T-2344	PP Comp 1	12	1.6	68.0		
G&T-2535	PP Comp 1	15	1.7	69.0	12.6	54.4
G&T-2535	PP Comp 1	15	1.6	81.1	10.9	54.7
G&T-2670	PP Comp 3	21	1.6	61.6		
G&T-2670	PP Comp 3	18	2.0	66.5	8.1	55.5
G&T-2670	PP Hi Mo	19	1.9	68.0	8.6	50.6
G&T-2670	BS Hi Mo	19	1.7	68.9	7.6	48.7
G&T-2897	Comp 46		1.1	63.5		
SGS	PP Comp 1	16	1.1	69.8		
Average - Mitchell		**17**	**1.7**	**68.8**	**9.6**	**55.8**

Some of the leaching tests were conducted separately on the 1st cleaner tailings and the gold bearing pyrite concentrate produced from the most recent testing programs. The test results indicated that the first cleaner tailings produced lower gold extractions, compared to the gold bearing pyrite concentrate. On average, the gold extraction from the gold bearing pyrite concentrate was 80% which is similar to the results obtained from the products of the open circuit tests. However, it appears that the first cleaner tailings generated lower gold extractions, averaging 61%.

G&T also tested the gold extraction on the 1st cleaner tailings and the gold bearing pyrite concentrate produced from the samples blended from the Mitchell zone and the other zones. The test results are provided in Table 16-18.

Table 16-18: Cyanidation Test Results on LCT Test Products, - Mitchell/Other Zones

Testing Program	Blend Sample	Regrind Size (P_{80} μm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
2670	Mitchell/Sulphurets[1]	18	1.7	61.0	5.4	51.4
2748	Mitchell/Iron Cap[2]	14	1.4	53.0		
2535	Mitchell/Kerr[3]	16	1.4	68.9	8.5	48.9
Average		**16**	**1.5**	**60.9**	**7.0**	**50.2**

1: 60% PP Comp 3 (Mitchell) + 40% Comp 49/50/51 (Sulphurets)
2: 1/3 PP Comp 1 (Mitchell) + 1/3 Iron Cap Comp 1+ 1/3 Iron Cap Comp 3
3: The composite: 80% PP Comp 1 (Mitchell) + 10% Comp 52 (Kerr) + 10% Comp 53 (Kerr).

The gold extraction of the blend sample from Mitchell zone and Iron Cap zone is much lower than the other samples.

Cyanidation Tests – Products from Pilot Plant Tests

The first cleaner tailings and gold bearing pyrite concentrate from the 2009 pilot plant runs (P3 and P5) were carbon-in-leach (CIL) tested for 24 hours. The gold extractions were 72.5% for the Test P3 product and 77.8% for the Test P5 product.

The CIL bottle roll cyanidation tests were also carried out on selected cleaner scavenger tailings and pyrite concentrate streams from the 2010 pilot plant testing. The products were tested using variable conditions of regrind sizing and target sodium cyanide concentration. The results obtained at 1000 mg/l NaCN dosage are summarized as follows:

- at an average regrind size of 80% passing 24 μm, the average gold extraction from the 1.6 g/t Au cleaner scavenger tailings was approximately 70%.
- at an average regrind size of 80% passing 20 μm, the average gold extraction from the gold bearing pyrite concentrate containing 1.9 g/t Au was approximately 77%.

Gravity Concentration Tests

Gravity Concentration Tests on Head Samples

In the 2008 testing program, ten of the drill interval samples were tested for free-gold recovery by gravity separation using centrifugal concentration (Knelson Concentrator) followed by panning. The test results are shown in Table 16-19.

Table 16-19: Gravity Separation Test Results, - Mitchell

Sample ID	Pan Concentrate		Knelson Concentrate	
	Grade (g/t Au)	Distribution (%)	Grade (g/t Au)	Distribution (%)
MET 4	231	55	103	61
MET 7	28	9	25	13
MET 10	3	6	4	19
MET 14	27	8	17	11
MET 16	50	17	33	20
MET 18	22	7	13	9
MET 19	15	15	11	20
MET 23	13	12	6	16
MET 29	44	6	11	10
MET 32	20	8	11	11
Average	**45**	**14**	**23**	**19**

On average, approximately 19% of the gold in the samples was recovered to the Knelson concentrate with an average grade of 23 g/t Au.

Most of the pan concentrates contained less than 50 g/t Au with a gold recovery of less than 17%, except for the MET 4 sample. Panning produced a 231 g/t Au concentrate and recovered 55% of the gold from the MET 4 sample.

Gravity Concentration Tests on Tailing Samples

G&T carried out a few of centrifugal gravity concentration tests to recover gold bearing minerals from flotation tailings. The test results show that the concentration was able to recover some of the gold in the tailings. Due to a poor match between the calculated gold and measured gold in the feeds, further tests should be conducted to confirm the findings.

16.2.2 Sulphurets Mineralization

Test Samples

Three composite samples were compiled from the crushed drill cores to investigate the metallurgical responses of Sulphurets mineralization. The chemical assay of these composites is presented in Table 16-20.

Table 16-20: Metal Contents of Composites – Sulphurets, 2009 (G&T)

Composite	Mineralization Type*	Metal Content						
		Cu (T) (%)	Cu (ox) (%)	Cu (CN) (%)	Au (T) (g/t)	Au (CN) (g/t)	Mo (%)	Ag (g/t)
Comp 49	Hazelton Volcanics	0.14	0.016	0.016	0.26	0.002	0.003	1.9
Comp 50	Raewyn Copper	0.26	0.007	0.012	0.66	0.006	0.005	1.2
Comp 51	Raewyn Copper	0.37	0.005	0.013	0.81	0.007	0.011	1.4

* **Hazelton Volcanics:** Propylitic altered (quartz, chlorite, pyrite) volcanics and sediments of the Main Copper zone (above Sulphurets Fault)
Raewyn Copper: Propylitic altered volcanics and sediments of the Sulphurets zone (beneath Sulphurets Fault); selected intervals are within crackled, veined, and brecciated transitional zone beneath the Gold Breccia zone, and have higher than average gold grades.

Mineralization Hardness

The test results, as presented in Table 16-21, indicate that the Sulphurets samples are more resistant to ball mill grinding compared to the Mitchell samples. The average Bond ball work index is 19 kWh/t for the Sulphurets samples; the Ai of the overall Sulphurets composite is 0.233 g.

Table 16-21: Bond Ball Mill Work Index Test Results - Sulphurets, 2009-2010)

Samples	Wi (kWh/t)	Ai (g)
2009 G&T		
Composite 49	15.8	
Composite 50	20.8	
Composite 51	19.8	
Sub Average	18.8	
2009/2010 SGS		
Composite	19.1	0.233
Total Average	**19.0**	

In 2009 SGS conducted bench scale HPGR tests on the Sulphurets composite samples. The tests included batch open circuit tests and locked cycle tests. The test results indicate that the Sulphurets mineralization is harder to HPGR crushing than the Mitchell mineralization. On average, the net specific energy requirement is 3.08 kWh/t for the Sulphurets sample compared to 2.33 kWh/t for the Mitchell sample. The locked cycle test results, including specific grinding force (N/mm^2) and specific throughput rate (ts/hm^3-(m_c)) are summarized previously in Table 16-10.

Flotation Tests

The test conditions developed from the Mitchell samples were used to test the metallurgical performance of the Sulphurets samples. The flotation open cycle test results produced were similar to these from the Mitchell samples. Table 16-22 summarizes the flotation locked cycle test results.

Table 16-22: Locked Cycle Test Results – Sulphurets

Test Program	Comp	Product	Grind Size* (P_{80} µm)	Mass	Grade (% or g/t)				Flotation Recovery, %			
				(%)	Cu	Au	Ag	Mo	Cu	Au	Ag	Mo
SGS	Composite	Head	125/20	100.0	0.20	0.66		0.007	100.0	100.0		100.0
		Cu/Mo Concentrate		0.75	22.7	49.1		0.630	85.7	56.1		66.6
		Bulk Cleaner Tailings + Au-Pyrite Concentrate		17.3	0.08	1.31		0.008	6.7	34.3		20.3
G&T 2670/44	Master Composite (Comp49/50/51)	Head	154/16	100.0	0.24	0.52	1.6	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.7	28.3	41.8	82	0.701	80.5	53.9	34.3	72.2
		Bulk Cleaner Tailings		6.3	0.13	1.94		0.016	3.5	23.5		15.1
		Au-Pyrite Concentrate		2.9	0.38	1.41		0.013	4.7	7.9		5.7
G&T 2897/22	Master Composite (Comp49/50/51)	Head	113/-	100.0	0.24	0.50	1.5	0.008	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.7	28.4	41.6	71.0	0.850	79.4	55.6	31.5	68.5
		Bulk Cleaner Tailings		6.3	0.17	1.82	4.2	0.013	4.5	23.0	17.5	9.9
		Au-Pyrite Concentrate		4.0	0.35	1.15	3.5	0.011	6.0	9.3	9.5	5.4

* Primary Grind Size/Regrind Size

SGS produced a higher copper recovery, averaging 85.7%, compared to approximately 80% by G&T. However, the concentrate grade produced by G&T is 28.4% Cu, compared to the copper content of 22.7% generated by SGS. The average gold recovery was approximately 55%, while the silver recovery was low, averaging 33%. Although the molybdenum head grade was approximately 0.007%, the molybdenum reporting to the bulk concentrate was high, ranging from 66% to 72%.

Further test work, including variability testing, is recommended to better define the metallurgical performance of the Sulphurets mineralization.

Cyanide Leach Tests

SGS conducted the cyanidation tests on the blend of the first cleaner tailings and the gold-pyrite concentrate from the locked cycle test. The test results are summarized in Table 16-23. By the carbon-in-leach procedure, the leaching tests showed that 70.5% of the gold was extracted from the gold bearing products containing approximately 1.5 g/t Au. Average cyanide consumption was 1.2 kg/t. The direct cyanide leach test produced inferior results.

Table 16-23: Locked Cyanidation Test Results – Flotation LCT Products, Sulphurets, 2009-2010 (SGS)

Test Method	Sample ID	Leach Head (Au g/t)	Gold Extraction (%)	Cyanide Consumption (kg/t)
DCN (Bottle-on-Roll)	LCT 2 – Cycle F	1.60	51.5	1.4
CIL (Bottle-on-Roll)	LCT 2 – Cycle E	1.34	70.5	1.2

16.2.3 Kerr Mineralization

Test Samples

Two composite samples from the Kerr zone, identified as Composites 52 and 53 were prepared for metallurgical testing. The samples were prepared from the drill core intervals obtained in late 2009. The metal assays in the composites are presented in Table 16-24.

Table 16-24: Metal Contents of Composites – Kerr, 2010 (G&T)

Composite	Mineralization Type*	Metal Content			
		Cu	Au	Mo	Ag
		(%)	(g/t)	(%)	(g/t)
Comp 52	Rubble Zone	0.59	0.22	0.004	2.0
Comp 53	Quartz Stockwork	0.61	0.17	0.001	1.5

- **Rubble Zone:** Quartz, sericite, chlorite, pyrite altered rocks with anhydrite ±gypsum veinlets, secondary chalcocite coatings, poor rock quality.
  Quartz Stockwork: Quartz, sericite, chlorite, pyrite altered rocks with crackled quartz stockwork veinlets, mylonitized, relatively competent.

Mineralization Hardness

The samples from the Kerr deposit are softer to ball mill grinding when compared to the Mitchell and Sulphurets mineralization. As shown in Table 16-25, the average Bond ball mill work index is 13.4 kWh/t. These results agree with the historical test results.

Table 16-25: Bond Ball Mill Work Index Test Results – Kerr, 2010 (G&T)

Samples	Wi (kWh/t)
Composite 52	13.8
Composite 53	13.0
Average	**13.4**

Flotation Tests

The test conditions used for the Mitchell and Sulphurets samples were also used for the two composite samples collected from the Kerr deposit. The open circuit batch flotation tests showed that the Kerr samples produced better concentrate grades than the Mitchell or Sulphurets samples. Copper recovery produced was slightly lower than the Mitchell or Sulphurets samples at equivalent copper concentrate tenor. Gold recovery for the Kerr samples was lower because the gold head grades were considerably lower than the samples from the other two ore deposits.

The locked cycle test results, presented in Table 16-26, indicate that the metallurgical performance of the Kerr samples was not as good as that achieved with the Mitchell and Sulphurets samples despite their lower copper head grades.

Table 16-26: Locked Cycle Test Results – Kerr, 2010 (G&T)

Test Program	Comp	Product	Grind Size* (P_{80} µm)	Mass	Grade (% or g/t)			Flotation Recovery (%)		
				(%)	Cu	Au	Ag	Cu	Au	Ag
G&T 2535/16	Comp 52	Head	119/15	100.0	0.59	0.22	1.9	100.0	100.0	100.0
		Cu/Mo Con		2.1	22.30	4.05	33.5	81.6	38.8	37.6
		Bulk Clean		7.9	0.43	0.97	6.3	5.7	34.2	26.0
		Au-Pyrite		7.7	0.39	0.62	4.2	5.2	21.5	17.0
G&T 2535/17	Comp 53	Head	122/14	100.0	0.62	0.25	1.4	100.0	100.0	100.0
		Cu/Mo Con		1.7	29.30	5.58	31.8	80.6	37.7	37.9
		Bulk Clean		6.8	0.40	0.51	3.6	4.5	13.8	17.5
		Au-Pyrite		13.6	0.42	0.66	3.0	9.1	36.0	28.2

* Primary Grind Size/Regrind Size

On average, the Kerr samples produced a 25.8% Cu concentrate. The copper and the gold reporting to the concentrate were 81% and 38%, respectively. Approximately 53% of the gold reported to the gold bearing pyrite products (first cleaner tailings and gold bearing pyrite concentrate).

Leach Tests

G&T conducted the cyanidation tests on the first cleaner tailings and the gold bearing pyrite concentrate produced from the locked cycle tests. The leach procedure was the same as that used previously on the Mitchell samples. Test results are provided in Table 16-27.

Table 16-27: Cyanidation Test Results on LCT Test Products – Kerr, 2010 (G&T)

Testing Program	Sample	Regrind Size (P_{80} µm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
G&T-2535	Comp 52	17	1.1	76.0	5.5	45.8
G&T 2535	Comp 53	15	0.6	59.7	3.2	18.7
Average - Kerr		**16**	**0.9**	**67.8**	**9.6**	**32.3**

On average, the gold extraction from both the gold bearing products was approximately 68% similar to the results obtained from the Mitchell samples. The average gold feed grade to the cyanide leach circuit was lower in comparison with the cyanide leach feeds of the Mitchell samples. The test results also indicated that the first cleaner tailings produced slightly lower gold and silver recoveries compared to the gold bearing pyrite concentrate. The average silver extraction was 32% which was lower than the average extraction of 56% obtained from the Mitchell samples.

16.2.4 Iron Cap Mineralization

Test Samples

The 2010 test work also conducted metallurgical testing on two composite samples generated from a total of 168 samples weighing a total of approximately 689 kg. The assay of the head samples are provided in Table 16-28.

Table 16-28: Metal Contents of Composites – Iron Cap, 2010 (G&T)

Composite	Metal Content						
	Cu (T) (%)	Cu (ox) (%)	Cu (CN) (%)	Au (g/t)	Mo (%)	Ag (g/t)	S (%)
IC 2010 Composite 1	0.14	0.001	0.015	1.06	0.002	6	4.5
IC 2010 Composite 2	0.36	0.004	0.023	0.32	0.003	5	3.6
Iron Cap Blend	0.25			0.75	0.003		3.7

Mineralogy

The mineral content, in each of the two master composites, was determined using the Bulk Mineral Analysis with Liberation (BMAL) function within QEMSCAN. The results of the BMAL analysis indicated that:

- Both composites analyzed contained about 6% to 8% sulphide minerals. The dominant sulphide mineral present was pyrite. The balance of each sample, about 93%, was comprised of nonsulphide gangue minerals consisting of quartz, feldspar, and muscovite.

- Copper is mostly contained in chalcopyrite. Composite 1 also contained chalcocite/covellite and tennantite/tetrahedrite at approximately 4% and 5% of the feed copper respectively.

Mineralization Hardness

The grindability determination tests on the two composite samples from the Iron Cap deposit showed that the mineralization is of moderate hardness to ball mill grinding. The Bond ball mill work indices of both the samples are 14.9 kWh/t.

Flotation Tests

The test conditions used for the Mitchell samples were tested for the two composite samples from the Iron Cap deposit. The open circuit batch flotation tests showed that the Iron Cap mineralization was not sensitive to the primary grind sizes ranging from 80% passing 120 μm to 170 μm.

The flotation locked cycle test results are presented in Table 16.29. On average, the mineralization produced a 25.7% Cu concentrate. The copper and the gold reporting to the concentrate were 85% and 51%, respectively. On average, approximately 39% of the gold reported to the gold bearing pyrite products (first cleaner tailings and gold bearing pyrite concentrate).

Table 16-29: Locked Cycle Test Results - Iron Cap

Test Program	Comp	Product	Grind Size* (P_{80} µm)	Mass	Grade (% or g/t)				Flotation Recovery, %			
				(%)	Cu	Au	Ag	Mo	Cu	Au	Ag	Mo
G&T 2748/11	Iron Cap 2010 Composite1	Head	150/15	100.0	0.14	1.28	6	0.002	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.5	25.4	147	774	0.180	81.6	55.2	61.0	37.9
		Bulk Cleaner Tailings		10.4	0.06	2.17	11.6	0.004	3.8	17.6	20.1	18.0
		Au-Pyrite Concentrate		7.9	0.11	1.88	6.6	0.002	5.9	11.7	8.7	8.6
G&T 2748/12	Iron Cap 2010 Composite2	Head	147/22	100.0	0.38	0.31	5	0.003	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.3	24.9	10	255	0.115	88.1	45.0	62.0	55.2
		Bulk Cleaner Tailings		10.5	0.06	1.21	7.9	0.003	1.7	40.7	16.6	11.2
		Au-Pyrite Concentrate		6	0.25	0.57	5.2	0.002	4	11.1	6.2	4.3
G&T 2748/17	50%Comp 1: 50%Comp 2	Head	108/19	100.0	0.26	0.82		0.003	100.0	100.0		100.0
		Cu/Mo Concentrate		0.8	26.7	51.9		0.144	85.2	53.3		41.5
		Bulk Cleaner Tailings		10.9	0.06	1.82		0.005	2.4	24.2		17.7
		Au-Pyrite Concentrate		7.3	0.16	1.37		0.003	4.4	12.1		6.2

* Primary Grind Size/Regrind Size

The results indicated that the copper recoveries from both the Iron Cap samples were comparable to the Mitchell mineralization. It appeared that the gold recoveries to the concentrate were lower than these achieved with the Mitchell mineralization; however, the silver recoveries were higher. Approximately 38% and 55% of the molybdenum from the two samples reported to the final bulk concentrate.

Cyanide Leach Tests

G&T conducted the cyanidation tests on the first cleaner tailings and the gold bearing pyrite concentrate produced from the locked cycle tests. The leach procedure used was developed from the Mitchell samples. Test results are provided in Table 16-30.

Table 16-30: Cyanidation Test Results on LCT Test Products - Iron Cap

Testing Program	Sample	Regrind Size (P_{80} µm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
G&T-2748	Iron Cap Comp 1	14	1.9	49.7	9.4	62.8
G&T-2748	Iron Cap Comp 2	15	1.1	40.4	6.9	56.8
G&T-2748	50% Comp 1/50% Comp 2	16	1.5	48.6		
Average – Iron Cap		**15**	**1.5**	**46.2**	**8.2**	**59.8**

On average, the gold extraction from both the gold bearing products was approximately 46%. The test results also indicated that both the first cleaner tailings and the gold bearing pyrite concentrate produced lower gold recoveries compared to the other mineralization, especially the first cleaner tailings. The average gold feed grade to the cyanide leach circuit was lower in comparison with the cyanide leach feeds of the Mitchell samples. The average silver extraction was high, averaging 60% which is slightly higher than the average extraction of 56% obtained the Mitchell samples.

The mineralogical study by Surface Science Western on the leaching residues found that the gold is present in colloidal type sub-microscopic gold, mainly in pyrite which occurs in coarse and porous types. Surface Science Western pointed out that the pre-treatment by pressure or biooxidation would be required to release the locked gold.

16.2.5 Flotation Concentrate Assay

The multi-element assay data are provided in Table 16-31 for the concentrates from the Mitchell deposit and Table 16-32 for the concentrates from the other deposits. On average, the impurities in the copper-gold concentrates produced from the Mitchell, Sulphurets and Kerr deposits should not attract smelting penalties as set out by most smelters.

Table 16-31: Multi-Element Assay – Mitchell Concentrate (1)

Element	Unit	Mitchell							Average
		2153/142 Master Comp	2344/73 Comp PP1	2535/18 Comp PP1	2535/20 Comp PP1	SGS/LCT1 Comp PP1	2670/18[2] Comp PP3	2670/Pilot Plant Comp PP3	
Cu	%	22.0	22.3	28	23.8	23.1	27.4	25.7	24.6
Au	g/t	64.7	55.7	77.8	62.0	53.7	70.5	65.5	64.3
Ag	g/t	257	-	260	248	-	275	304	269
Mo	%	-	0.23	0.12	0.12	0.41	0.62	0.33	0.31
S (T)	%	33.4	34.4	34.7	32.9	38.1	34.5	31.1	34.2
S (-2)	%	-	-	32.9	32.1	-	33.3	28.7	31.8
Fe	%	26.8	30.8	29.6	30.7	32.7	30.1	27.6	29.8
Sb	ppm	696	698	539	597	-	466	338	556
As	ppm	1,184	934	824	878	-	1174	821	969
Co	ppm	48	76	52	52	-	68	56	58.7
Cd	ppm	72	44	60	84	-	88	80	71
Bi	ppm	36	43	150	127	-	<10	<10	63
Hg	ppm	0.6	<1	<1	<1	-	1	<1	<1
Ni	ppm	120	240	112	156	-	48	80	126
F	ppm	346	150	100	148	-	89	230	177
Cl	ppm						<0.01	<0.01	
Se	ppm	72	102	82	70	-	73	70	78
P	ppm	230	215	146	189	-	55	492	221
Pb	%	0.92	0.19	0.19	0.22	-	0.32	0.23	0.34
Zn	%	0.42	0.23	0.25	0.38	-	0.43	0.32	0.34
SiO_2	%	9.84	6.67	2.39	7.11	-	3.04	8.23	6.21
CaO	%	0.54	0.53	0.39	0.54	-	0.27	0.74	0.50
Al_2O_3	%	3.31	1.76	0.62	1.37	-	0.57	1.83	1.58
MgO	%	0.48	0.36	0.18	0.34	-	0.15	0.47	0.33
MnO	%	0.02	0.03	0.011	0.026	-	0.015	0.035	0.023
Insol	%	-	8.46	4.02	8.87	'-	3.23	10.3	6.98

(1) Copper-gold/molybdenum concentrate before molybdenum separation
(2) Testing program and Test ID

However, arsenic (As) and antimony (Sb) contents of the concentrates from the Iron Cap deposit and arsenic content of the concentrate from the Comp 53 of the Kerr sample may attract smelting penalties. Also the lead (Pb) content of the concentrate from the Iron Cap Comp 1 may be higher than the penalty thresholds. Fluorine (F) levels in some of the concentrates may be also higher than the penalty thresholds. It is anticipated that the Iron Cap ore and the Kerr ore will be processed together with the ores from the Mitchell deposit. Impurities in the copper concentrates produced from these blended ores should be further reviewed with respect to smelting penalties.

Table 16-32: Multi-Element Assay – Sulphurets/Kerr/Iron Cap/Blend Concentrate [1]

Element	Unit	Sulphurets	Sulph/Mit	Iron Cap		Kerr		Mitchell/Kerr
		2670/44	2670/62	2748/11	2748/12	2535/16	2535/17	2535/19[2]
		Comp	Blend	Comp 1	Comp 2	Comp 52	Comp 53	Blend
Cu	%	28.3	24.2	25.4	24.9	22.3	29.3	25.3
Au	g/t	41.8	52.0	146.8	10.9	4.1	5.6	40.0
Ag	g/t	82.0	178.0	774.0	1.3	33.5	31.8	168.0
Mo	%	0.70	0.66	0.18	0.12	0.01	0.02	0.056
S (T)	%	33.6	34.9	32.6	33.5	27.1	35.3	35.0
S (-2)	%	31.2	32.2	32.4	32.2	25.9	33.8	33.4
Fe	%	29.6	30.0	26.5	27.8	23.7	27.5	29.3
Sb	ppm	445.0	500.0	4379.0	2876.0	24.0	121.0	492.0
As	ppm	224.0	969.0	3067.0	1107.0	143.0	3276.0	1369.0
Co	ppm	92.0	104.0	50.0	68.0	40.0	52.0	68.0
Cd	ppm	180.0	144.0	320.0	128.0	20.0	8.0	80.0
Bi	ppm	<10	<10	205.0	164.0	95.0	105.0	121.0
Hg	ppm	2.0	1.0	<1	2.0	3.4	12.0	2.4
Ni	ppm	88.0	96.0	50.0	88.0	132.0	168.0	164.0
F	ppm	155.0	174.0	162.0	494.0	320.0	88.0	116.0
Cl	ppm	<0.01	<0.01	<0.01	<0.01			
Se	ppm	118.0	89.0	180.0	108.0	140.0	109.0	76.0
P	ppm	92.0	113.0	143.0	135.0	1045.0	233.0	224.0
Pb	%	0.26	0.26	1.31	0.43	0.03	0.05	0.15
Zn	%	0.54	0.92	2.29	1.02	0.30	0.10	0.42
SiO2	%	4.14	5.82	3.16	5.59	14.00	3.90	5.12
CaO	%	0.41	0.38	0.34	0.29	0.83	0.17	0.43
Al2O3	%	0.92	1.18	0.85	1.28	3.92	0.85	0.99
MgO	%	0.25	0.29	0.12	0.18	0.70	0.14	0.26
MnO	%	0.017	0.022	0.011	0.017	0.050	0.015	0.018
Insol	%	4.90	7.21	5.15	7.66	19.60	5.42	6.67

(1) Copper-gold/molybdenum concentrate before molybdenum separation
(2) Testing program and Test ID

16.2.6 **Ancillary Tests**

The testing programs also conducted various environment related testing and determined engineering related parameters. The key tests are listed below:

- leach residue cyanide destruction, including sulphur dioxide (SO_2)/air, Caro's acid (H_2SO_5), and hydrogen peroxide (H_2O_2) process

- cyanide recovery from barren solutions, including AVR process (Acidification, Volatilization of HCN gas and Re-neutralization) and SART process (Sulphidization, Acidification, Recycling of precipitate and Thickening of precipitate)

- static and dynamic thickening tests for conventional thickener sizing and for high rate thickener sizing for primary grinding product, first cleaner tailings + gold bearing pyrite concentrate, cyanidation residues and rougher/scavenger flotation tailings.

- filtration testing, including: vacuum filtration and pressure filtration for bulk flotation concentrate.

16.3 **Conclusions**

The substantial test results indicate that the mineral samples from the four separate mineralization deposits are amenable to the flotation-cyanidation combined process. The process consists of:

- copper-gold-molybdenum bulk rougher flotation followed by gold-bearing pyrite flotation

- regrinding the resulting bulk rougher concentrate followed by three stages of cleaner flotation to produce a copper-gold-molybdenum bulk cleaner flotation concentrate

- molybdenum separation of the bulk cleaner flotation concentrate to produce a molybdenum concentrate and a copper/gold concentrate containing associated silver

- cyanide leaching of the gold-bearing pyrite flotation concentrate and the scavenger cleaner tailing to further recover gold and silver values as dore bullion.

The samples from the Mitchell and Sulphurets deposits produced better metallurgical results with the chosen flotation circuit and cyanide leach extraction when compared to the metallurgical results from the samples taken from the Iron Cap and Kerr deposits.

16.4 **Recommended Test Work**

Wardrop recommends further metallurgical test work to optimize process conditions and to establish design-related parameters for the next stage of study. Wardrop makes the following recommendations:

- Additional metallurgical test work and mineralogical evaluations should be conducted to optimize process conditions and to establish design-related parameters for the next stage of study. The test work should include variability testing of samples from Sulphurets, Kerr and Iron Cap zones. The cost of the test work is estimated at $500,000.

- Further investigation of the separation between copper and molybdenum from the bulk concentrate should be included in the next study phase. The potential additional value of rhenium in the molybdenum concentrate should be evaluated ($150,000).

- Further study should be conducted to optimize the proposed cyanide recovery and destruction methods ($100,000).

- Test work to confirm the slurry pumping arrangement to deliver the ore slurry from mine site and plant site should be conducted to confirm the current preliminary design ($200,000).

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Mineral Resources were estimated for the KSM project by Mr. Michael J. Lechner, President of Resource Modeling Inc. (RMI). Mr. Lechner is a P. Geo. (British Columbia), a Registered Professional Geologist in the State of Arizona, is a Certified Professional Geologist with the AIPG, and a registered member of SME. These professional registrations together with Mr. Lechner's professional background and work experience allow him to be the Qualified Person for this report as per the requirements as set out by NI 43-101. Neither Mr. Lechner nor RMI have any vested interest in Seabridge Gold securities or the property that is the subject of this technical report. Mr. Lechner and RMI have worked as an independent consultant for Seabridge Gold since 2001.

The Kerr, Sulphurets, Mitchell, and Iron Cap resource models were updated by RMI by combining newly acquired drill hole data and geologic interpretations with previously collected information. Various statistical analyses were updated and new grade models constructed.

17.1 **Gold Grade Distribution**

Block gold grades were estimated by assay grades that were composited into 15-meter-long drill hole composites after high-grade outlier values were capped. Section 17.3 discusses grade capping. Various geologic wireframes were used to constrain the estimate of block grades for each zone. These geologic wireframes represent either distinct alteration types (e.g. Kerr and Iron Cap) or a combination of alteration/lithology and gold grade (Sulphurets and Mitchell).

The distribution of gold based on the composited data used to estimate block grades is summarized at four different cutoff grades by the geologic constraint that was used in the estimation process in Tables 17-1 through 17-4 for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively.

As can be seen Tables 17-1 through 17-4 the average gold grade increases going from the Kerr deposit in the southern part of the district to the Sulphurets (middle portion of district) to the Mitchell deposit in the north. The average gold grade of the Iron Cap zone is between the mean grade of the Sulphurets and Mitchell zones. In addition to the gold grade increasing from south to north the percentage of material above a 0.50 g/t gold cutoff also increases from Kerr (6%) to Sulphurets (29%) to Mitchell (44%). The percentage of Iron Cap gold grades above 0.50 g/t is 22%. Another important statistical parameter is that the coefficient of variation (CV) is below 1.00 for all for mineralized zones. The CV for Mitchell gold grades based on composited data is 0.69. CV's less than 1.0 indicates that the gold assay population contains few high-grade outliers and that local grade estimation should be feasible.

In general, it has not been possible to identify any particular lithologic unit or alteration type that adequately defines a mineralized gold population for any of the KSM mineralized zones except for the Kerr deposit. Quartz-sericite-pyrite alteration tends to be one of the key mineralized units but gold grades are seen to cross cut the various logged alteration types. Given these observations, RMI elected to use grade envelopes to constrain the estimate of block gold grades (AUZON). Mineral zones and constraints used to estimate block grades are discussed in Section 17.5.

Table 17-1: Distribution of Gold by Alteration - Kerr

Alteration Type	Uncapped Au Statistics Above Cutoff								Capped Au Statistics Above Cutoff				
	Au Cutoff (g/t)	Total Meters	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	26,871	73%	0.22	5,879	44.7%	0.23	1.04	0.22	5,814	45.2%	0.21	0.98
All Data	0.25	7,170	20%	0.45	3,253	31.8%	0.33	0.72	0.44	3,188	32.2%	0.29	0.66
All Data	0.50	1,664	5%	0.83	1,382	15.5%	0.51	0.62	0.79	1,317	16.2%	0.44	0.56
All Data	1.00	274	1%	1.71	469	8.0%	0.76	0.45	1.59	378	6.5%	0.73	0.46
Chlorite-Propylytic	0.00	10,309	61%	0.25	2,535	36.8%	0.16	0.64	0.25	2,535	36.8%	0.16	0.64
Chlorite-Propylytic	0.25	4,061	33%	0.39	1,602	45.3%	0.14	0.37	0.39	1,602	45.3%	0.14	0.37
Chlorite-Propylytic	0.50	695	7%	0.65	454	17.9%	0.14	0.22	0.65	454	17.9%	0.14	0.22
Chlorite-Propylytic	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
QSP	0.00	9,869	84%	0.18	1,812	61.8%	0.18	1.01	0.18	1,799	62.3%	0.17	0.92
QSP	0.25	1,584	13%	0.44	692	23.0%	0.34	0.79	0.43	678	23.2%	0.29	0.69
QSP	0.50	305	2%	0.90	275	8.2%	0.58	0.64	0.86	261	8.2%	0.45	0.53
QSP	1.00	75	1%	1.69	127	7.0%	0.69	0.41	1.51	113	6.3%	0.48	0.32
Wk. CLQSP	0.00	1,928	73%	0.22	432	40.0%	0.23	1.02	0.22	427	40.5%	0.21	0.96
Wk. CLQSP	0.25	518	17%	0.50	259	27.0%	0.28	0.56	0.49	254	27.3%	0.24	0.49
Wk. CLQSP	0.50	188	8%	0.76	142	24.1%	0.32	0.42	0.73	137	24.4%	0.25	0.34
Wk. CLQSP	1.00	27	1%	1.41	38	8.8%	0.33	0.23	1.23	33	7.8%	0.16	0.13
Premier Dyke	0.00	423	83%	0.18	78	54.5%	0.22	1.19	0.18	78	54.5%	0.22	1.19
Premier Dyke	0.25	71	14%	0.50	36	25.9%	0.38	0.76	0.50	36	25.9%	0.38	0.76
Premier Dyke	0.50	11	0%	1.39	15	0.0%	0.00	0.00	1.39	15	0.0%	0.00	0.00
Premier Dyke	1.00	11	3%	1.39	15	19.6%	0.00	0.00	1.39	15	19.6%	0.00	0.00
Hornblende Dyke	0.00	359	96%	0.07	26	81.6%	0.06	0.86	0.07	26	81.6%	0.06	0.86
Hornblende Dyke	0.25	15	4%	0.32	5	18.4%	0.00	0.00	0.32	5	18.4%	0.00	0.00
Hornblende Dyke	0.50	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Hornblende Dyke	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Undefined	0.00	3,983	77%	0.25	996	33.8%	0.40	1.61	0.24	950	35.5%	0.36	1.53
Undefined	0.25	921	11%	0.72	659	16.5%	0.63	0.88	0.67	613	17.3%	0.56	0.85
Undefined	0.50	466	8%	1.06	495	20.8%	0.74	0.69	0.96	449	24.5%	0.67	0.69
Undefined	1.00	161	4%	1.79	288	28.9%	0.85	0.47	1.74	216	22.7%	0.90	0.52

Table 17-2: Distribution of Gold by AUZON – Sulphurets

AUZON	Uncapped Au Statistics Above Cutoff								Capped Au Statistics Above Cutoff				
	Au Cutoff (g/t)	Total Meters	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	23,376	43%	0.42	9,868	13.3%	0.45	1.05	0.41	9,680	13.6%	0.40	0.96
	0.25	13,379	28%	0.64	8,554	24.6%	0.48	0.75	0.63	8,366	25.1%	0.41	0.66
	0.50	6,809	22%	0.90	6,122	36.8%	0.56	0.62	0.87	5,935	37.6%	0.46	0.52
	1.00	1,579	7%	1.58	2,491	25.2%	0.83	0.53	1.47	2,296	23.7%	0.61	0.42
Lower Au Zone	0.00	2,190	15%	0.56	1,223	5.7%	0.44	0.78	0.53	1,152	6.1%	0.31	0.58
	0.25	1,853	43%	0.62	1,153	29.5%	0.45	0.72	0.58	1,082	31.3%	0.30	0.51
	0.50	911	32%	0.87	793	37.6%	0.53	0.61	0.79	721	41.3%	0.30	0.38
	1.00	213	10%	1.56	332	27.2%	0.72	0.46	1.24	246	21.4%	0.29	0.23
Au Leach Zone	0.00	926	2%	1.21	1,119	0.3%	0.91	0.75	1.19	1,102	0.3%	0.82	0.69
	0.25	905	9%	1.23	1,116	3.0%	0.90	0.73	1.21	1,098	3.0%	0.82	0.67
	0.50	819	45%	1.32	1,083	29.8%	0.90	0.68	1.30	1,065	30.3%	0.81	0.62
	1.00	401	43%	1.87	749	66.9%	1.03	0.55	1.82	731	66.4%	0.88	0.49
Au Breccia Zone	0.00	1,008	33%	0.46	465	11.9%	0.39	0.84	0.46	460	11.9%	0.38	0.84
	0.25	678	38%	0.60	410	31.7%	0.40	0.66	0.60	406	32.0%	0.39	0.66
	0.50	291	24%	0.90	262	37.1%	0.45	0.50	0.89	258	36.7%	0.45	0.51
	1.00	51	5%	1.75	90	19.3%	0.45	0.26	1.74	89	19.4%	0.45	0.26
Canyon Zone	0.00	868	0%	0.71	614	0.0%	0.33	0.46	0.70	612	0.0%	0.33	0.47
	0.25	868	35%	0.71	614	19.7%	0.33	0.46	0.70	612	19.7%	0.33	0.47
	0.50	568	47%	0.87	493	46.7%	0.30	0.34	0.86	491	46.6%	0.30	0.35
	1.00	162	19%	1.27	206	33.6%	0.17	0.14	1.27	206	33.7%	0.17	0.14
Raewyn Copper	0.00	5,319	10%	0.60	3,186	2.9%	0.34	0.57	0.59	3,156	2.9%	0.32	0.54
	0.25	4,803	37%	0.64	3,093	24.7%	0.33	0.51	0.64	3,063	24.9%	0.30	0.47
	0.50	2,828	44%	0.82	2,307	50.7%	0.32	0.40	0.81	2,278	51.1%	0.29	0.36
	1.00	511	10%	1.35	692	21.7%	0.39	0.29	1.30	665	21.1%	0.30	0.23
Main Copper	0.00	1,305	90%	0.14	178	61.0%	0.19	1.36	0.14	178	61.0%	0.19	1.36
	0.25	126	7%	0.55	70	19.7%	0.37	0.67	0.55	70	19.7%	0.37	0.67
	0.50	29	1%	1.20	34	7.5%	0.21	0.18	1.20	34	7.5%	0.21	0.18
	1.00	15	1%	1.40	21	11.8%	0.00	0.00	1.40	21	11.8%	0.00	0.00
Undefined	0.00	11,761	65%	0.26	3,083	31.9%	0.33	1.26	0.26	3,020	32.6%	0.27	1.05
	0.25	4,146	24%	0.51	2,099	30.8%	0.46	0.91	0.49	2,036	31.4%	0.34	0.68
	0.50	1,363	10%	0.84	1,149	24.3%	0.68	0.81	0.80	1,087	24.8%	0.44	0.55
	1.00	226	2%	1.77	400	13.0%	1.30	0.73	1.49	337	11.2%	0.71	0.48

Table 17-3: Distribution of Gold by AUZON – Mitchell

AUZON	Uncapped Au Statistics Above Cutoff								Capped Au Statistics Above Cutoff				
	Au Cutoff (g/t)	Total Meters	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	49,732	30%	0.49	24,179	7.7%	0.35	0.72	0.48	23,991	7.7%	0.33	0.69
	0.25	35,039	27%	0.64	22,329	20.6%	0.31	0.49	0.63	22,141	20.8%	0.28	0.44
	0.50	21,833	37%	0.79	17,345	54.0%	0.29	0.37	0.79	17,157	54.5%	0.24	0.31
	1.00	3,370	7%	1.27	4,292	17.7%	0.43	0.34	1.22	4,077	17.0%	0.24	0.20
Upper Plate	0.00	1,583	64%	0.23	358	37.7%	0.17	0.77	0.22	356	37.9%	0.17	0.76
	0.25	564	27%	0.39	223	36.6%	0.19	0.47	0.39	221	36.8%	0.18	0.45
	0.50	135	8%	0.68	92	21.1%	0.18	0.27	0.67	90	25.3%	0.16	0.24
	1.00	15	1%	1.08	16	4.5%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Lower Plate	0.00	35,832	9%	0.61	21,855	2.8%	0.32	0.53	0.61	21,689	2.9%	0.30	0.49
	0.25	32,517	31%	0.65	21,236	19.8%	0.31	0.47	0.65	21,070	20.0%	0.28	0.43
	0.50	21,248	50%	0.80	16,904	58.2%	0.29	0.37	0.79	16,739	58.7%	0.24	0.31
	1.00	3,300	9%	1.27	4,195	19.2%	0.43	0.34	1.22	4,015	18.5%	0.24	0.20
Bornite Breccia	0.00	150	70%	0.27	40	55.0%	0.12	0.45	0.27	40	55.0%	0.12	0.45
	0.25	45	20%	0.40	18	22.6%	0.14	0.35	0.40	18	22.6%	0.14	0.35
	0.50	15	10%	0.60	9	22.4%	0.00	0.00	0.60	9	22.4%	0.00	0.00
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Bornite Leach Halo	0.00	270	67%	0.21	56	33.4%	0.16	0.76	0.21	56	33.4%	0.16	0.76
	0.25	90	28%	0.41	37	53.0%	0.05	0.13	0.41	37	53.0%	0.05	0.13
	0.50	15	6%	0.50	8	13.6%	0.00	0.00	0.50	8	13.6%	0.00	0.00
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Undefined	0.00	11,897	85%	0.16	1,871	56.4%	0.17	1.10	0.16	1,850	57.0%	0.16	1.01
	0.25	1,823	12%	0.45	816	25.8%	0.27	0.59	0.44	795	26.1%	0.20	0.46
	0.50	421	3%	0.79	333	13.5%	0.37	0.47	0.74	312	13.5%	0.19	0.26
	1.00	55	0%	1.47	81	4.3%	0.63	0.43	1.12	61	3.3%	0.06	0.06

Table 17-4: Distribution of Gold by AUZON - Iron Cap

AUZON	Uncapped Au Statistics Above Cutoff								Capped Au Statistics Above Cutoff				
	Au Cutoff (g/t)	Total Meters	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	17,564	44%	0.39	6,886	16.8%	0.42	1.07	0.38	6,676	17.4%	0.36	0.95
	0.25	9,846	34%	0.58	5,728	31.1%	0.48	0.82	0.56	5,513	32.5%	0.40	0.70
	0.50	3,891	16%	0.92	3,588	28.1%	0.61	0.67	0.88	3,346	28.5%	0.48	0.54
	1.00	1,018	6%	1.62	1,654	24.0%	0.85	0.52	1.50	1,444	21.6%	0.59	0.39
Lower Zone	0.00	9,188	22%	0.53	4,847	8.2%	0.49	0.93	0.51	4,725	8.4%	0.41	0.80
	0.25	7,128	43%	0.62	4,451	30.3%	0.52	0.83	0.61	4,330	31.0%	0.43	0.70
	0.50	3,149	25%	0.95	2,984	32.1%	0.64	0.67	0.91	2,863	32.9%	0.49	0.54
	1.00	876	10%	1.63	1,429	29.5%	0.88	0.54	1.49	1,308	27.7%	0.59	0.39
Middle Zone	0.00	1,799	83%	0.18	318	57.8%	0.16	0.92	0.17	301	61.2%	0.12	0.71
	0.25	311	13%	0.43	134	24.6%	0.25	0.58	0.37	117	27.6%	0.12	0.32
	0.50	73	3%	0.77	56	11.3%	0.32	0.42	0.58	34	11.2%	0.09	0.16
	1.00	15	1%	1.34	20	6.3%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Upper Zone	0.00	1,130	45%	0.30	337	28.9%	0.15	0.49	0.29	329	29.7%	0.13	0.44
	0.25	626	47%	0.38	240	51.4%	0.15	0.38	0.37	231	56.4%	0.12	0.33
	0.50	98	9%	0.68	66	19.7%	0.13	0.19	0.67	46	14.0%	0.07	0.10
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
PMON Au-Cu Zone	0.00	350	47%	0.30	105	28.2%	0.15	0.51	0.27	95	36.5%	0.13	0.47
	0.25	185	40%	0.41	75	46.2%	0.13	0.32	0.38	61	46.3%	0.10	0.27
	0.50	44	12%	0.62	27	25.6%	0.08	0.12	0.58	16	17.2%	0.00	0.01
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
FW Weak Min	0.00	3,217	62%	0.29	939	27.8%	0.35	1.19	0.28	890	29.3%	0.29	1.05
	0.25	1,218	24%	0.56	679	28.9%	0.45	0.80	0.52	630	31.6%	0.35	0.69
	0.50	449	10%	0.91	407	21.6%	0.58	0.64	0.82	349	24.0%	0.46	0.56
	1.00	128	4%	1.61	205	21.8%	0.68	0.43	1.55	135	15.2%	0.55	0.36
Mo-Zn Zone	0.00	1,135	74%	0.20	227	48.2%	0.14	0.70	0.20	223	49.1%	0.13	0.67
	0.25	297	21%	0.40	117	35.7%	0.12	0.30	0.38	114	38.8%	0.11	0.28
	0.50	60	5%	0.61	36	16.0%	0.04	0.07	0.60	27	12.1%	0.04	0.07
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Barren PMON	0.00	330	86%	0.15	50	72.1%	0.08	0.54	0.15	50	72.1%	0.08	0.54
	0.25	45	14%	0.31	14	27.9%	0.04	0.13	0.31	14	27.9%	0.04	0.13
	0.50	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Undefined	0.00	414	91%	0.15	62	71.5%	0.13	0.84	0.15	62	71.5%	0.13	0.84
	0.25	36	4%	0.50	18	10.8%	0.14	0.27	0.50	18	10.8%	0.14	0.27
	0.50	19	5%	0.59	11	17.7%	0.09	0.15	0.59	11	17.7%	0.09	0.15
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00

17.2 Copper Grade Distribution

The distribution of copper grades based on 15-meter-long drill hole composites is summarized at four different cutoff grades by the geologic constraints that were used to estimate block copper grades in Tables 17-5 through 17-8 for the Kerr, Sulphurets, Mitchell, and Iron Cap deposits, respectively.

As can be seen Tables 17-5 through 17-8 the average copper grade decreases in going from the Kerr deposit in the southern part of the district to the Sulphurets (middle portion of district) to the Mitchell deposit. This is an inverse relationship to that of gold. In the Kerr deposit about 45% of the copper assays are above a 0.25% copper cutoff. These distributions decrease dramatically going northward, with Sulphurets at 19% and Mitchell at only 11% of the composites above a 0.25% cutoff grade. About 25% of the Iron Cap composites are above a 0.25% cutoff. In general, the CV decreases in going from Kerr (1.02) to Iron Cap (0.61). The CV for Mitchell copper composites is 0.75.

Table 17-5: Distribution of Copper by Alteration - Kerr

Alteration Type	Uncapped Cu Statistics Above Cutoff								Capped Cu Statistics Above Cutoff				
	Cu Cutoff (%)	Total Meters	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	26,732	19%	0.31	8,280	1.6%	0.32	1.02	0.31	8,264	1.6%	0.32	1.02
	0.05	21,761	14%	0.37	8,151	3.3%	0.32	0.85	0.37	8,135	3.3%	0.32	0.85
	0.10	18,003	22%	0.44	7,881	12.2%	0.31	0.72	0.44	7,866	12.2%	0.31	0.72
	0.25	12,043	45%	0.57	6,869	83.0%	0.31	0.54	0.57	6,854	82.9%	0.30	0.53
Chlorite-Propylytic	0.00	10,309	6%	0.50	5,158	0.3%	0.36	0.72	0.50	5,156	0.3%	0.36	0.72
	0.05	9,710	4%	0.53	5,140	0.5%	0.35	0.66	0.53	5,138	0.5%	0.35	0.66
	0.10	9,322	15%	0.55	5,113	5.5%	0.34	0.63	0.55	5,111	5.5%	0.34	0.63
	0.25	7,741	75%	0.62	4,828	93.6%	0.33	0.53	0.62	4,826	93.6%	0.33	0.53
QSP	0.00	9,869	9%	0.25	2,443	1.3%	0.23	0.93	0.25	2,436	1.3%	0.22	0.91
	0.05	9,002	21%	0.27	2,412	6.2%	0.23	0.86	0.27	2,405	6.2%	0.22	0.84
	0.10	6,921	35%	0.33	2,260	23.5%	0.23	0.71	0.33	2,254	23.6%	0.23	0.69
	0.25	3,502	35%	0.48	1,686	69.0%	0.24	0.49	0.48	1,679	68.9%	0.23	0.47
Wk. CLQSP	0.00	1,928	22%	0.17	333	3.9%	0.19	1.11	0.17	333	3.9%	0.19	1.10
	0.05	1,511	28%	0.21	320	11.9%	0.20	0.94	0.21	320	11.9%	0.20	0.93
	0.10	969	28%	0.29	280	24.2%	0.21	0.73	0.29	280	24.2%	0.21	0.73
	0.25	428	22%	0.47	200	60.0%	0.21	0.44	0.47	199	60.0%	0.20	0.44
Premier Dyke	0.00	423	41%	0.15	65	5.8%	0.20	1.28	0.15	65	5.8%	0.20	1.28
	0.05	250	24%	0.24	61	10.4%	0.21	0.86	0.24	61	10.4%	0.21	0.86
	0.10	150	7%	0.36	54	6.1%	0.20	0.54	0.36	54	6.1%	0.20	0.54
	0.25	120	28%	0.42	50	77.7%	0.18	0.42	0.42	50	77.7%	0.18	0.42
Hornblende Dyke	0.00	359	71%	0.06	21	34.5%	0.06	1.09	0.06	21	34.5%	0.06	1.09
	0.05	105	13%	0.13	13	13.5%	0.08	0.60	0.13	13	13.5%	0.08	0.60
	0.10	60	13%	0.18	11	33.9%	0.07	0.38	0.18	11	33.9%	0.07	0.38
	0.25	15	4%	0.25	4	18.2%	0.00	0.00	0.25	4	18.2%	0.00	0.00
Undefined	0.00	3,844	69%	0.07	261	21.5%	0.11	1.67	0.07	254	22.1%	0.11	1.62
	0.05	1,183	16%	0.17	205	15.8%	0.16	0.92	0.17	198	16.2%	0.15	0.89
	0.10	580	9%	0.28	164	23.7%	0.17	0.60	0.27	157	24.3%	0.16	0.57
	0.25	237	6%	0.43	102	39.0%	0.17	0.40	0.40	95	37.4%	0.17	0.42

Table 17-6: Distribution of Copper by CUZON - Sulphurets

CUZON	Uncapped Cu Statistics Above Cutoff								Capped Cu Statistics Above Cutoff				
	Cu Cutoff (%)	Total Meters	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	22,917	30%	0.15	3,425	4.9%	0.17	1.13	0.15	3,391	4.9%	0.17	1.13
	0.05	16,037	24%	0.20	3,259	11.7%	0.18	0.87	0.20	3,224	11.8%	0.17	0.86
	0.10	10,512	27%	0.27	2,860	28.2%	0.18	0.68	0.27	2,823	29.1%	0.18	0.67
	0.25	4,252	19%	0.45	1,894	55.3%	0.18	0.40	0.45	1,835	54.1%	0.17	0.38
Au Leach Zone	0.00	765	48%	0.07	50	25.4%	0.05	0.72	0.06	50	25.7%	0.05	0.72
	0.05	401	39%	0.09	37	41.8%	0.05	0.54	0.09	37	41.5%	0.05	0.55
	0.10	105	12%	0.16	16	24.1%	0.06	0.38	0.15	16	24.0%	0.06	0.40
	0.25	15	2%	0.29	4	8.6%	0.00	0.00	0.29	4	8.7%	0.00	0.00
Au Breccia Zone	0.00	907	50%	0.08	71	14.7%	0.10	1.29	0.08	71	14.8%	0.10	1.29
	0.05	457	29%	0.13	61	27.3%	0.12	0.90	0.13	61	27.4%	0.12	0.90
	0.10	190	16%	0.22	41	31.3%	0.15	0.68	0.22	41	36.5%	0.15	0.68
	0.25	45	5%	0.42	19	26.6%	0.18	0.43	0.51	15	21.3%	0.17	0.34
Canyon Zone	0.00	412	0%	0.17	69	0.0%	0.11	0.63	0.17	68	0.0%	0.10	0.62
	0.05	412	25%	0.17	69	10.0%	0.11	0.63	0.17	68	10.0%	0.10	0.62
	0.10	307	56%	0.20	62	50.9%	0.10	0.50	0.20	61	51.4%	0.10	0.48
	0.25	75	18%	0.36	27	39.1%	0.07	0.20	0.35	26	38.6%	0.07	0.19
Raewyn Copper	0.00	5,183	3%	0.35	1,823	0.2%	0.22	0.62	0.35	1,817	0.2%	0.21	0.61
	0.05	5,042	5%	0.36	1,820	1.1%	0.21	0.59	0.36	1,814	1.1%	0.21	0.58
	0.10	4,787	30%	0.38	1,800	14.4%	0.21	0.55	0.37	1,795	14.4%	0.20	0.54
	0.25	3,246	63%	0.47	1,538	84.4%	0.18	0.39	0.47	1,532	84.3%	0.18	0.37
Main Copper	0.00	1,305	46%	0.07	92	16.7%	0.06	0.85	0.07	90	17.0%	0.06	0.82
	0.05	709	33%	0.11	76	31.9%	0.06	0.54	0.11	75	32.5%	0.05	0.52
	0.10	281	19%	0.17	47	43.9%	0.05	0.29	0.16	45	50.5%	0.04	0.27
	0.25	27	2%	0.26	7	7.5%	0.01	0.03	0.00	0	0.0%	0.00	0.00
Undefined	0.00	14,346	37%	0.09	1,320	9.5%	0.09	0.98	0.09	1,295	9.7%	0.08	0.93
	0.05	9,017	29%	0.13	1,195	23.0%	0.09	0.69	0.13	1,169	23.5%	0.08	0.64
	0.10	4,843	28%	0.18	892	44.9%	0.10	0.53	0.18	865	46.9%	0.09	0.48
	0.25	844	6%	0.35	299	22.6%	0.11	0.32	0.34	257	19.9%	0.09	0.27

Table 17-7: Distribution of Copper by CUZON - Mitchell

CUZON	Uncapped Cu Statistics Above Cutoff								Capped Cu Statistics Above Cutoff				
	Cu Cutoff (%)	Total Meters	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	49,732	15%	0.14	7,034	2.5%	0.11	0.79	0.14	6,990	2.5%	0.10	0.75
	0.05	42,360	23%	0.16	6,860	12.4%	0.11	0.67	0.16	6,816	12.5%	0.10	0.62
	0.10	30,830	51%	0.19	5,988	57.8%	0.11	0.57	0.19	5,944	58.2%	0.10	0.52
	0.25	5,714	11%	0.34	1,925	27.4%	0.18	0.54	0.33	1,879	26.9%	0.15	0.47
Upper Plate	0.00	1,125	7%	0.14	161	1.6%	0.09	0.64	0.14	157	1.6%	0.08	0.61
	0.05	1,050	30%	0.15	159	14.9%	0.09	0.59	0.15	155	15.2%	0.08	0.56
	0.10	713	51%	0.19	135	54.8%	0.09	0.45	0.18	131	58.1%	0.08	0.42
	0.25	135	12%	0.34	46	28.8%	0.05	0.13	0.33	39	25.0%	0.04	0.11
Lower Plate	0.00	30,137	1%	0.18	5,392	0.1%	0.08	0.44	0.18	5,384	0.1%	0.08	0.44
	0.05	29,898	12%	0.18	5,385	5.5%	0.08	0.44	0.18	5,377	5.5%	0.08	0.43
	0.10	26,331	71%	0.19	5,088	65.2%	0.07	0.39	0.19	5,080	65.3%	0.07	0.38
	0.25	5,045	17%	0.31	1,571	29.1%	0.07	0.22	0.31	1,563	29.0%	0.06	0.19
Bornite Breccia	0.00	150	0%	1.03	154	0.0%	0.52	0.51	0.83	125	0.0%	0.42	0.50
	0.05	150	0%	1.03	154	0.0%	0.52	0.51	0.83	125	0.0%	0.42	0.50
	0.10	150	10%	1.03	154	2.4%	0.52	0.51	0.83	125	2.9%	0.42	0.50
	0.25	135	90%	1.11	150	97.6%	0.47	0.42	0.90	121	97.1%	0.39	0.44
Bornite Leach Halo	0.00	270	22%	0.16	42	5.0%	0.12	0.74	0.15	39	5.4%	0.09	0.63
	0.05	210	17%	0.19	40	8.5%	0.11	0.57	0.18	37	9.1%	0.08	0.45
	0.10	165	39%	0.22	37	39.3%	0.10	0.47	0.20	34	42.3%	0.07	0.34
	0.25	60	22%	0.33	20	47.2%	0.08	0.25	0.28	17	43.1%	0.01	0.03
Undefined	0.00	18,050	39%	0.07	1,285	12.6%	0.08	1.17	0.07	1,285	12.6%	0.08	1.17
	0.05	11,052	42%	0.10	1,123	42.6%	0.09	0.92	0.10	1,123	42.6%	0.09	0.92
	0.10	3,471	17%	0.17	575	34.0%	0.15	0.88	0.17	575	34.0%	0.15	0.88
	0.25	339	2%	0.41	138	10.7%	0.38	0.93	0.41	138	10.7%	0.38	0.93

Table 17-8: Distribution of Copper by AUZON - Iron Cap

AUZON	Uncapped Cu Statistics Above Cutoff								Capped Cu Statistics Above Cutoff				
	Cu Cutoff (%)	Total Meters	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. Of Variation
All Data	0.00	17,138	8%	0.19	3,239	1.3%	0.12	0.63	0.19	3,214	1.3%	0.11	0.61
	0.05	15,713	15%	0.20	3,197	5.9%	0.11	0.55	0.20	3,172	6.0%	0.11	0.54
	0.10	13,214	52%	0.23	3,006	47.4%	0.11	0.47	0.23	2,980	48.1%	0.10	0.45
	0.25	4,220	25%	0.35	1,469	45.4%	0.11	0.30	0.34	1,435	44.6%	0.10	0.28
Lower Zone	0.00	9,164	0%	0.22	2,044	0.0%	0.12	0.55	0.22	2,027	0.0%	0.12	0.53
	0.05	9,164	11%	0.22	2,044	3.9%	0.12	0.55	0.22	2,027	4.0%	0.12	0.53
	0.10	8,176	57%	0.24	1,964	43.8%	0.12	0.49	0.24	1,946	44.5%	0.11	0.47
	0.25	2,964	32%	0.36	1,069	52.3%	0.11	0.32	0.36	1,045	51.6%	0.10	0.29
Middle Zone	0.00	1,799	6%	0.18	316	1.1%	0.09	0.50	0.17	314	1.1%	0.09	0.50
	0.05	1,700	18%	0.18	312	7.8%	0.08	0.46	0.18	311	7.8%	0.08	0.45
	0.10	1,376	56%	0.21	288	56.2%	0.07	0.34	0.21	286	56.4%	0.07	0.34
	0.25	364	20%	0.30	110	34.9%	0.05	0.17	0.30	109	34.7%	0.05	0.17
Upper Zone	0.00	1,130	1%	0.23	265	0.3%	0.11	0.45	0.23	262	0.3%	0.10	0.42
	0.05	1,115	6%	0.24	265	2.1%	0.10	0.44	0.23	261	2.2%	0.09	0.40
	0.10	1,046	52%	0.25	259	40.4%	0.10	0.40	0.24	255	40.9%	0.09	0.36
	0.25	458	41%	0.33	152	57.3%	0.09	0.28	0.32	148	56.7%	0.07	0.22
PMON Au-Cu Zone	0.00	350	8%	0.20	71	1.7%	0.11	0.53	0.20	71	1.7%	0.11	0.53
	0.05	323	13%	0.22	69	3.9%	0.10	0.47	0.21	69	3.9%	0.10	0.47
	0.10	278	43%	0.24	67	36.7%	0.09	0.36	0.24	67	42.0%	0.09	0.36
	0.25	128	36%	0.32	41	57.7%	0.05	0.15	0.33	37	52.3%	0.04	0.13
FW Weak Min	0.00	2,840	36%	0.09	252	11.5%	0.07	0.79	0.09	250	11.6%	0.07	0.78
	0.05	1,811	29%	0.12	223	23.1%	0.07	0.54	0.12	221	23.2%	0.06	0.53
	0.10	1,001	31%	0.16	164	51.8%	0.06	0.38	0.16	163	51.8%	0.06	0.37
	0.25	108	4%	0.32	34	13.6%	0.04	0.12	0.31	34	13.4%	0.03	0.11
Mo-Zn Zone	0.00	1,135	0%	0.20	223	0.0%	0.07	0.36	0.20	221	0.0%	0.06	0.32
	0.05	1,135	3%	0.20	223	1.1%	0.07	0.36	0.20	221	1.1%	0.06	0.32
	0.10	1,105	84%	0.20	221	76.2%	0.07	0.35	0.20	219	76.8%	0.06	0.30
	0.25	157	14%	0.32	51	22.7%	0.10	0.32	0.31	49	22.1%	0.07	0.23
Barren PMON	0.00	330	50%	0.06	21	31.5%	0.03	0.54	0.06	21	31.5%	0.03	0.54
	0.05	165	41%	0.09	14	47.1%	0.03	0.40	0.09	14	47.1%	0.03	0.40
	0.10	30	9%	0.15	4	21.4%	0.03	0.20	0.15	4	21.4%	0.03	0.20
	0.25	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Undefined	0.00	390	23%	0.12	48	1.0%	0.09	0.77	0.12	48	1.0%	0.09	0.77
	0.05	300	25%	0.16	47	17.0%	0.08	0.50	0.16	47	17.0%	0.08	0.50
	0.10	201	41%	0.19	39	54.9%	0.07	0.36	0.19	39	54.9%	0.07	0.36
	0.25	41	11%	0.31	13	27.2%	0.05	0.15	0.31	13	27.2%	0.05	0.15

Like gold, copper is seen to be distributed in a number of logged lithologic and alteration types in the four mineralized zones. In general, it has not been possible to identify any particular lithologic unit or alteration type that adequately defines a mineralized copper population for any of the KSM deposits except for Kerr where alteration was used to constrain the estimate of block grades. Copper grades tend to be somewhat lower in chlorite-propylitic alteration than quartz-sericite-pyrite alteration, but this relationship is not well developed. Given these observations, RMI elected to use grade envelopes for Sulphurets, Mitchell, and Iron Cap to constrain the estimate of block copper grades (see Section 17.5).

17.3 Assay Grade Capping

The author used cumulative probability plots to identify high-grade outliers for both gold and copper assays. Figures 17-1 through 17-8 show cumulative probability plots using the cumulative normal distribution function for gold and copper by mineral zone.

Figure 17-1: Kerr Au Assay Cumulative Probability Plot

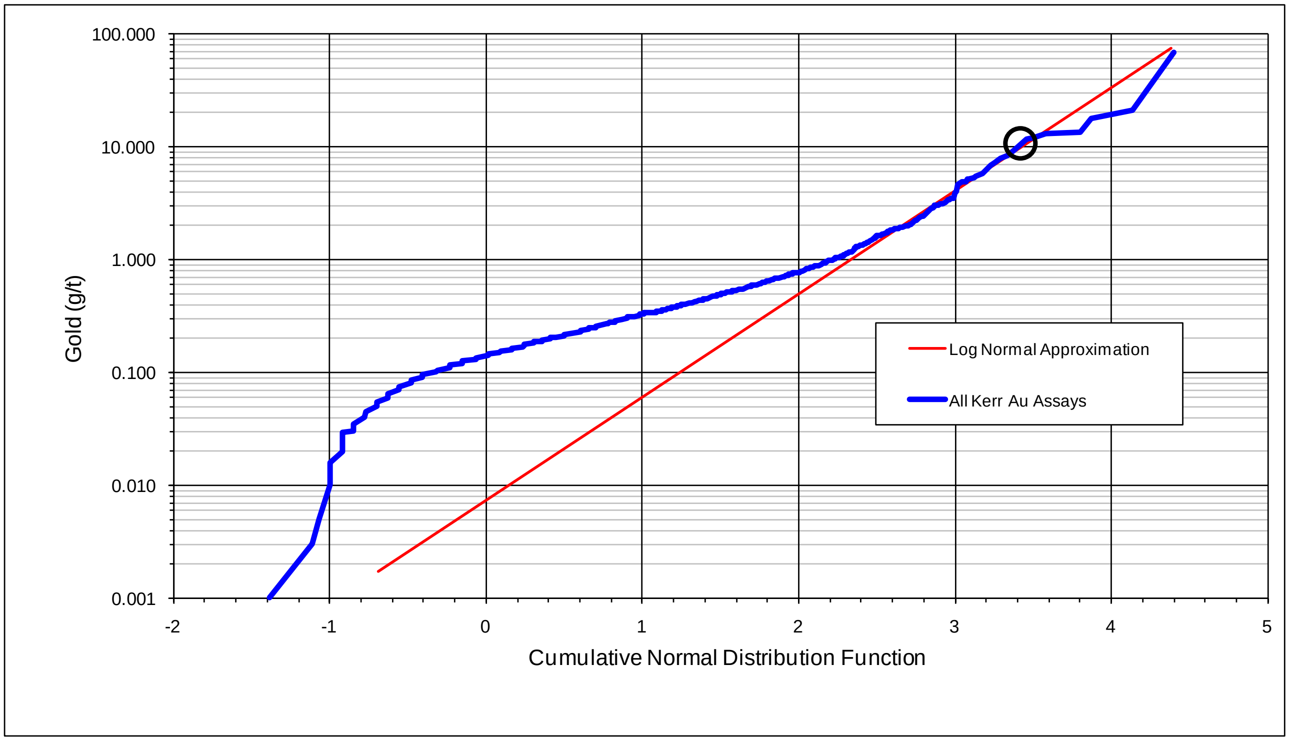


Figure 17-2: Sulphurets Au Assay Cumulative Probability Plot

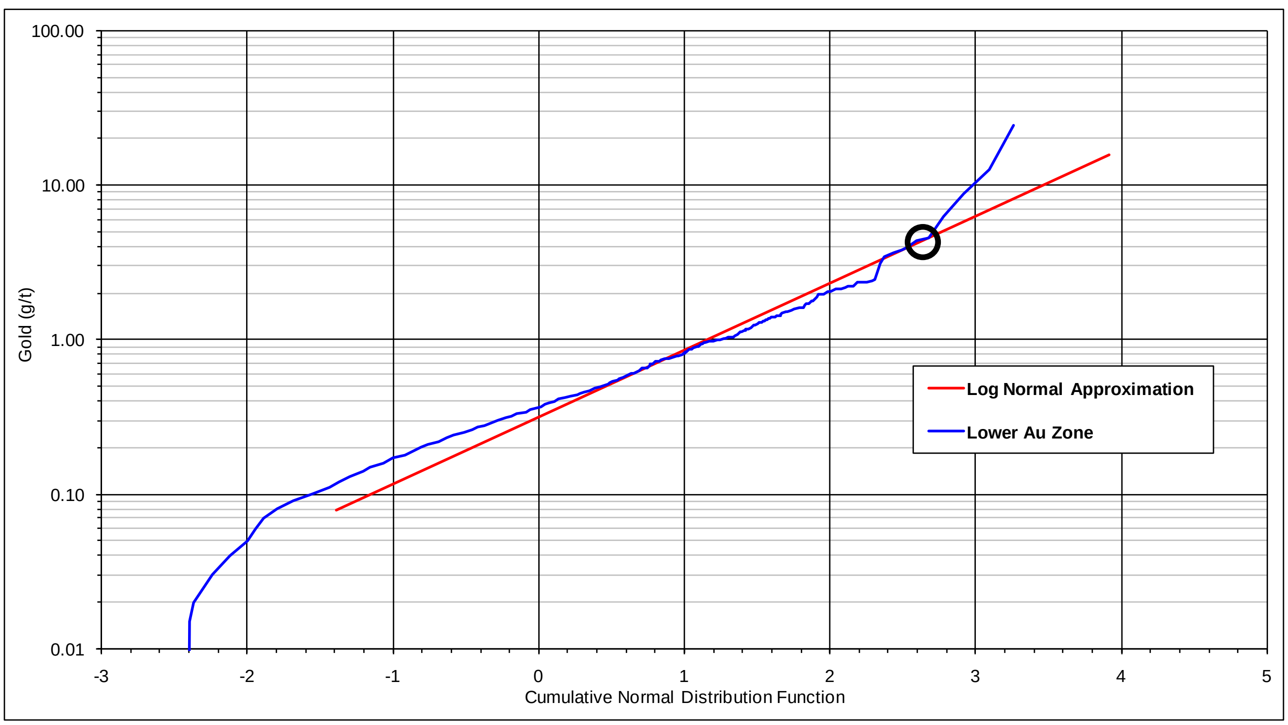


Figure 17-3: Mitchell Au Assay Cumulative Probability Plot

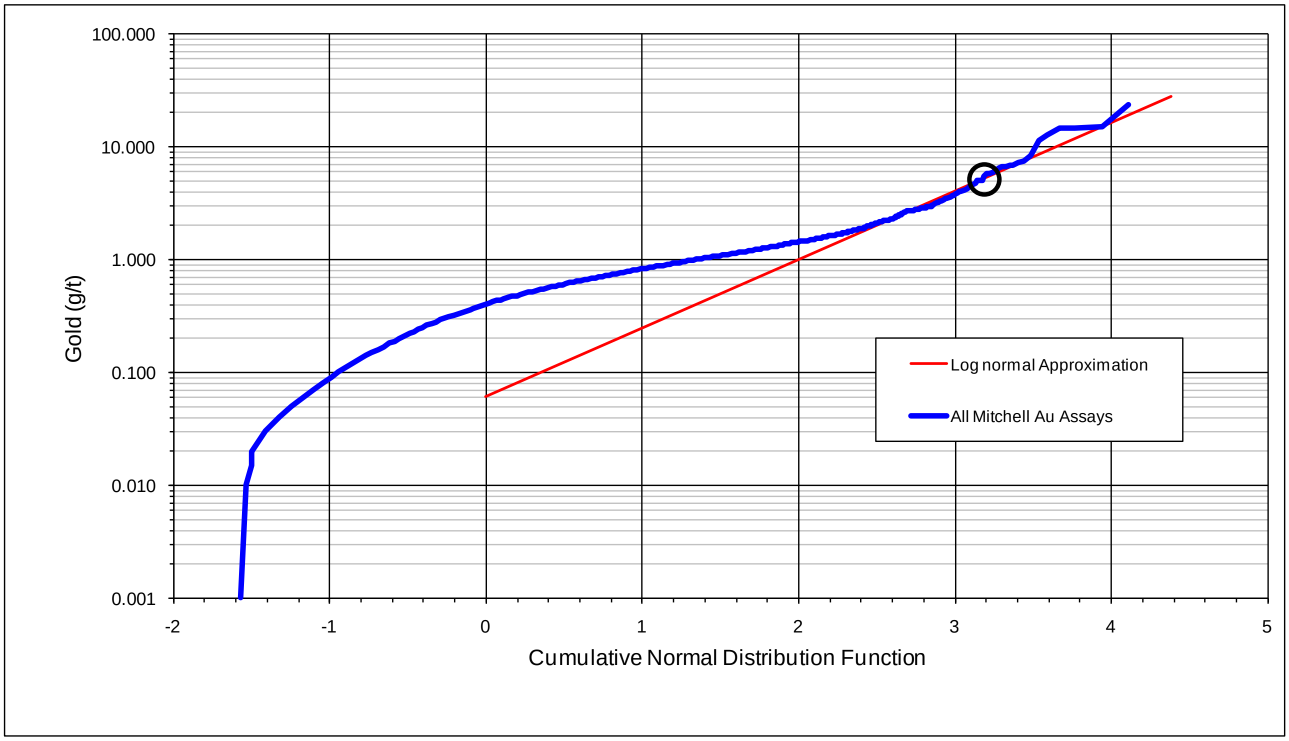


Figure 17-4: Iron Cap Au Assay Cumulative Probability Plot

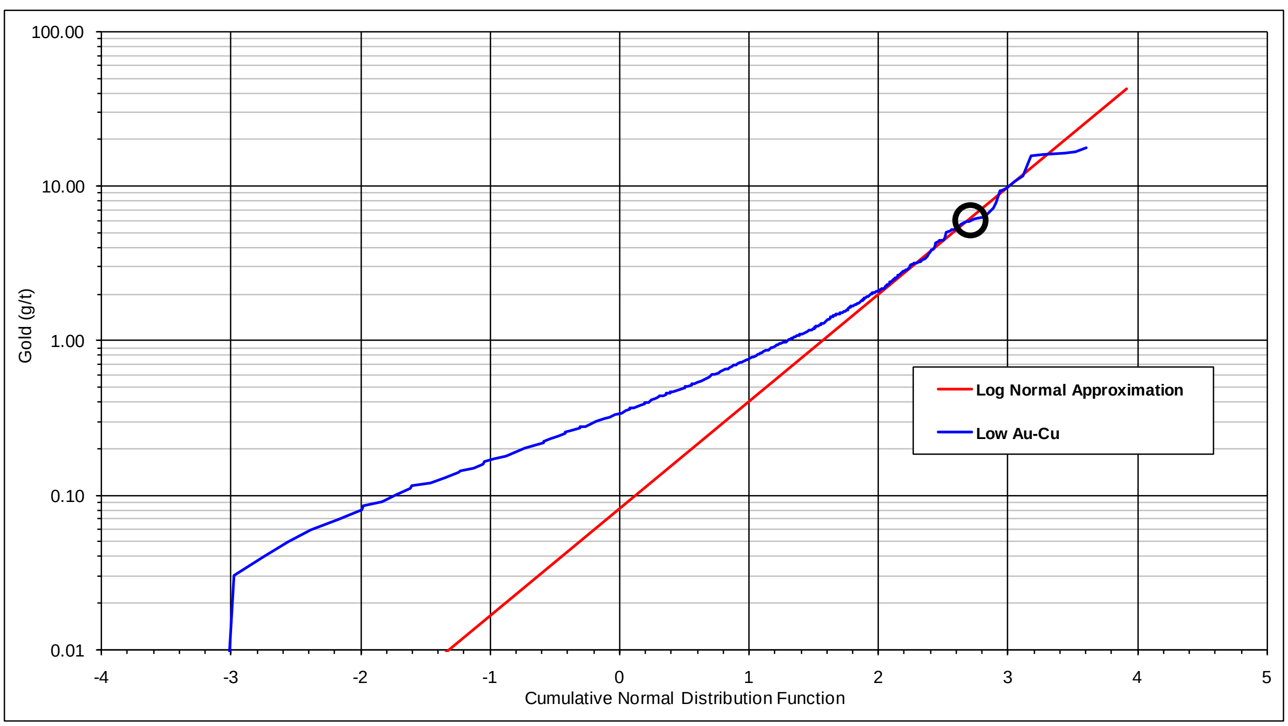


Figure 17-5: Kerr Cu Assay Cumulative Probability Plot

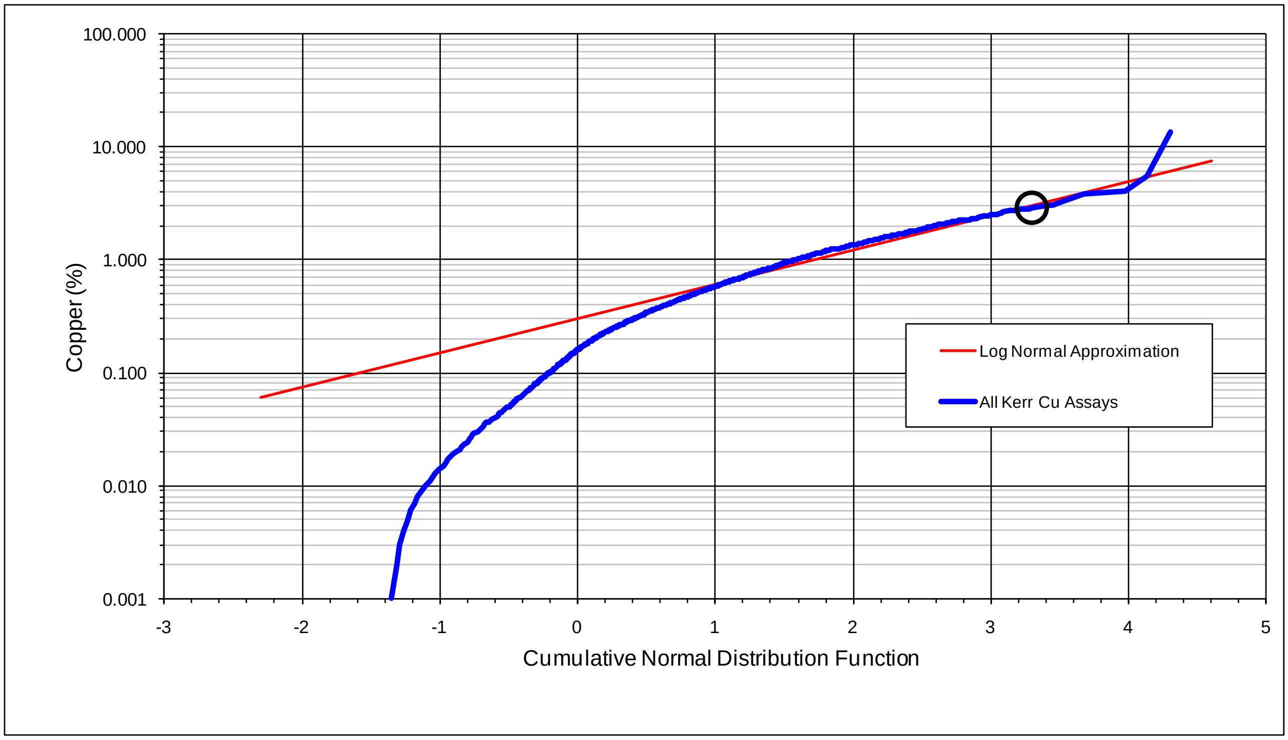


Figure 17-6: Sulphurets Cu Assay Cumulative Probability Plot

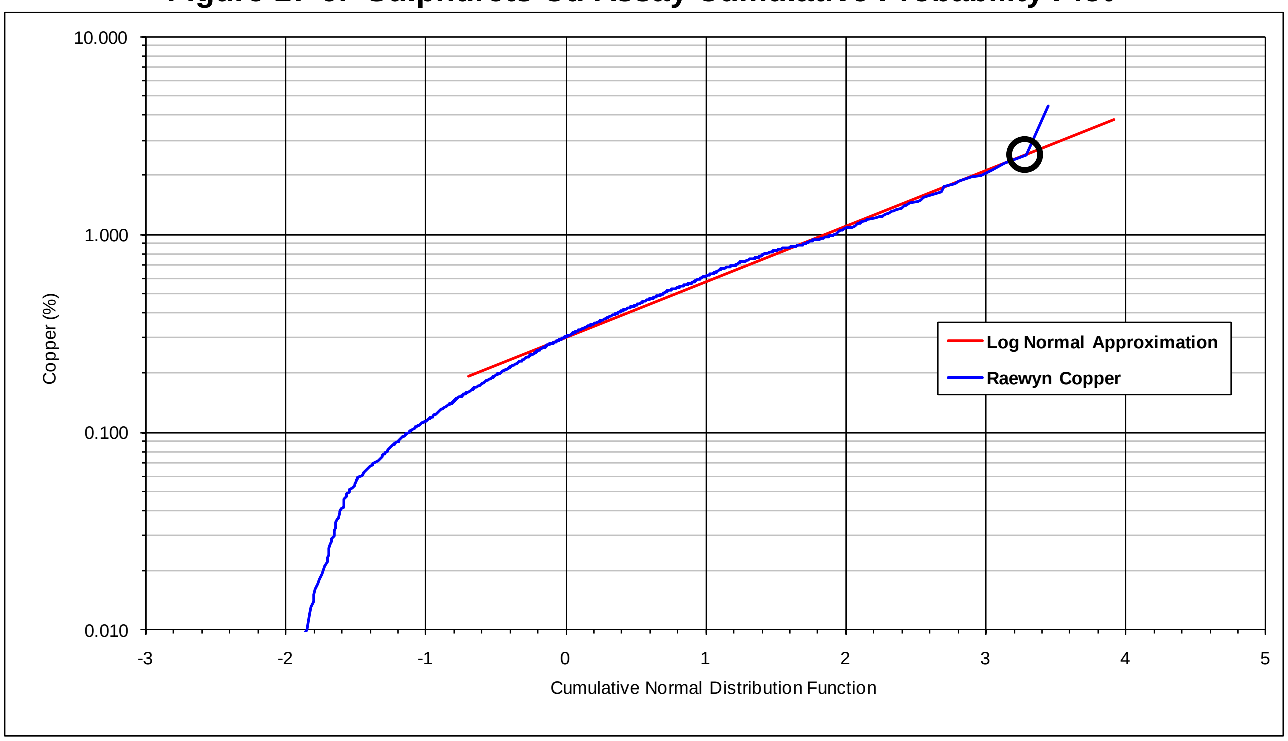


Figure 17-7: Mitchell Cu Assay Cumulative Probability Plot

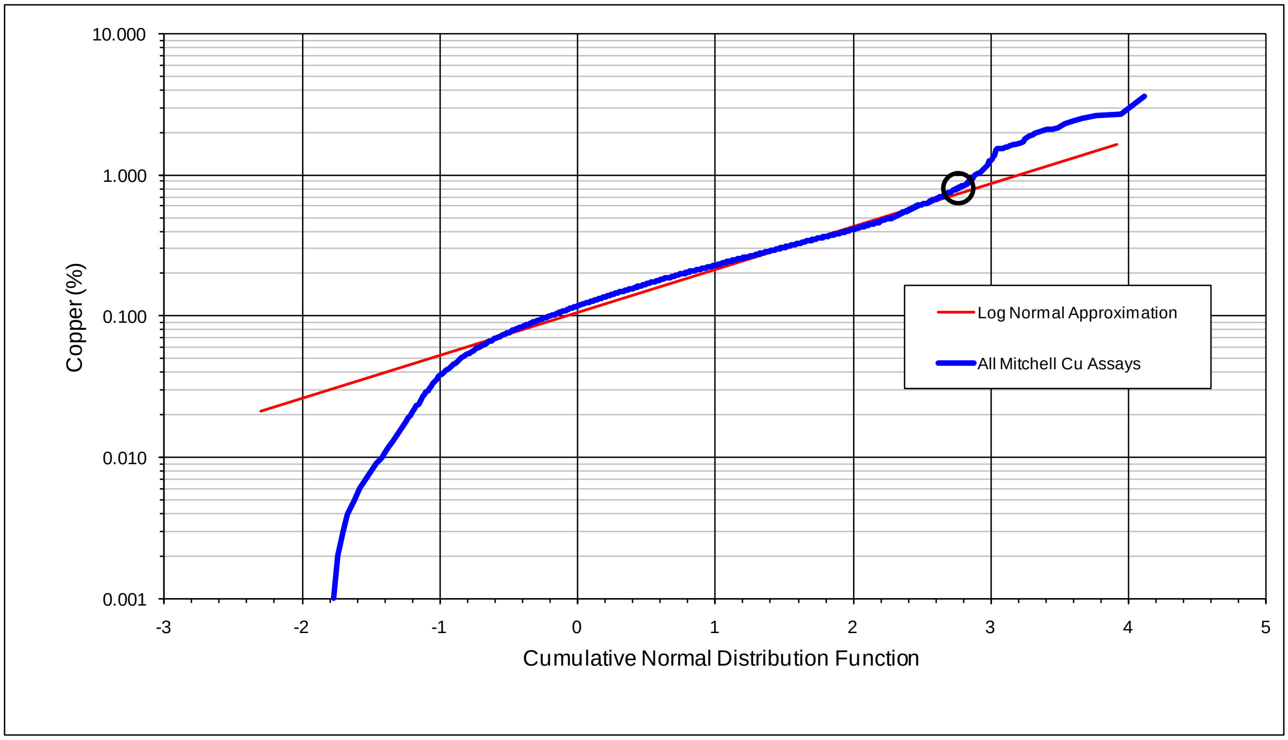
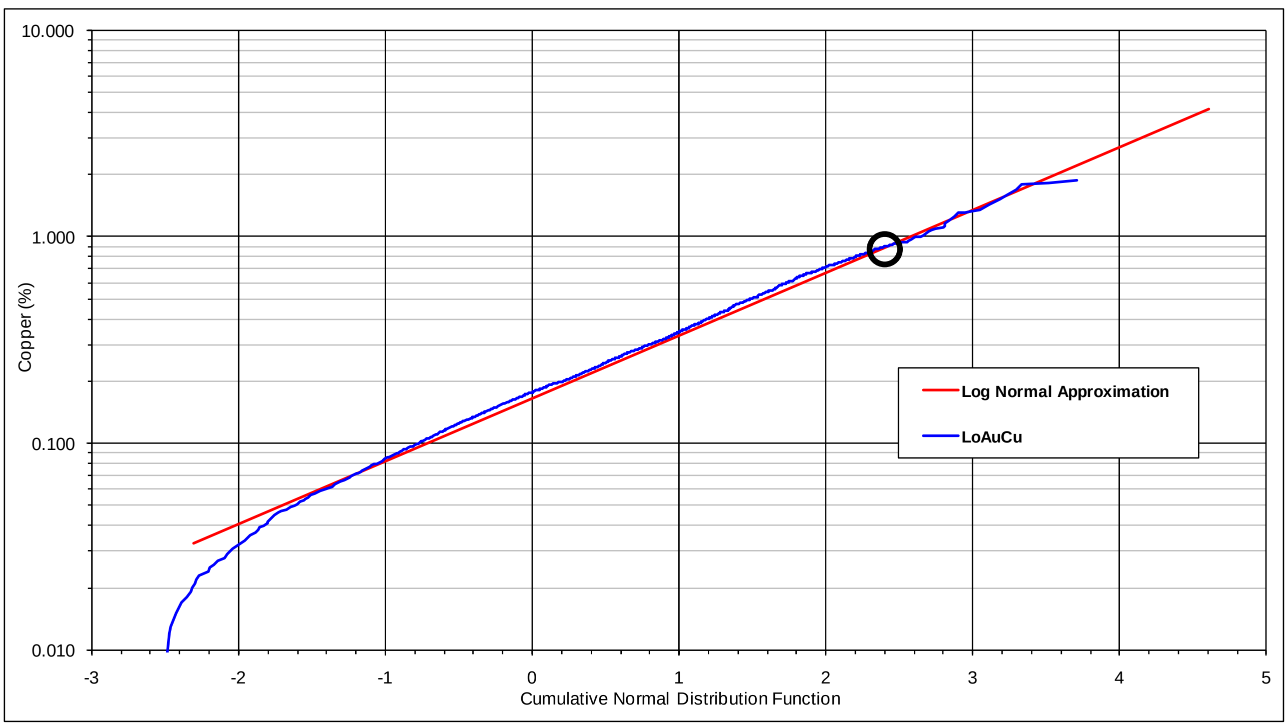


Figure 17-8: Iron Cap Cu Assay Cumulative Probability Plot



Based on the information shown in Figures 17-1 through 17-8 and other cumulative probability plots not shown, the author capped raw gold and copper assays at the area highlighted by the black circle where the distribution of grades becomes erratic.

Tables 17-9 through 17-11 summarize the capping limits that were established for gold, copper, and silver/molybdenum by mineral zone.

Table 17-9: Gold Grade Capping Limits

Zone	Attribute	Cap Grade (g/t)
Kerr	All	10.0
Sulphurets	Lower Plate Au Zone	5.0
	Au Breccia Zone	10.0
	Au Leach Zone	1.0
	Canyon Zone	6.0
	Raewyn Copper	5.0
	Undefined	5.0
Mitchell	All	5.0
Iron Cap	Lower Au Zone	6.5
	Middle Au Zone	1.5
	Upper Au Zone	1.5
	PMON Au-Cu Zone	0.7
	FW Weak Zone	3.0
	Mo-Zn Zone	1.5
	Undefined	1.0

Table 17-10: Copper Grade Capping Limits

Zone	Attribute	Cap Grade (%)
Kerr	All	2.75
Sulphurets	Au Breccia Zone	0.30
	Au Leach Zone	0.40
	Canyon Zone	0.70
	Raewyn Copper	2.50
	Main Copper	0.30
	Undefined	0.60
Mitchell	Upper Plate	0.90
	Lower Plate	0.90
	Bornite Breccia	1.50
	Bornite Leach Breccia	0.35
Iron Cap	Lower Au Zone	0.90
	Middle Au Zone	0.70
	Upper Au Zone	0.60
	PMON Au-Cu Zone	0.70
	FW Weak Zone	0.60
	Mo-Zn Zone	0.70
	Undefined	0.60

Table 17-11: Silver and Molybdenum Grade Capping Limits

Zone	Ag (g/t)	Mo (ppm)
Kerr	500	n/a
Sulphurets	100	1000
Mitchell	180	1200
Iron Cap	n/a	n/a

17.4 Drill Hole Composites

The raw drill hole data were composited into 15-meter-long composites starting from the drill hole collar. Most of the original assay data were in the range of 1.5 to 3.0 meters long, with the majority being 2-meters-long. Based on the scale of the deposit, 15-meter-long composites were deemed to be an appropriate length for estimating Mineral Resources.

The assays were composited using MineSight® software. Various geologic data were assigned to the 15-meter-long composites using the majority rule method.

17.5 Geologic Constraints

Various lithologic, alteration, structural domains, and metal grade envelopes were constructed for each of the deposits by RMI and Seabridge personnel. Most of these three-dimensional wireframes were initially interpreted onto cross sections which were then reconciled in bench plan prior to building the final wireframe.

As previously mentioned, gold and copper grades within the deposits are not necessarily confined to distinct geologic units (e.g. lithology, alteration, etc.). For this reason alteration zones were used for Kerr while hybrid gold and copper envelopes were used to constrain the estimate of block grades for Sulphurets, Mitchell, and Iron Cap. Constraints used to estimate gold, silver, copper, and molybdenum are summarized in Table 17-12 for each deposit.

Table 17-12: Constraints Used to Estimate Block Grades

Mineral Zone	Gold	Silver	Copper	Molybdenum
Kerr	Alteration	Alteration	Alteration	n/a
Sulphurets	AUZON	AUZON	CUZON	CUZON
Mitchell	AUZON	AUZON	CUZON	CUZON
Iron Cap	AUZON	AUZON	AUZON	AUZON

Descriptions for alteration types used to constrain the estimate of Kerr gold, silver, and copper grades are summarized in Table 17-3.

Table 17-13: Alteration Code Definitions

Code	Description
1	Chlorite-propylytic
2	Quartz-sericite-pyrite (QSP)
3	Mitchell IARG
4	Kerr weak CLQSP
5	Kerr Premier-style dike
6	Kerr hornblende dike
7	Iron Cap CL-SIL
8	Iron Cap FW SIH
9	Iron Cap KP
10	Iron Cap KP-PMON
11	Iron Cap SIH
12	Iron Cap SIL

The AUZON and CUZON wireframes for the Sulphurets and Mitchell zones are a combination of lithology/alteration and grade. In the case of the Mitchell zone, the AUZON and CUZON's were more heavily weighted towards grade. A 0.25 g/t gold cutoff and 0.10% copper cutoff were used to design the AUZON and CUZON wireframes for the Mitchell zone. In the Sulphurets zone, the Sulphurets Thrust Fault (STF) was used to define upper and lower plates. In the Mitchell zone, the Mitchell Thrust Fault (MTF) was used to define upper and lower plates. The AUZON codes used to constrain the estimate of gold, copper, silver, and molybdenum grades for the Iron Cap zone are a combination of lithology and degree of mineralization. Tables 17-14 and 17-15 summarize definitions for AUZON and CUZON, respectively.

Table 17-14: AUZON Code Definitions

AUZON	Description
1	Mitchell and Sulphurets Upper Plate
2	Mitchell and Sulphurets Lower Plate
3	Mitchell Bornite Breccia and Sulphurets Au Leach Breccia
4	Mitchell Bornite Leach Halo and Sulphurets Au Leach Zone
5	Sulphurets Canyon Zone
6	Iron Cap Lower Au Zone
7	Iron Cap Middle Au Zone
8	Iron Cap Upper Au Zone
9	Iron Cap PMON Au-Cu Zone
10	Iron Cap Footwall Weak Mineralized Zone
11	Iron Cap Molybdenum-Zinc Zone
12	Iron Cap Barren PMON
13	Sulphurets Raewyn Copper Zone
14	Main Copper Monzonite
29	Default Code

Table 17-15: CUZON Code Definitions

CUZON	Description
1	Mitchell and Sulphurets Upper Plate
2	Mitchell and Sulphurets Lower Plate
3	Mitchell Bornite Breccia and Sulphurets Au Leach Breccia
4	Mitchell Bornite Leach Halo and Sulphurets Au Leach Zone
5	Sulphurets Canyon Zone
6	Iron Cap Lower Au Zone
7	Iron Cap Middle Au Zone
8	Iron Cap Upper Au Zone
9	Iron Cap PMON Au-Cu Zone
10	Iron Cap Footwall Weak Mineralized Zone
11	Iron Cap Molybdenum-Zinc Zone
12	Iron Cap Barren PMON
13	Sulphurets Raewyn Copper Zone
14	Main Copper Monzonite
29	Default Code

17.6 **Variography**

The author generated a number of gold and copper correlograms and variograms using both drill hole assays and 15-meter-long drill hole composites.

Figures 17-9 through 17-12 show gold grade correlograms for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. Figures 17-13 through 17-16 show copper grade correlograms for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. Figures 17-17 through 17-20 show 0.5 g/t gold equivalent (AUEQV) correlograms for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively.

Figure 17-9: Kerr Au Grade Correlogram

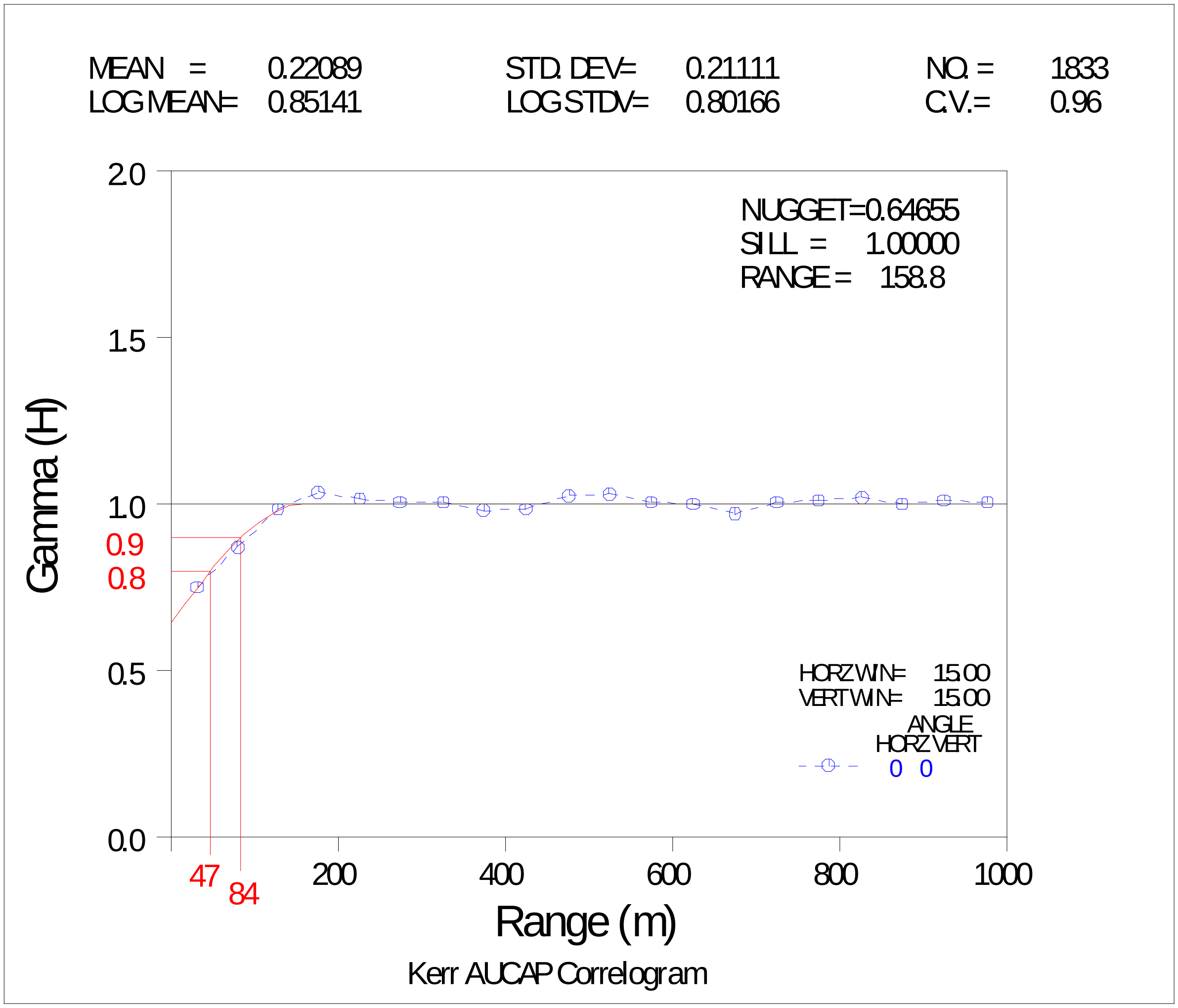


Figure 17-10: Sulphurets Au Grade Correlogram

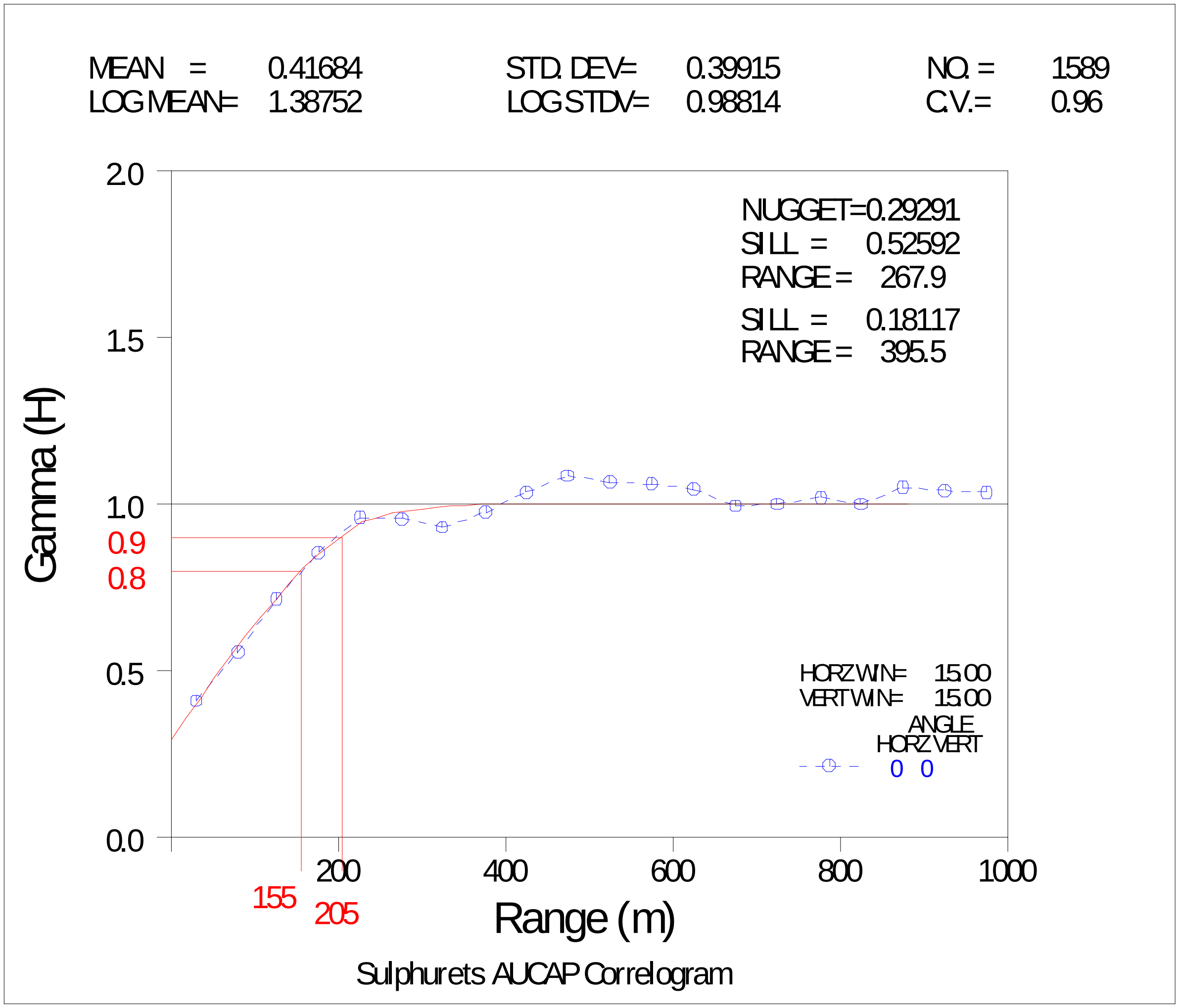


Figure 17-11: Mitchell Au Grade Correlogram

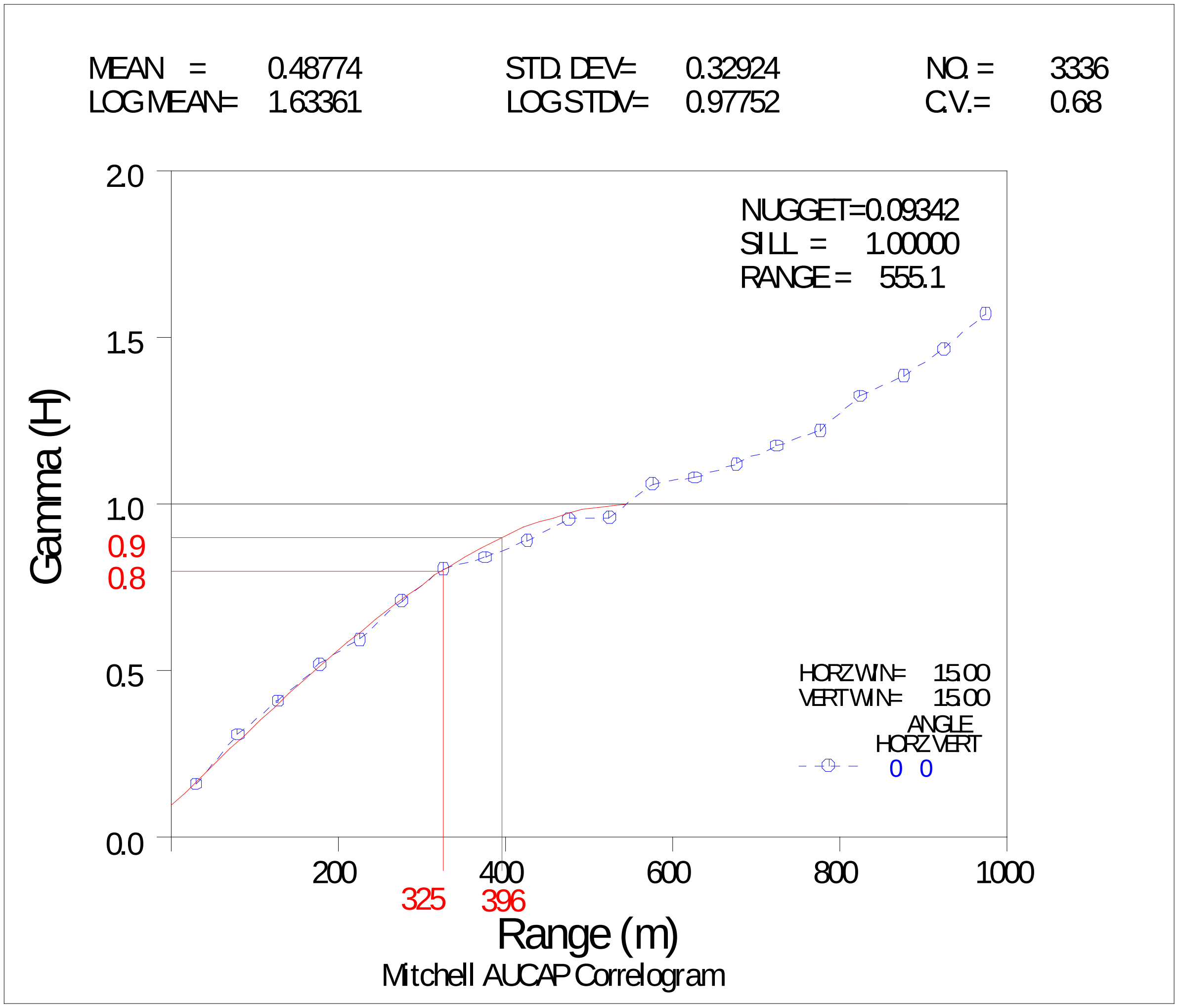


Figure 17-12: Iron Cap Au Grade Correlogram

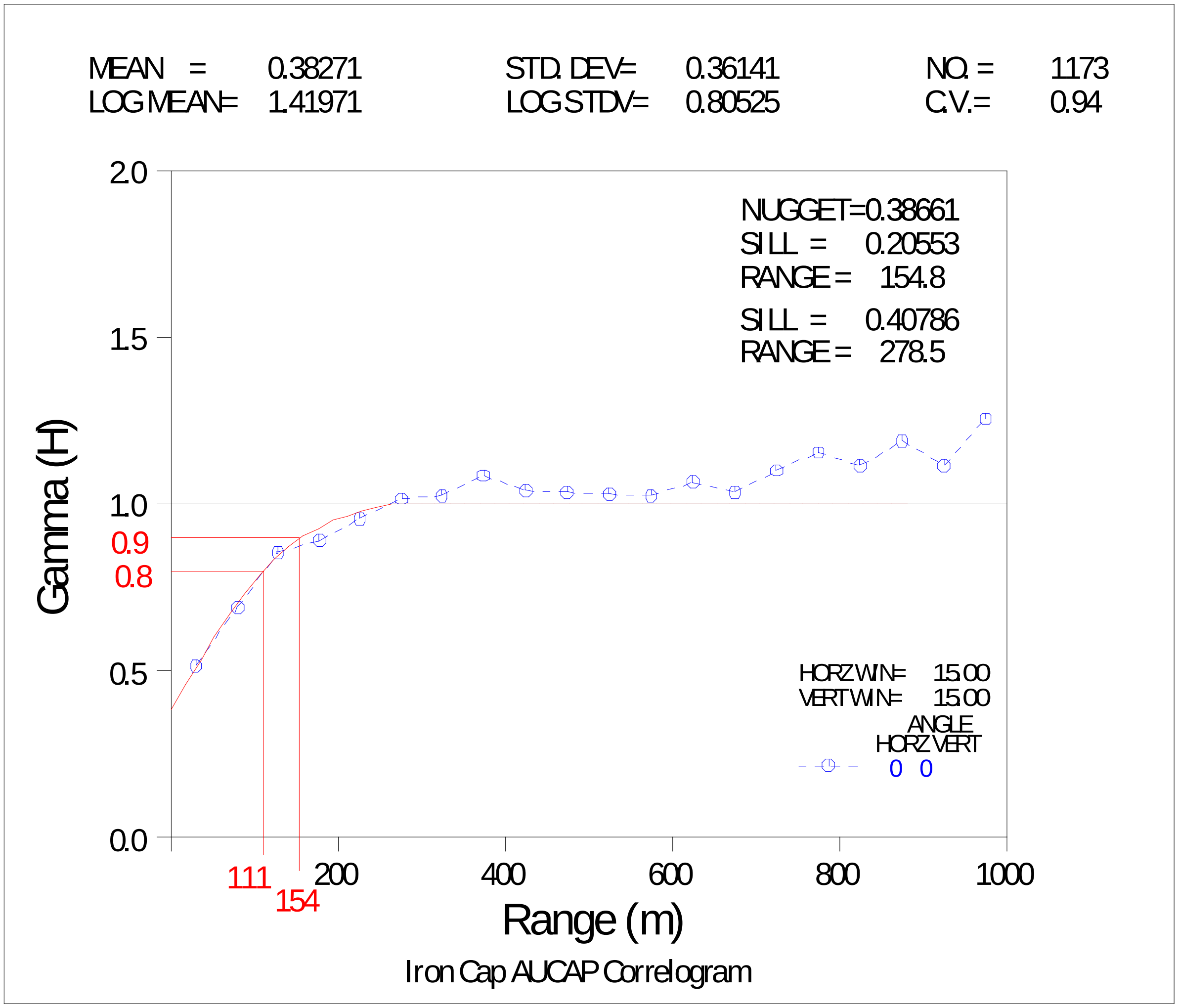


Figure 17-13: Kerr Cu Grade Correlogram

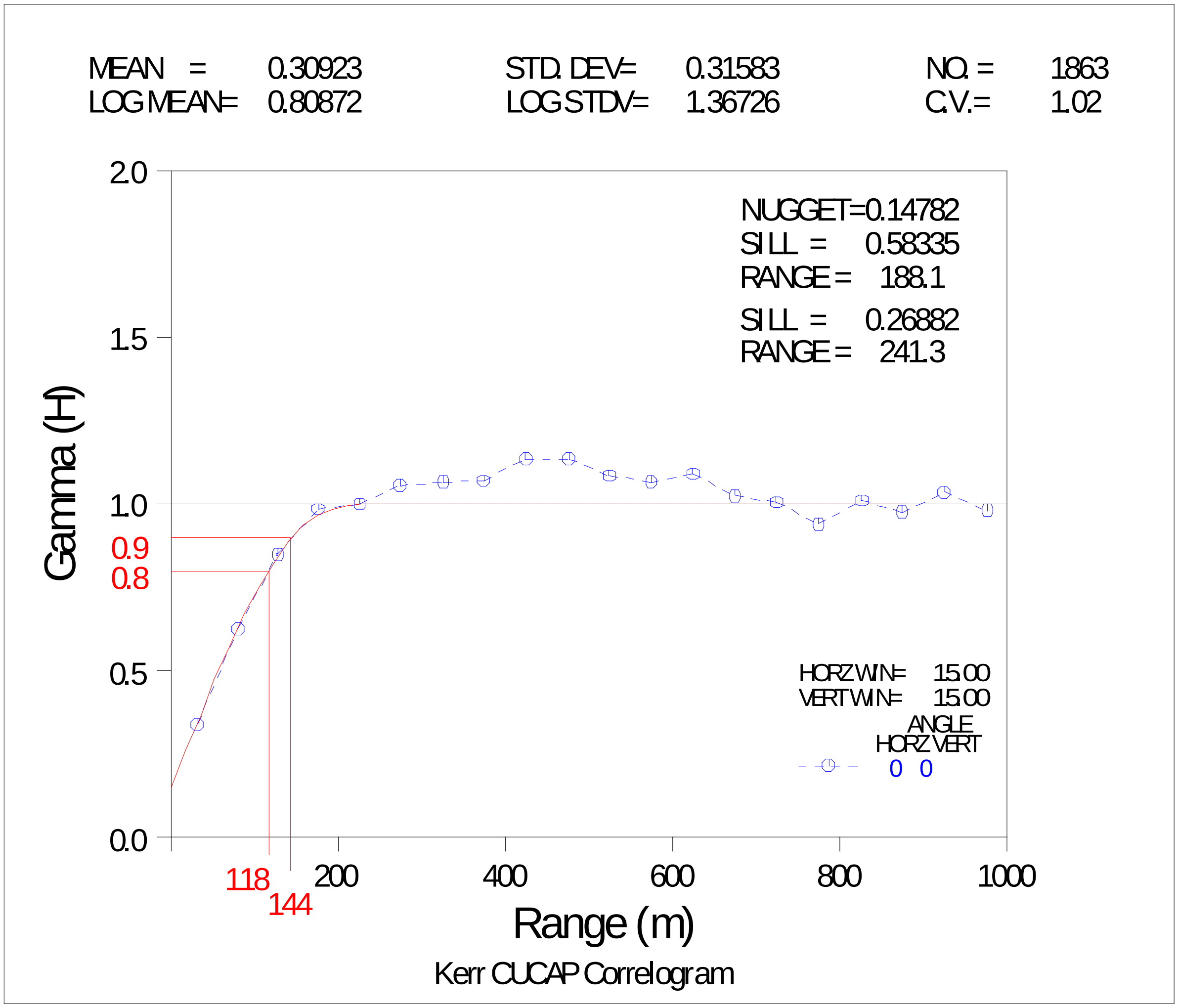


Figure 17-14: Sulphurets Cu Grade Correlogram

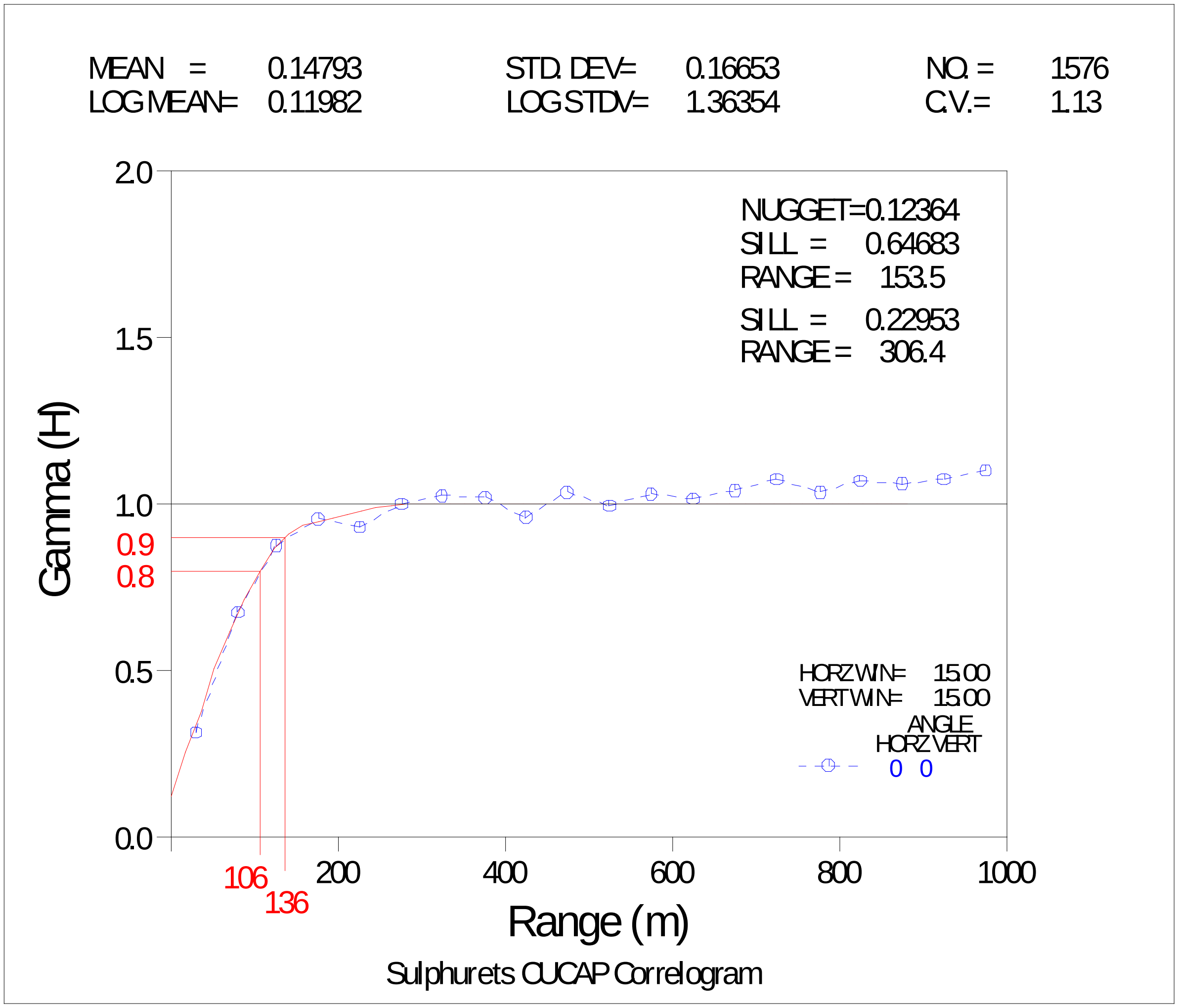


Figure 17-15: Mitchell Cu Grade Correlogram

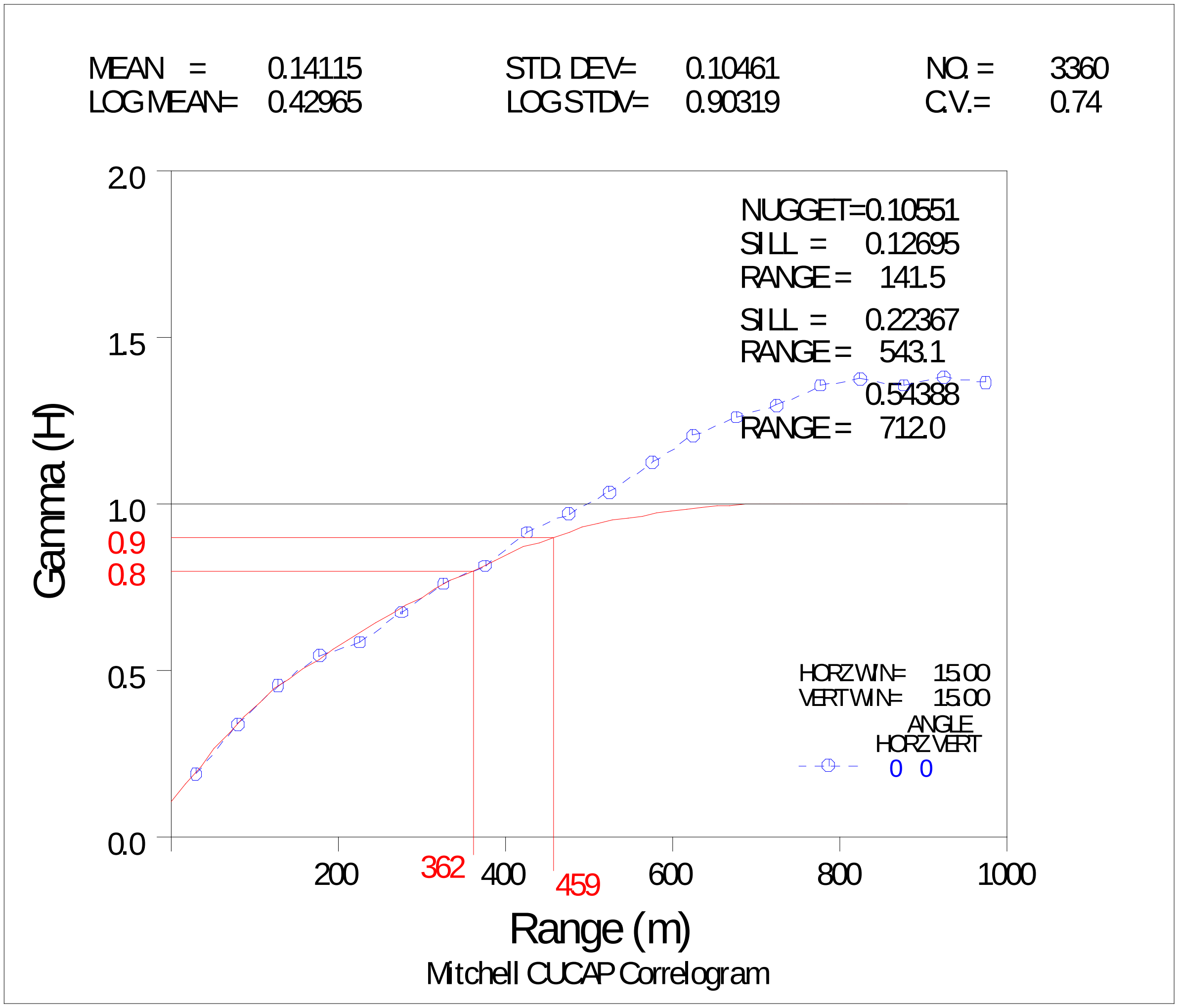


Figure 17-16: Iron Cap Cu Grade Correlogram

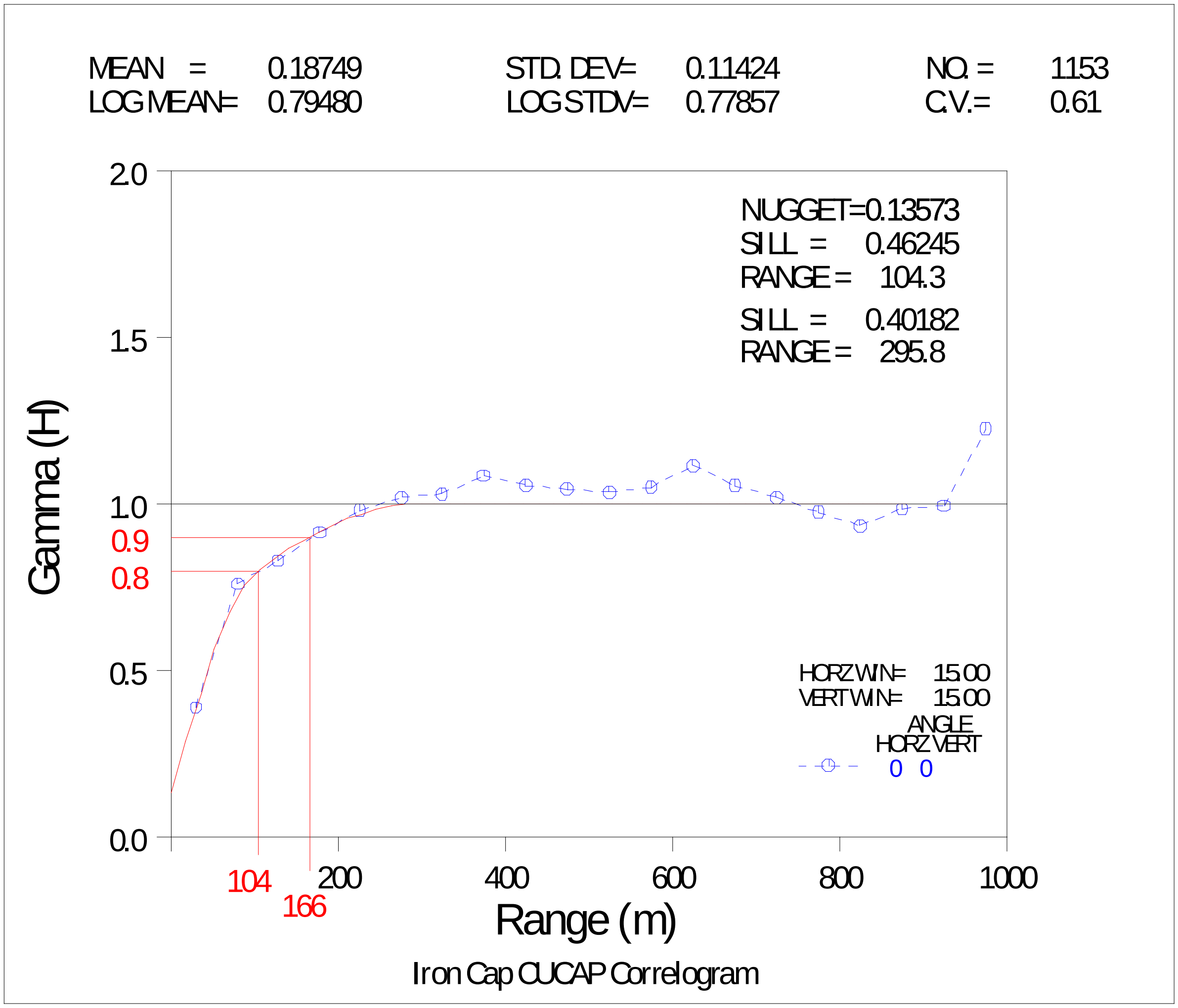


Figure 17-17: Kerr 0.5 g/t Gold Equivalent Correlogram

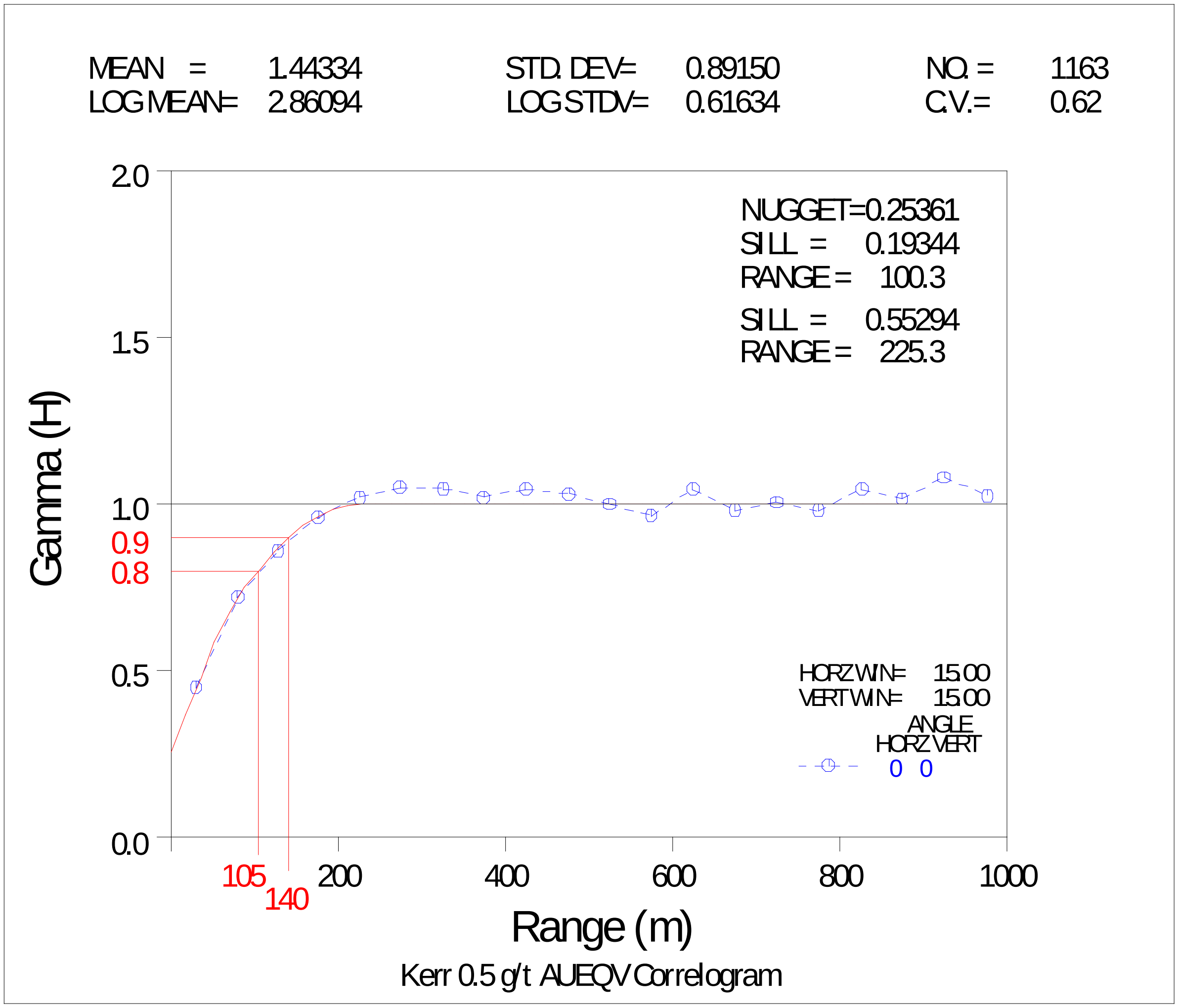


Figure 17-18: Sulphurets 0.5 g/t Gold Equivalent Correlogram

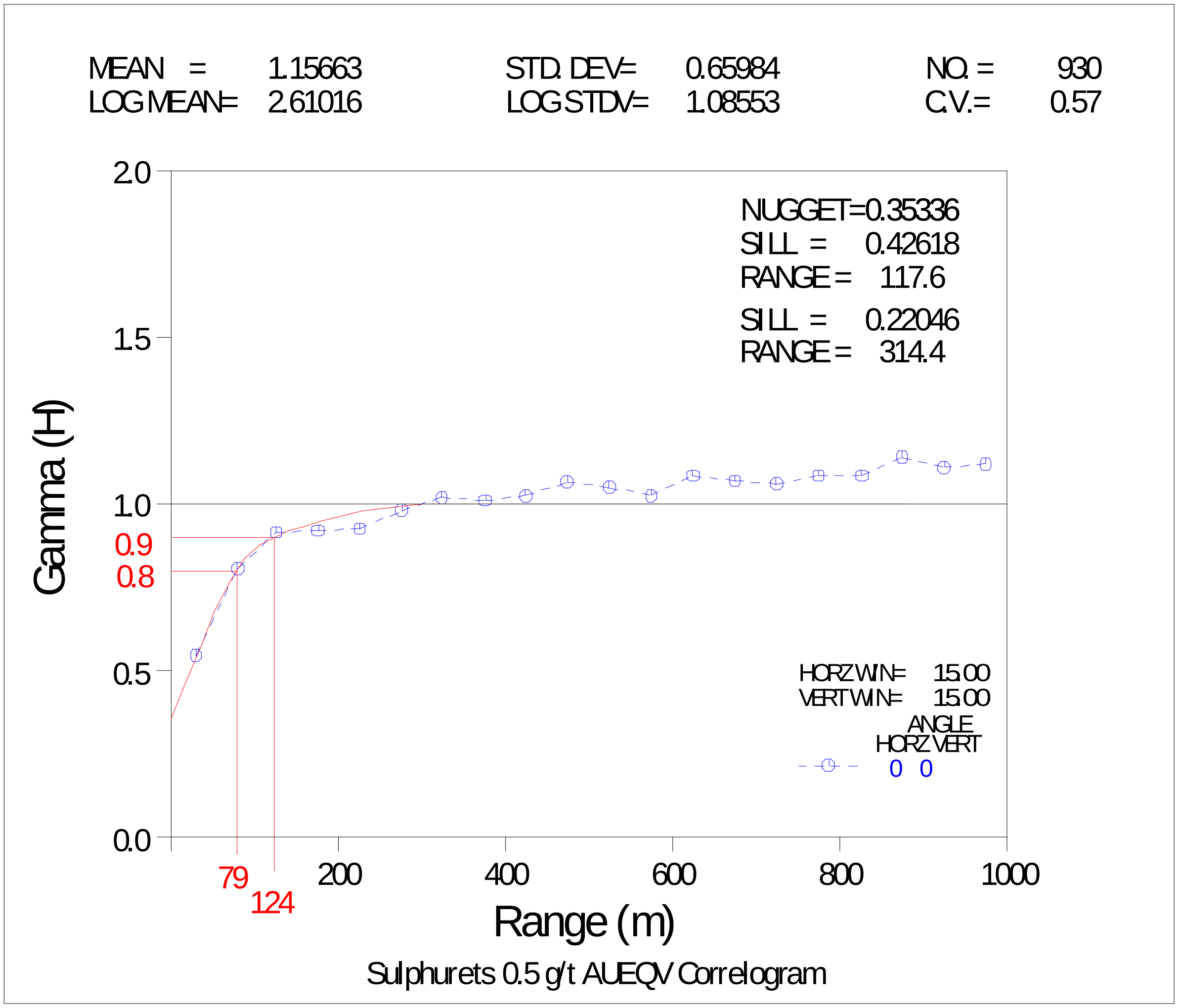


Figure 17-19: Mitchell 0.5 g/t Gold Equivalent Correlogram

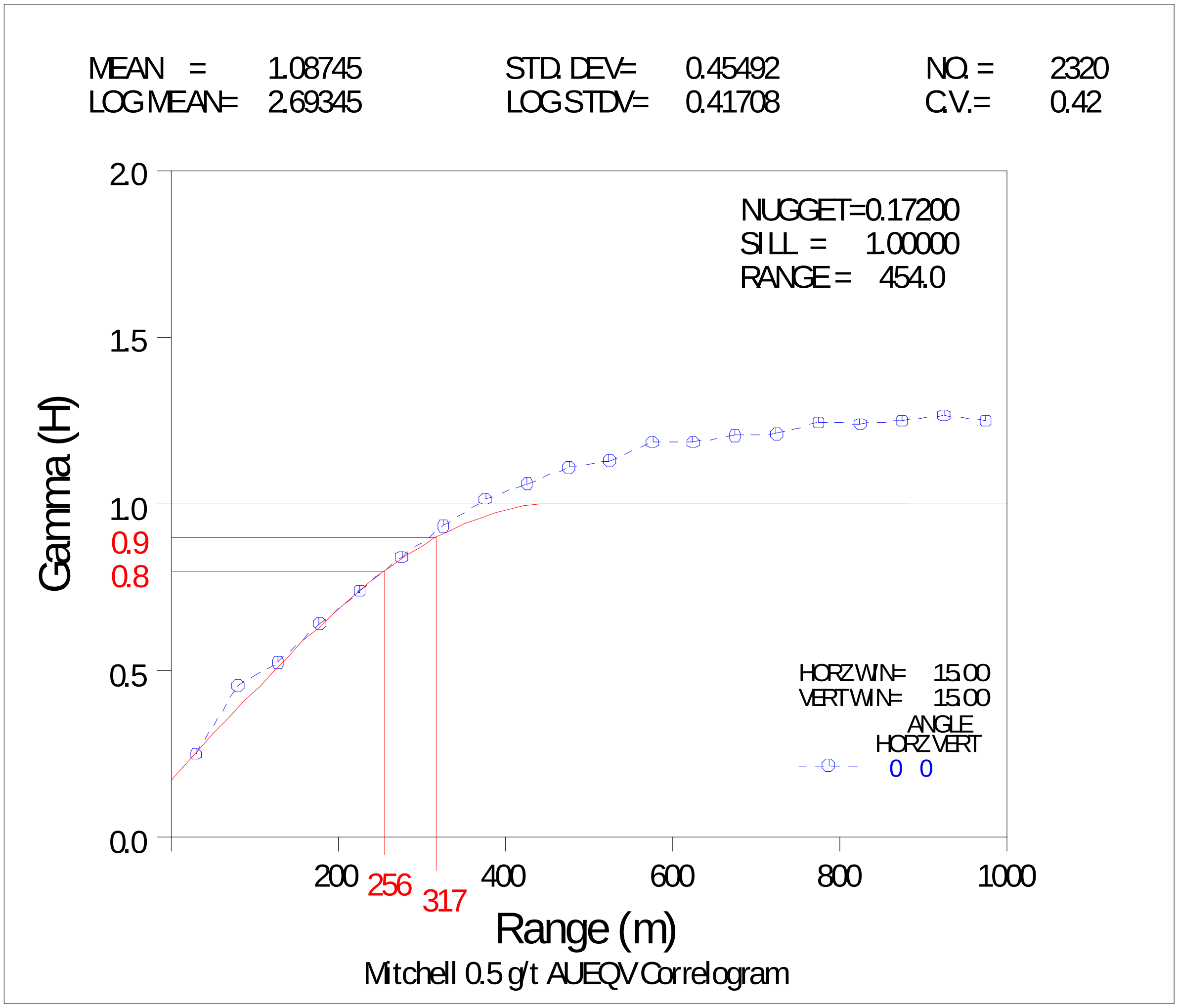
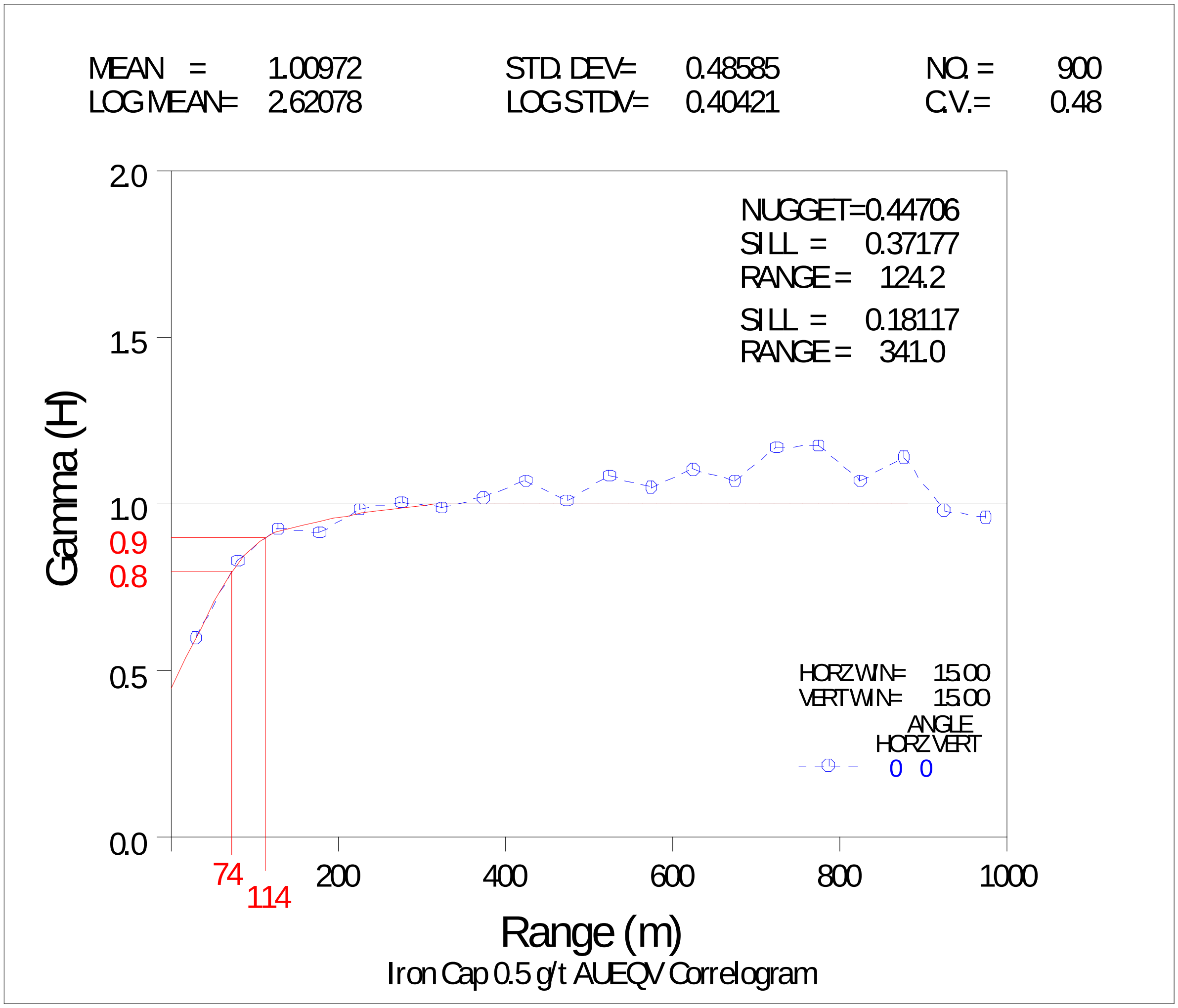


Figure 17-20: Iron Cap 0.5 g/t Gold Equivalent Correlogram



The correlograms shown in Figures 17-9 through 17-20 were modeled as either single structure spherical or nested spherical models. Total ranges for gold are 159m, 396m, 555m, and 279m for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. At 80% of the total sill, gold ranges of 47m, 155m, 325m, and 111m were interpreted for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. Total ranges for copper are 241m, 306m, 712m, and 296m for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. At 80% of the total sill, copper ranges of 118m, 106m, 362m, and 104m were interpreted for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. Total ranges for gold equivalent grades are 225m, 314m, 454m, and 314m for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. At 80% of the total sill, gold equivalent ranges of 105m, 79m, 256m, and 74m were interpreted for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively.

17.7 Grade Estimation Parameters

RMI constructed a three-dimensional block model using MineSight®, a widely recognized commercial mine engineering software package. Table 17-16 summarizes various block parameters for this non-rotated model which uses NAD83 UTM coordinates.

Table 17-16: KSM Block Model Dimensions

Parameter	NAD83 Coordinates		Block	Number of	Areal
	Minimum	Maximum	Size (m)	Blocks	Extent (m)
Easting	420,500	425,900	25	216	5,400
Northing	6,257,800	6,269,000	25	448	11,200
Elevation	-210	2,145	15	157	2,355

Block gold, silver, copper, and molybdenum grades were estimated by two methods: 1) inverse distance weighting and 2) nearest neighbor. Gold and copper resources summarized in this report are based on inverse distance squared or inverse distance cubed methods.

A multi-pass estimation strategy was used for gold, silver, copper and molybdenum. The first and second estimation passes required two or more drill holes to estimate block grades while the final pass acted as "cleanup" run that filled un-estimated blocks by using a larger search ellipse and requiring fewer drill holes. The inverse distance estimation plans used strict block/composite matching.

Tables 17-17 summarizes the key estimation parameters that were used to estimate block gold, silver, and copper grades for the Kerr zone. No molybdenum assays were available for a significant portion of the Kerr drill hole data so no estimate was made for that metal. The estimate of Kerr block grades was constrained (controlled) by matching block and drill hole composite alteration codes (see Table 17-13 for definition of alteration codes). Once a block was estimated, it was flagged so it would not be re-estimated by subsequent runs.

Table 17-17: Kerr Block Grade Estimation Parameters

Estimation	Alteration	ID	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
Pass	Codes	Power	X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	1, 2, 4, 5, 6	3	75	75	15	3	6	2	20	0	60
2	1, 2, 4, 5, 6	3	125	125	25	3	6	2	20	0	60
3	1, 2, 4, 5, 6	3	200	200	40	1	3	1	20	0	60
1	29	3	100	100	40	3	6	2	20	0	60
2	29	3	100	100	40	1	3	1	20	0	60

Notes:
ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule

The number of composites used to estimate block gold and copper grades were stored along with the distance to the closest composite and the number of drill holes used to estimate the block.

Table 17-18 summarizes the parameters used to estimate block gold and silver grades for the Sulphurets zone.

Table 17-18: Sulphurets Au/Ag Grade Estimation Parameters

Estimation	AUZON	ID	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
Pass	Codes	Power	X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	2, 3, 4, 5, 13	3	75	75	15	3	6	2	35	15	35
2	2, 3, 4, 5, 13	3	125	125	25	3	6	2	35	15	35
3	2, 3, 4, 5, 13	3	200	200	25	1	3	1	35	15	35
1	29	3	75	75	15	3	6	2	35	15	35
2	29	3	125	125	25	1	3	1	35	15	35
1	14, 29	3	75	75	15	3	6	2	35	15	35
2	14, 29	3	125	125	25	1	3	1	35	15	35

Notes:
ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule

The estimate of Sulphurets gold and silver block grades was constrained (controlled) by matching block and drill hole AUZON composite codes (see Table 17-14 for definition of AUZON codes). The last two interpolation runs shown in Table 17-18 estimated block grades above the Sulphurets Thrust Fault (STF) while all of the prior runs estimated blocks below the STF. The number of composites and drill holes used to estimate block gold and silver grades were stored along with the distance to the closet composite.

Table 17-19 summarizes the parameters used to estimate block gold and silver grades for the Mitchell zone. Similar to Sulphurets, AUZON codes were used to constrain the estimate of block gold/silver grades for the Mitchell zone. In addition to AUZON codes, block/composite position relative to the Mitchell Thrust Fault (MTF) was also used to limit or constrain the estimate of block grades. The field "FLTAR" shown in Table 17-19 shows two codes where 5 means above the MTF and 6 means below the MTF. Similar to the Kerr and Sulphurets estimation plan, the number of composites and drill holes used to estimate block grades were stored in addition to the distance of the closest composite.

Table 17-19: Mitchell Au/AG Grade Estimation Parameters

Estimation Pass	AUZON	ID Power	FLTAR	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
				X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	2	2	6	125	125	30	3	8	2	75	0	40
2	2	2	6	250	250	60	3	8	2	75	0	40
3	2	2	6	375	375	90	3	8	2	75	0	40
4	2	2	6	500	500	120	1	3	1	75	0	40
1	3	2	6	250	250	60	3	8	2	275	0	65
2	3	2	6	375	375	90	1	3	1	275	0	65
1	4	2	6	250	250	60	3	8	2	275	0	65
2	4	2	6	375	375	90	1	3	1	275	0	65
1	29	2	5	150	150	45	3	8	2	75	0	40
2	29	2	5	300	300	100	1	3	1	75	0	40
1	29	2	6	150	150	45	3	8	2	75	0	40
2	29	2	6	75	75	15	1	3	1	75	0	40
1	1	2	5	125	125	30	3	8	2	75	0	40
2	1	2	5	250	250	60	3	8	2	75	0	40
3	1	2	5	375	375	90	3	8	2	75	0	40
4	1	2	5	500	500	120	1	3	1	75	0	40

Notes:
ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule

Table 17-20 summarizes the key estimation parameters that were used to estimate block gold, silver, copper, and molybdenum grades using inverse distance squared methods for the Iron Cap zone.

Table 17-20: Iron Cap Grade Estimation Parameters

Estimation Pass	AUZON Codes	ID Power	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
			X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	6-12, 29	2	75	75	25	3	8	2	45	0	45
2	6-12, 30	2	150	150	50	3	8	2	45	0	45
3	6-12, 31	2	150	150	50	1	3	1	45	0	45

Notes:
ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule

Table 17-21 summarizes the key estimation parameters that were used to estimate block copper and molybdenum grades using inverse distance methods for the Sulphurets zone. The plan used copper zones (CUZON) and fault block (FLTAR) codes to constrain the estimate of block grades. CUZON codes are described in Table 17-15. FLTAR codes 1 and 2 refer to blocks/drill holes below and above the STF, respectively. Like the previously described estimation plans, the number of composites and drill holes were stored along with the distance to the closest composite.

Table 17-21: Sulphurets Cu/Mo Grade Estimation Parameters

Estimation Pass	CUZON	ID Power	FLTAR	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
				X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	3, 4, 5, 13	3	2	75	75	15	3	6	2	35	15	35
2	3, 4, 5, 14	3	2	125	125	25	3	6	2	35	15	35
3	3, 4, 5, 15	3	2	200	200	25	1	3	1	35	15	35
1	14	3	1	75	75	15	3	6	2	35	15	35
2	14	3	1	175	175	25	1	3	1	35	15	35
1	29	3	2	75	75	15	3	6	2	35	15	35
2	29	3	2	175	175	25	1	3	1	35	15	35
1	29	3	1	75	75	15	3	6	2	35	15	35
2	29	3	1	175	175	25	1	3	1	35	15	35

Notes:
ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule

Table 17-22 summarizes the key estimation parameters that were used to estimate block copper grades using inverse distance methods for the Mitchell zone. The plan used copper zones (CUZON) and fault block (FLTAR) codes to constrain the estimate of block grades. CUZON codes are described in Table 17-15. FLTAR codes 5 and 6 refer to blocks/drill holes above and below the MTF, respectively. Like the previously described estimation plans, the number of composites and drill holes were stored along with the distance to the closest composite.

Table 17-22: Mitchell Cu Grade Estimation Parameters

Estimation Pass	Population	ID Power	FLTAR	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
				X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	CUZON 1	2	5	250	250	60	3	8	2	75	0	40
2	CUZON 1	2	5	500	500	120	1	3	1	75	0	40
1	CUZON 2	2	6	250	250	60	3	8	2	75	0	40
2	CUZON 2	2	6	500	500	120	1	3	1	75	0	40
1	CUZON 3	2	6	250	250	60	3	8	2	275	0	65
2	CUZON 3	2	6	375	375	90	1	3	1	275	0	65
1	CUZON 4	2	6	250	250	60	3	8	2	275	0	65
2	CUZON 4	2	6	500	500	120	1	3	1	275	0	65
1	LITH 3	2	5	150	150	45	3	8	2	75	0	40
2	LITH 3	2	5	75	75	15	1	3	1	75	0	40
1	CUZON 29	2	6	300	300	100	1	3	1	75	0	40
1	CUZON 29	2	6	150	150	45	3	8	2	75	0	40
2	CUZON 29	2	5	150	150	45	1	3	1	75	0	40

Notes:
ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule

Table 17-23 summarizes the key estimation parameters that were used to estimate block molybdenum grades using inverse distance squared methods for the Mitchell zone. The estimate of block molybdenum grades were constrained by a three-dimensional molybdenum grade shell wireframe that was constructed using a 50 ppm cutoff grade. Blocks located inside and outside of that wireframe could only be estimated by drill hole composites located inside or outside of the wireframe, respectively.

Table 17-23: Mitchell Mo Grade Estimation Parameters

Estimation Pass	ID Power	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	2	300	300	300	1	3	1	20	0	45
2	2	250	250	60	3	8	2	20	0	45

Notes:
ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule

17.8 Grade Model Verification

Estimated block grades were verified by visual and statistical methods. The author visually compared estimated block grades (gold, silver, copper, and molybdenum) with drill hole composite grades. In the author’s opinion there is a reasonable comparison between the drill hole composite grades and the estimated block grades. Figures 17-21 and 17-22 are east-west cross sections through the Kerr block model drawn at northing coordinate 6,259,600. These figures show estimated block/composite gold grades (Figure 17-21) and block/composite copper grades (Figure 17-23). Figures 17-23 and 17-24 are block model level maps drawn at the 1200m elevation through the Kerr model showing estimated block/composite gold and copper grades, respectively. Figures 17-25 and 17-26 are northwest-southeast cross sections through the Sulphurets block model drawn at Section 23. These figures show estimated block/composite gold grades (Figure 17-25) and block/composite copper grades (Figure 17-26). Figures 17-27 and 17-28 are block model level maps drawn at the 1275m elevation through the Sulphurets model showing estimated block/composite gold and copper grades, respectively. Figures 17-29 and 17-30 are northeast-southwest cross sections through the Mitchell block model drawn at Section 11. These figures show estimated block/composite gold grades (Figure 17-29) and block/composite copper grades (Figure 17-30). Figures 17-31 and 17-32 are block model level maps drawn at the 660m elevation through the Mitchell model showing estimated block/composite gold and copper grades, respectively. Figures 17-33 and 17-34 are northwest-southeast cross sections through the Iron Cap block model drawn at Section 50,700. These figures show estimated block/composite gold grades (Figure 17-33) and block/composite copper grades (Figure 17-34). Figures 17-35 and 17-36 are block model level maps drawn at the 1395m elevation through the Iron Cap model showing estimated block/composite gold and copper grades, respectively.

The heavy dashed black line shown on the block model cross sections and level plans shown in Figures 17-21 through 17-36 represents a conceptual pit generated by RMI using gold and copper prices of $1000/oz and $3.00/lb, respectively.

Figure 17-21: Kerr Au Block Model Section 6,259,600 North

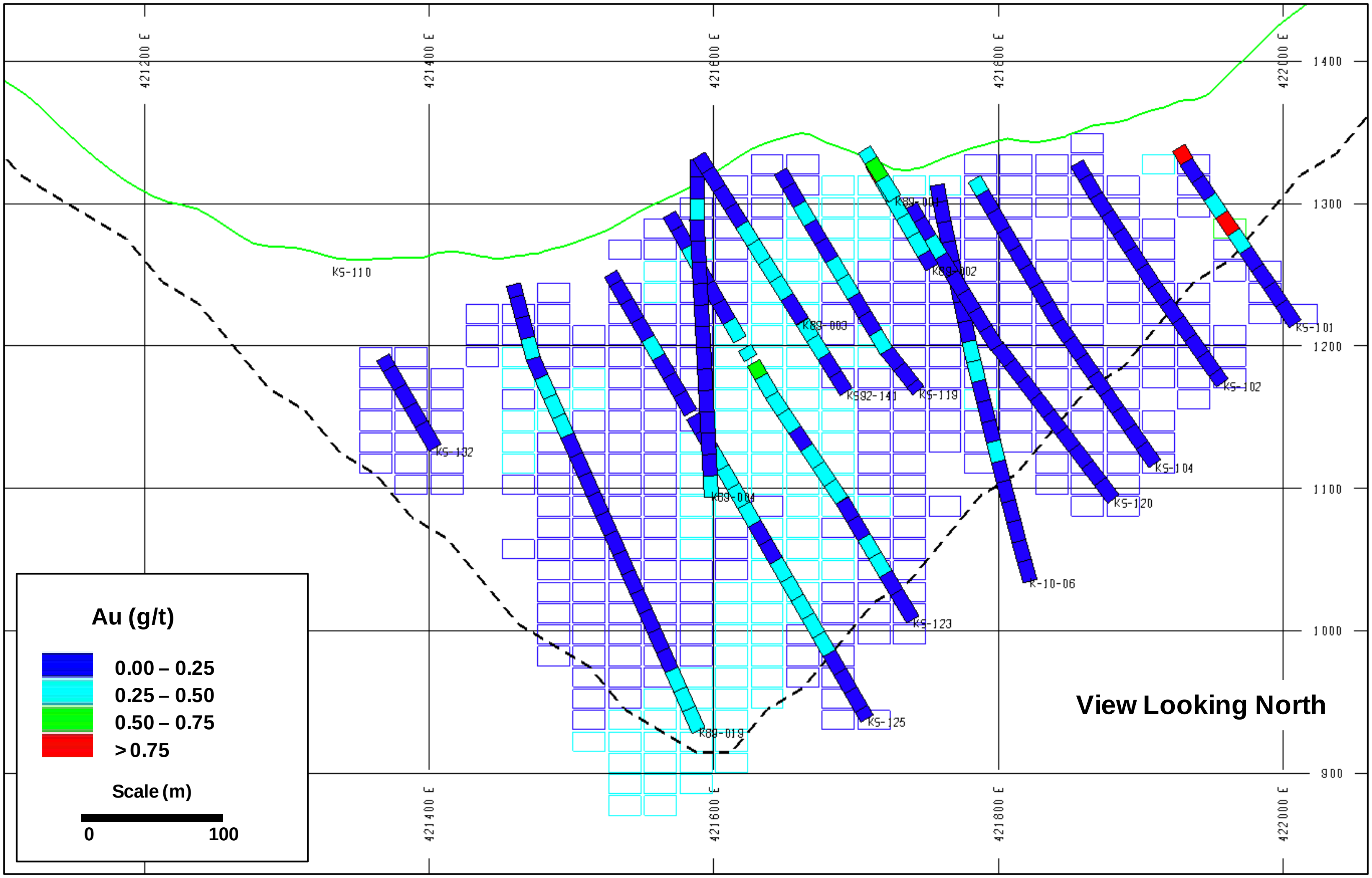


Figure 17-22: Kerr Cu Block Model Section 6,259,600 North

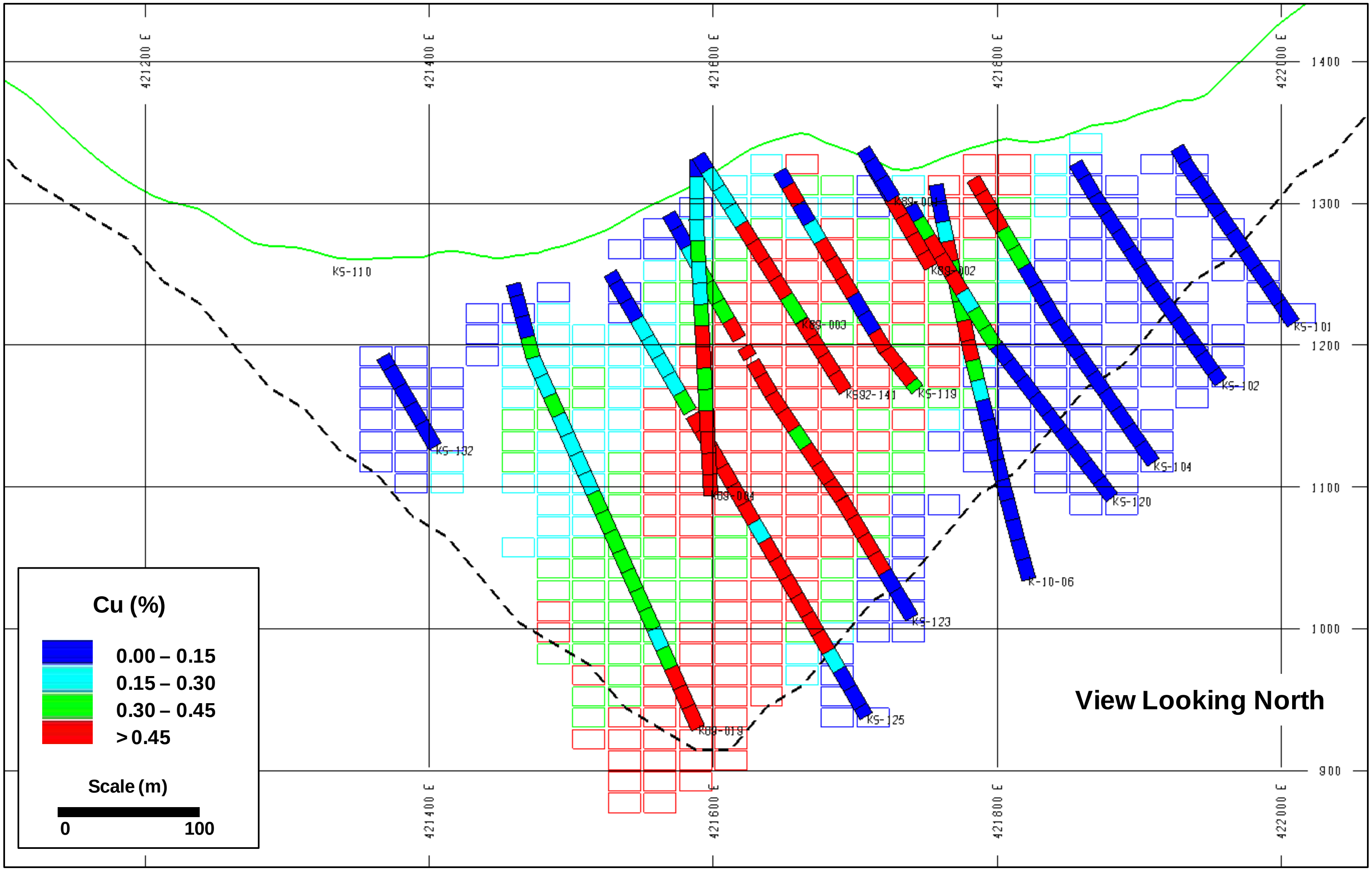


Figure 17-23: Kerr Au Block Model - 1200 Level

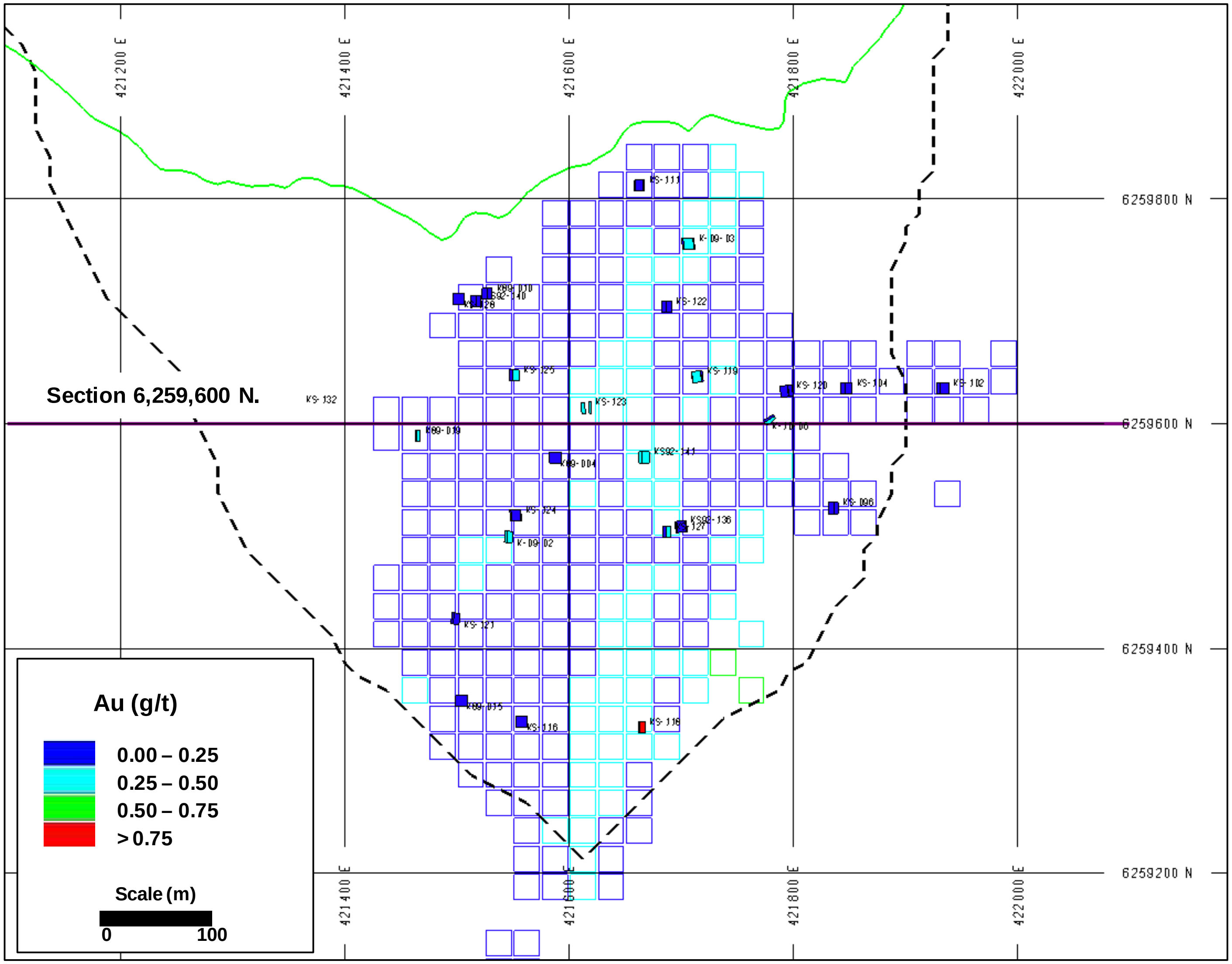


Figure 17-24: Kerr Cu Block Model - 1200 Level

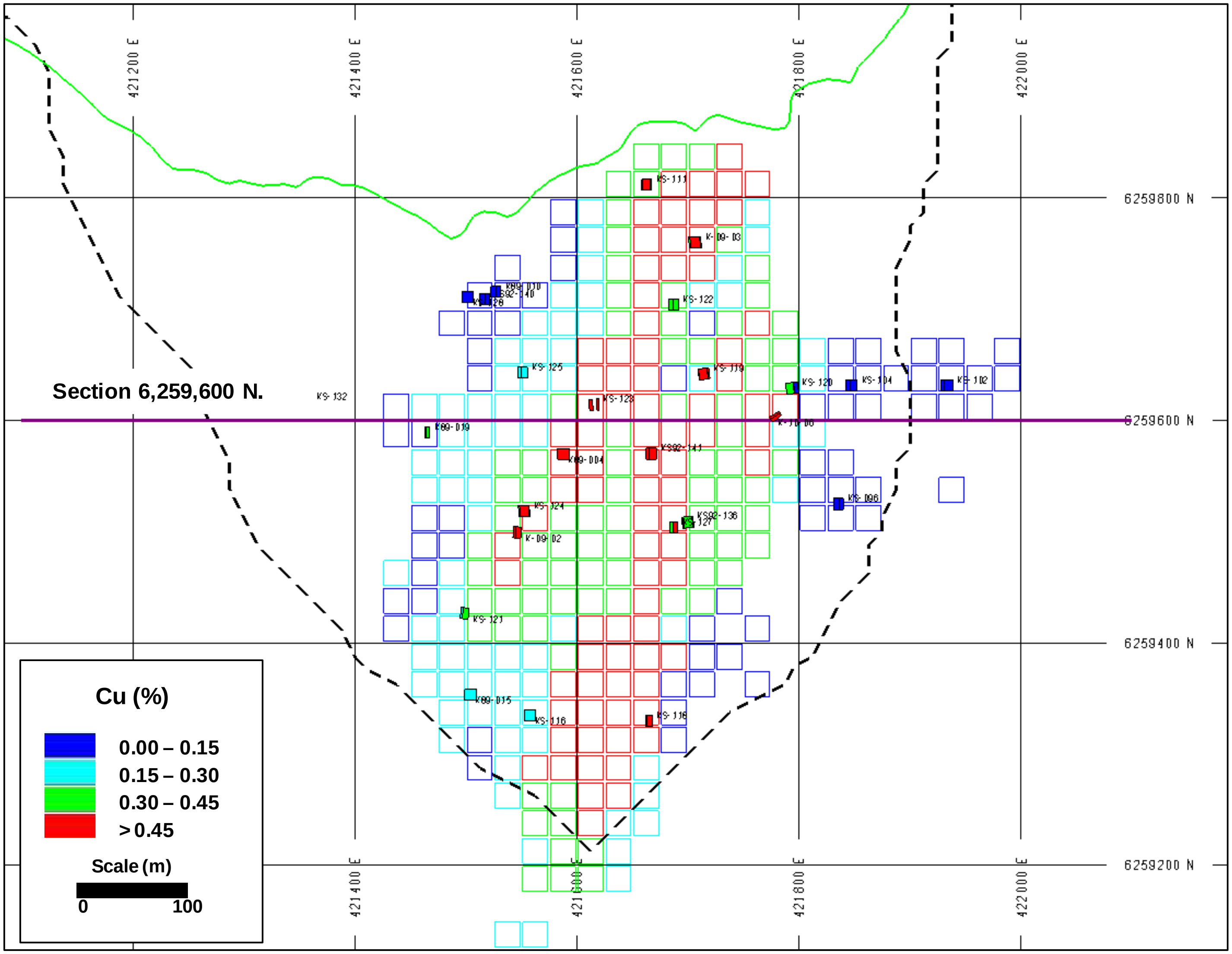


Figure 17-25: Sulphurets Au Block Model Cross Section 23

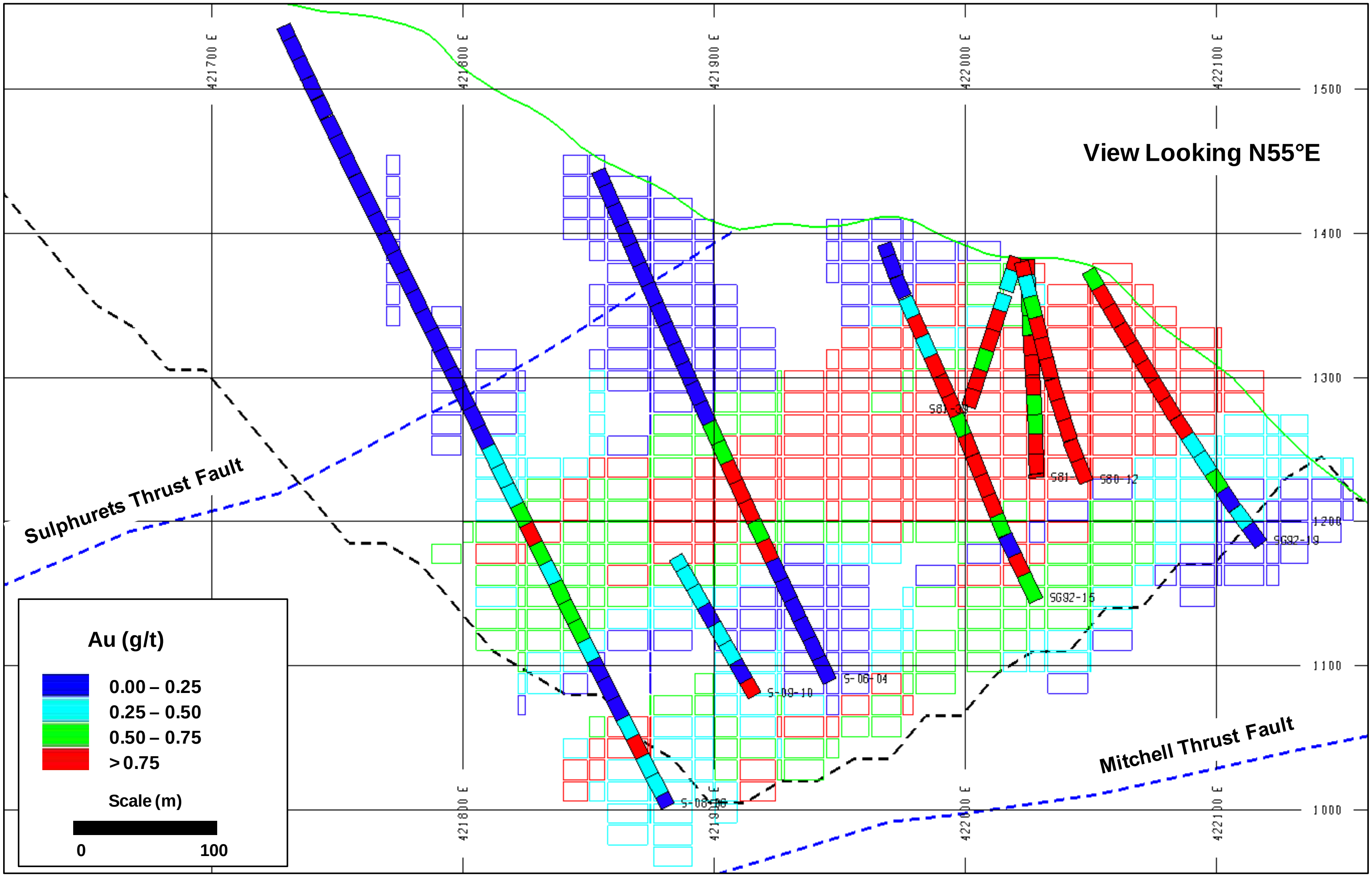


Figure 17-26: Sulphurets Cu Block Model Cross Section 23

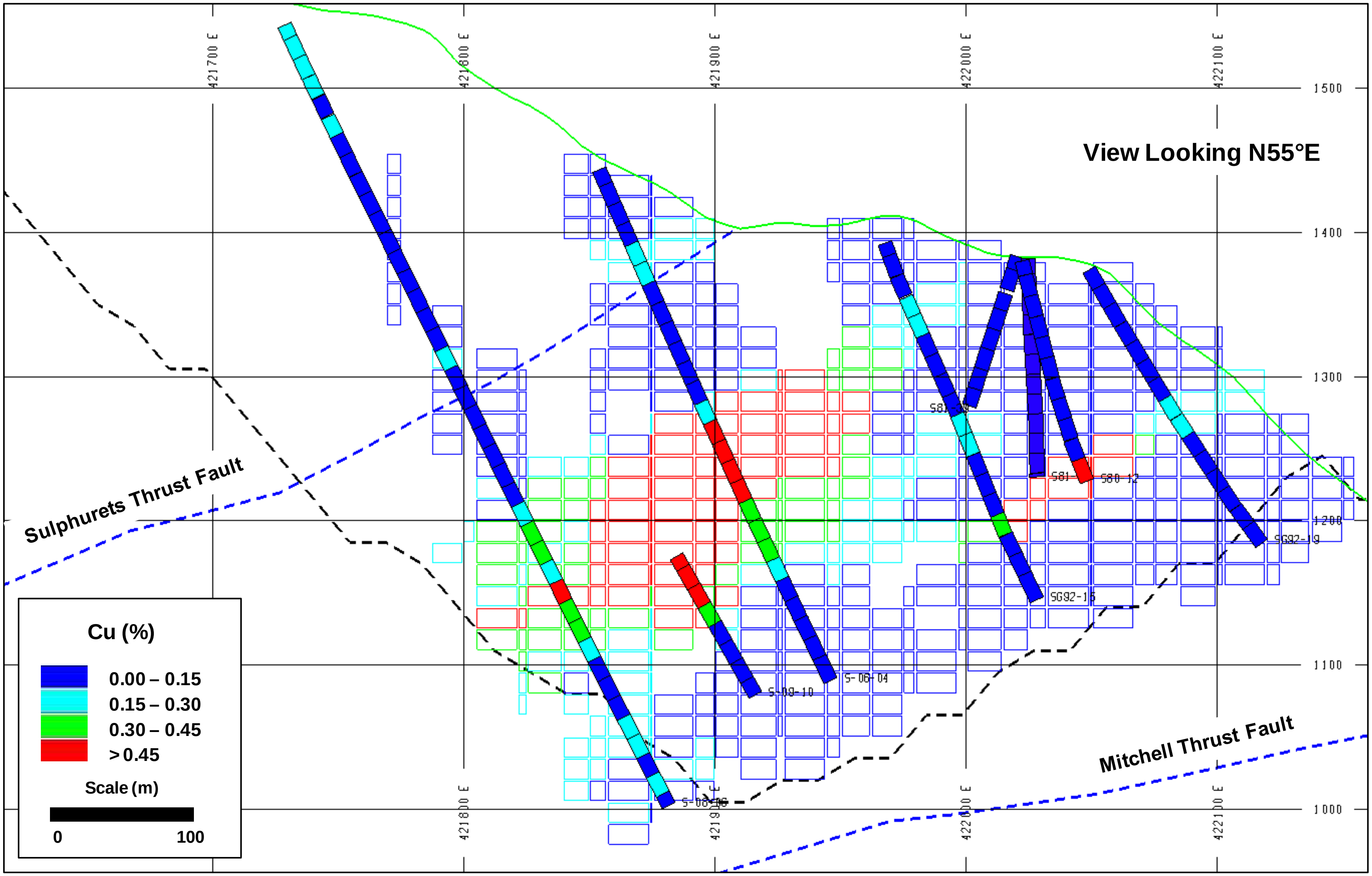


Figure 17-27: Sulphurets Au Block Model - 1275 Level

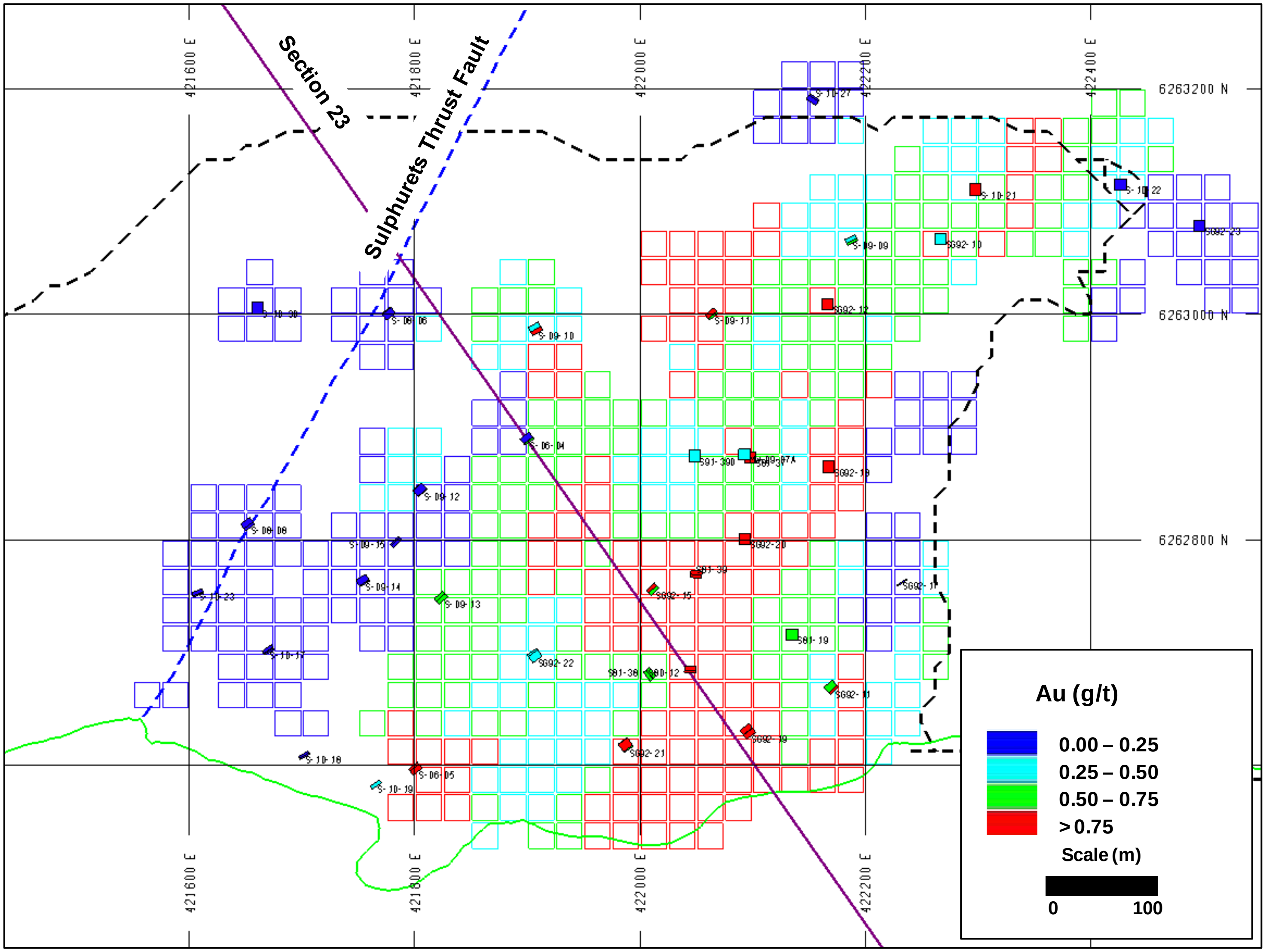


Figure 17-28: Sulphurets Cu Block Model - 1275 Level

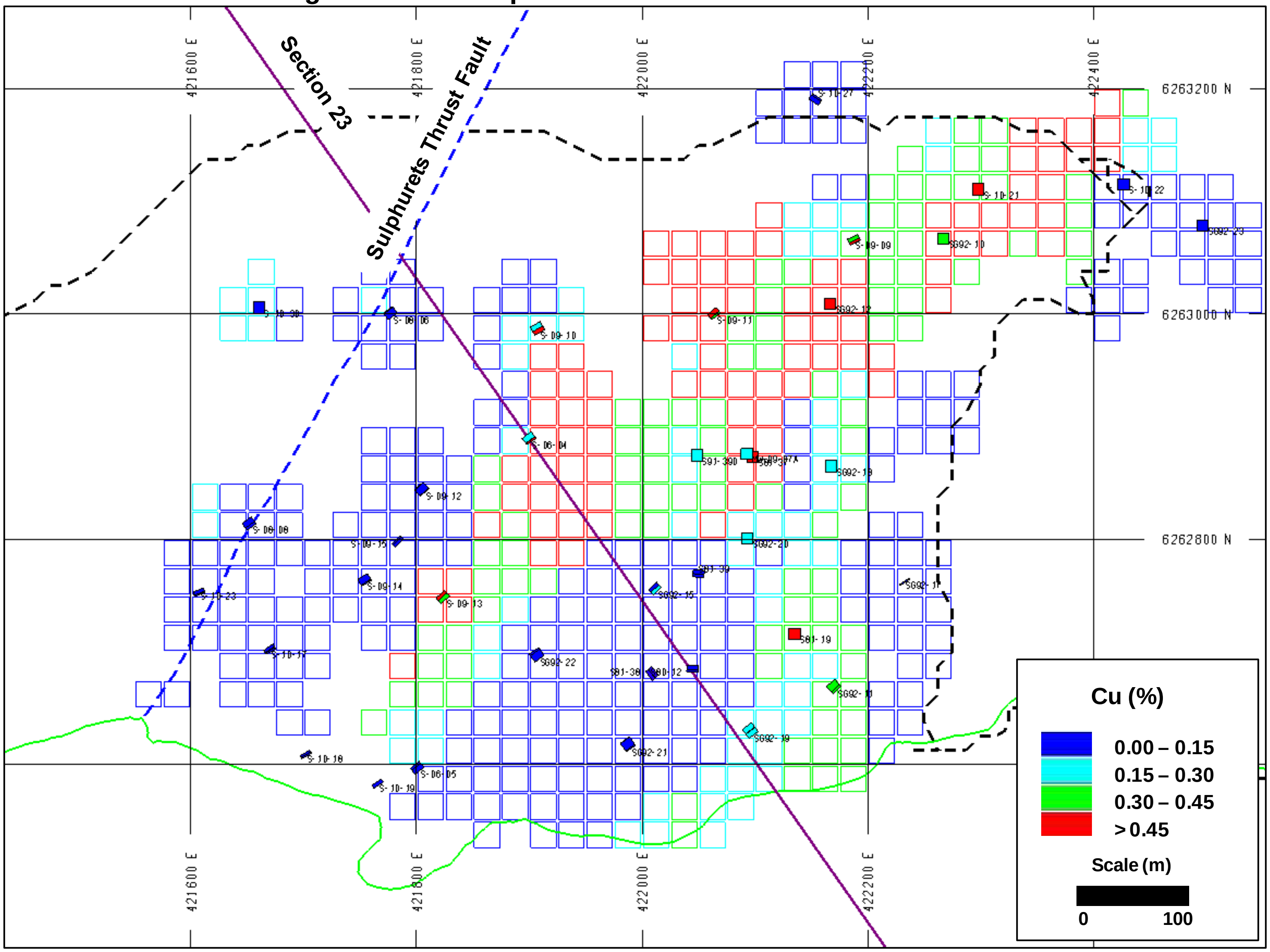


Figure 17-29: Mitchell Au Block Model Cross Section 11

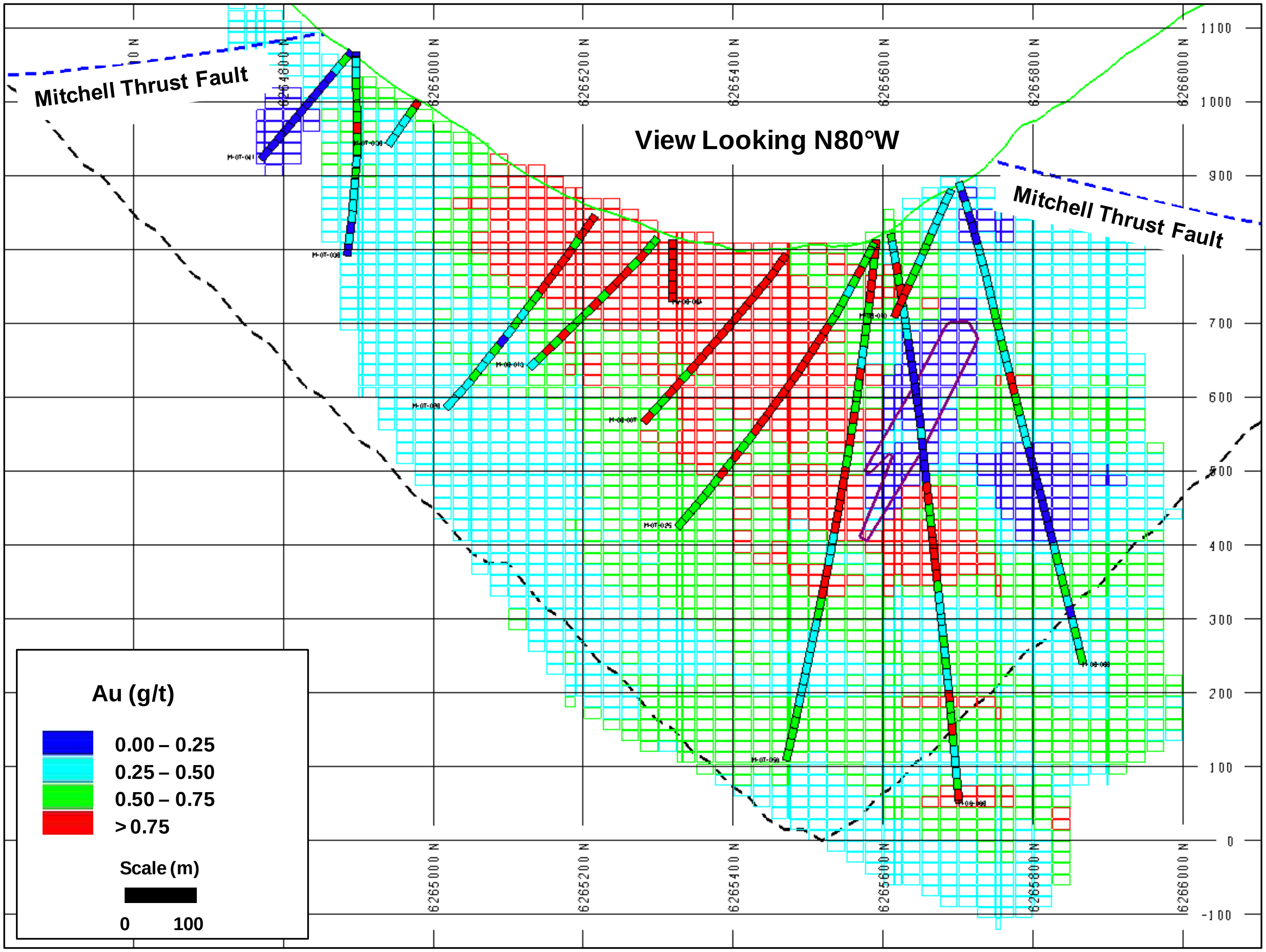


Figure 17-30: Mitchell Cu Block Model Cross Section 11

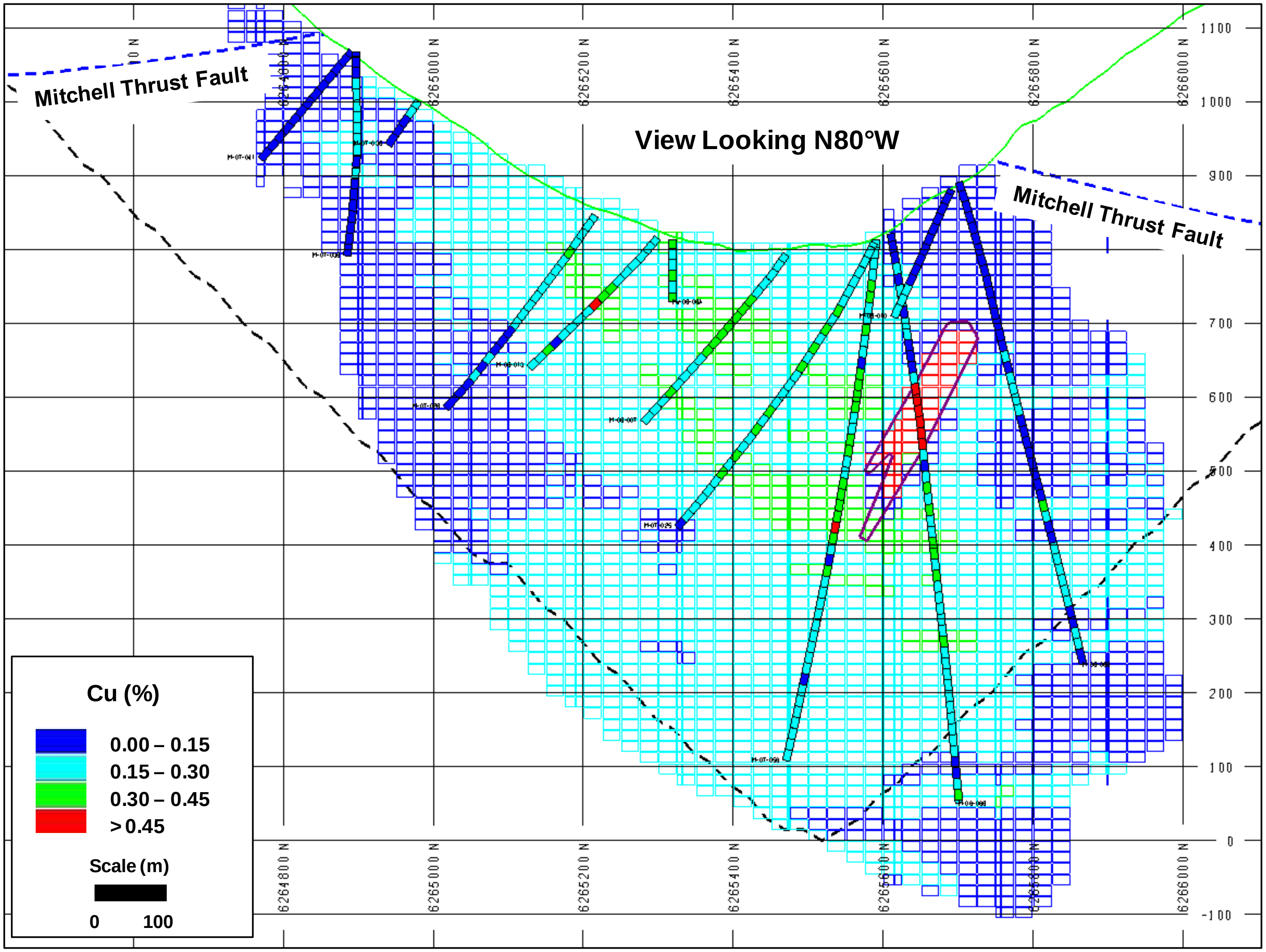


Figure 17-31: Mitchell Au Block Model - 660 Level

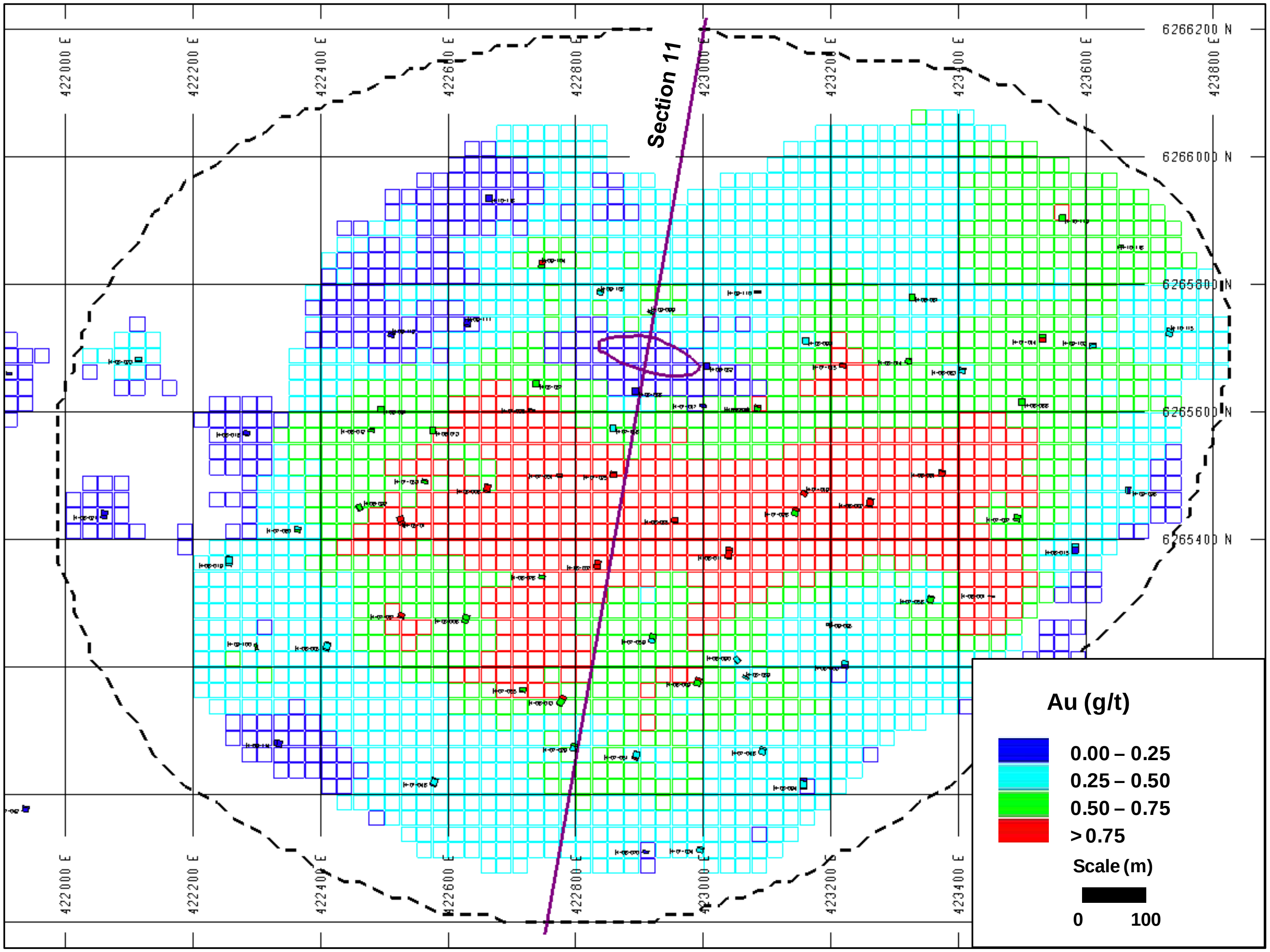


Figure 17-32: Mitchell Cu Block Model - 660 Level

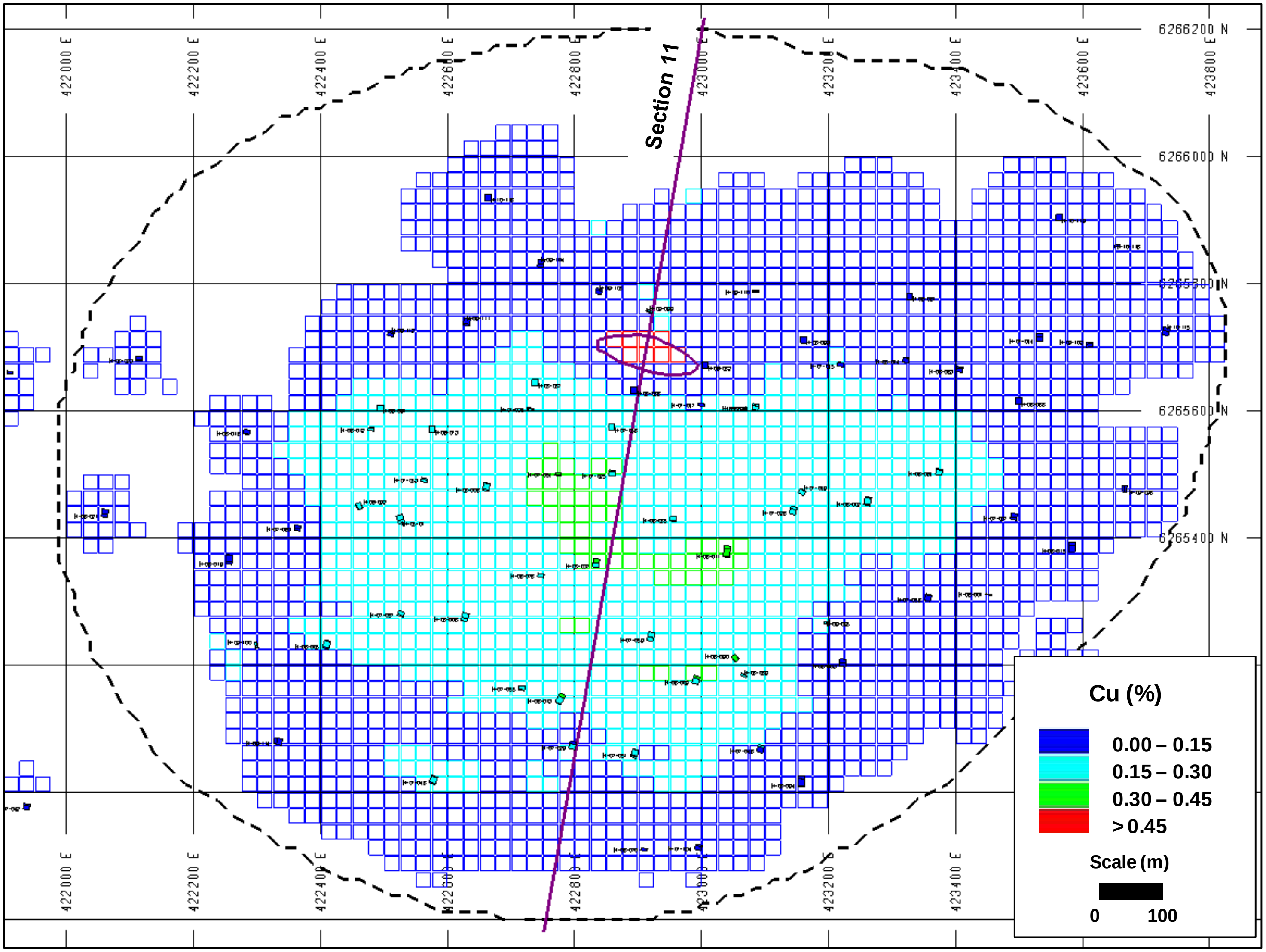


Figure 17-33: Iron Cap Au Block Model - Section 50,700

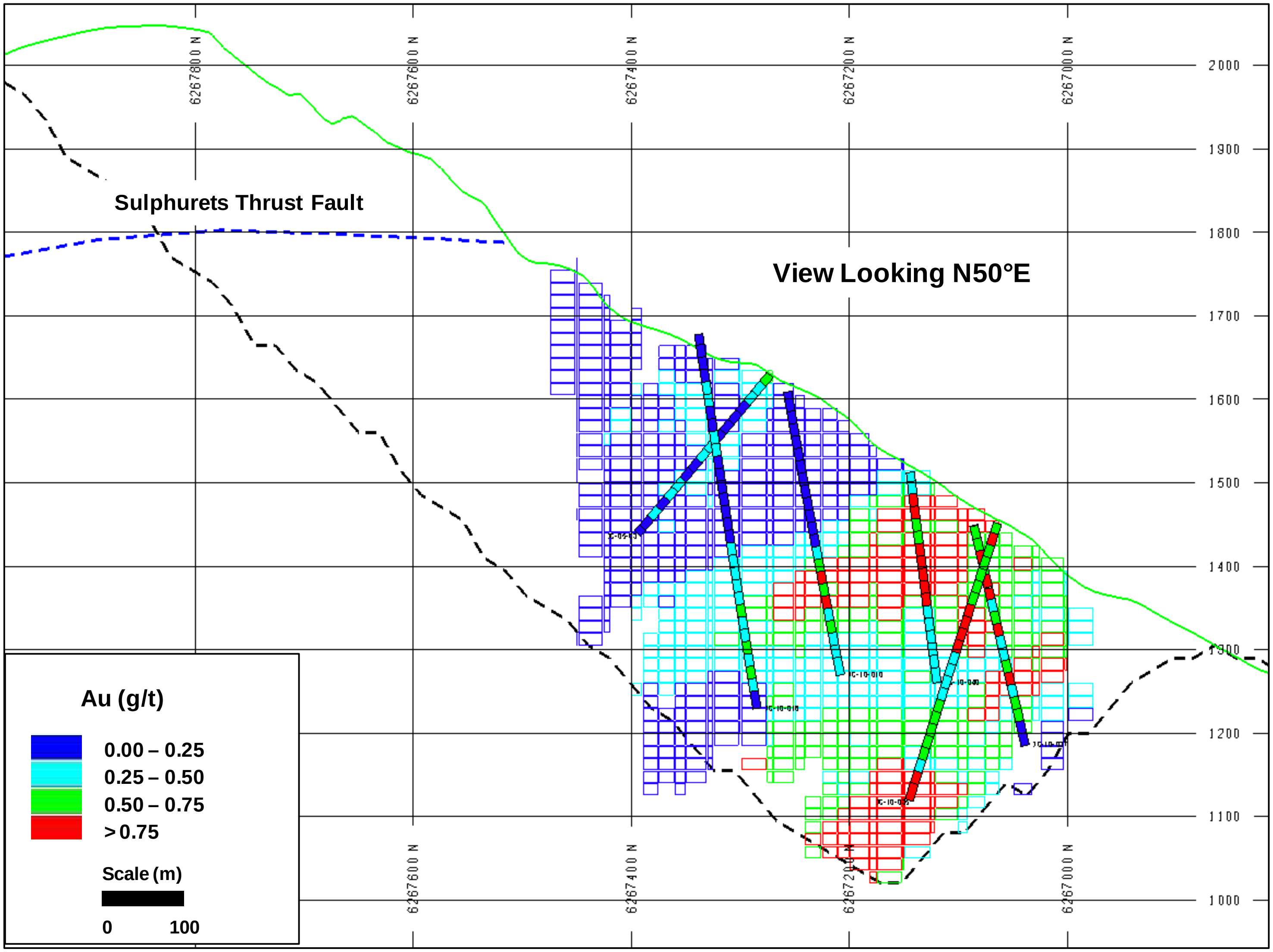


Figure 17-34: Iron Cap Cu Block Model - Section 50,700

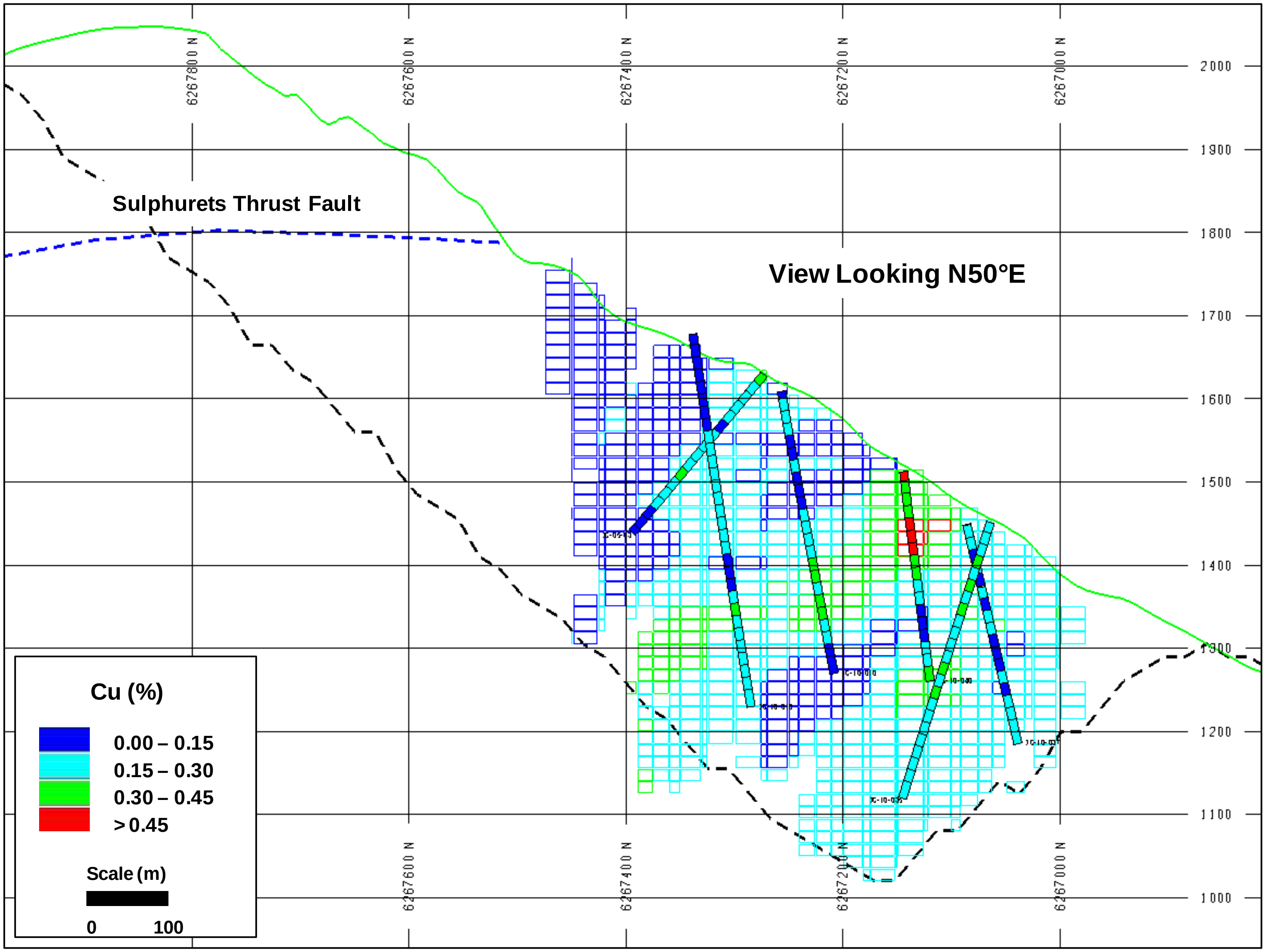


Figure 17-35: Iron Cap Au Block Model - 1395 Level

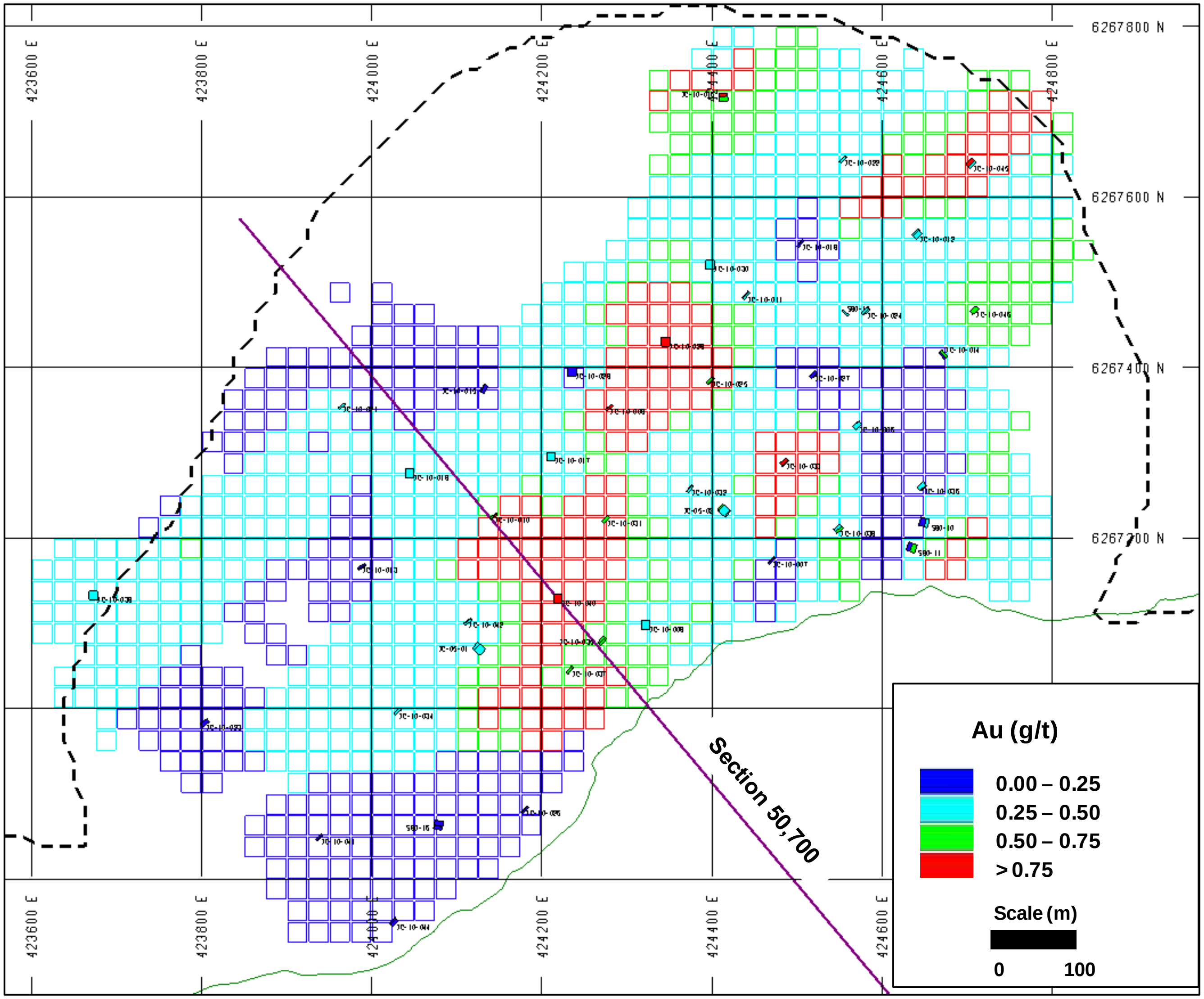
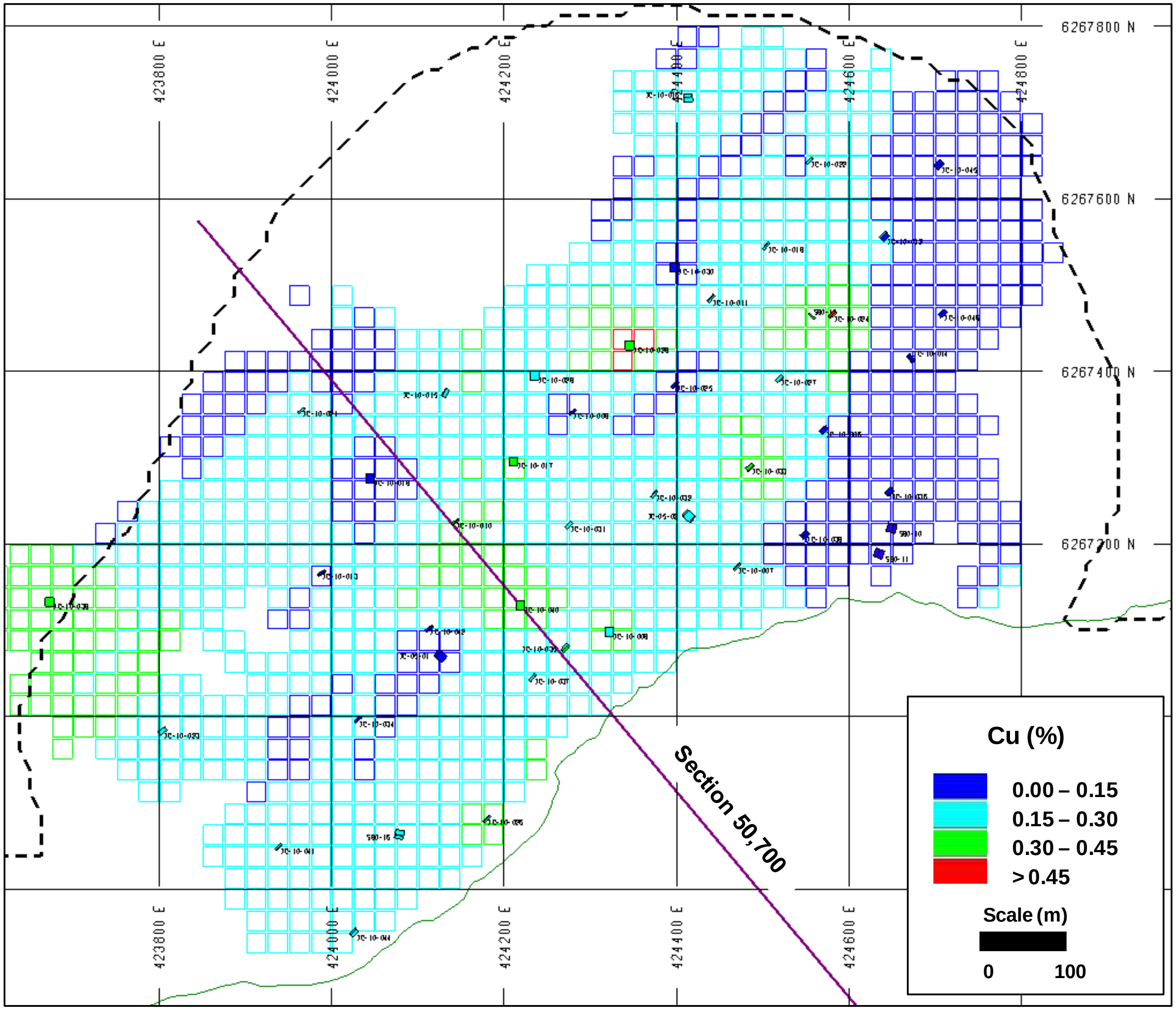


Figure 17-36: Iron Cap Cu Block Model - 1395 Level



The author generated nearest neighbor models for gold, copper, silver, and molybdenum in order to check for potential global biases in the estimated block grades. Table 17-24 compares mean nearest neighbor (NN) and inverse distance weighted (IDW) grades at a zero cutoff grade for Kerr, Sulphurets, Mitchell, and Iron Cap zones by resource category.

Table 17-24: Grade Model Bias Checks

Kerr Zone						
Metal	Indicated			Inferred		
	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.2470	0.2477	-0.3%	0.1755	0.1698	3.4%
Copper (%)	0.4472	0.4431	0.9%	0.1819	0.1797	1.2%
Silver (g/t)	1.6981	1.6097	5.5%	0.9349	0.8946	4.5%
Molybdenum (ppm)	n/a	n/a	n/a	n/a	n/a	n/a

Sulphurets Zone						
Metal	Indicated			Inferred		
	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.6089	0.6193	-1.7%	0.3768	0.3770	-0.1%
Copper (%)	0.2446	0.2459	-0.5%	0.1098	0.1098	0.0%
Silver (g/t)	0.7675	0.7765	-1.2%	1.2324	1.1371	8.4%
Molybdenum (ppm)	78.6	74.1	6.1%	28.7	27.6	4.1%

Mitchell Zone						
Metal	Measured+Indicated			Inferred		
	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.5848	0.5849	0.0%	0.3322	0.3183	4.4%
Copper (%)	0.1639	0.1631	0.5%	0.1073	0.1006	6.7%
Silver (g/t)	3.1401	3.2587	-3.6%	2.6264	2.6438	-0.7%
Molybdenum (ppm)	60.5	61.1	-0.9%	46.1	47.3	-2.6%

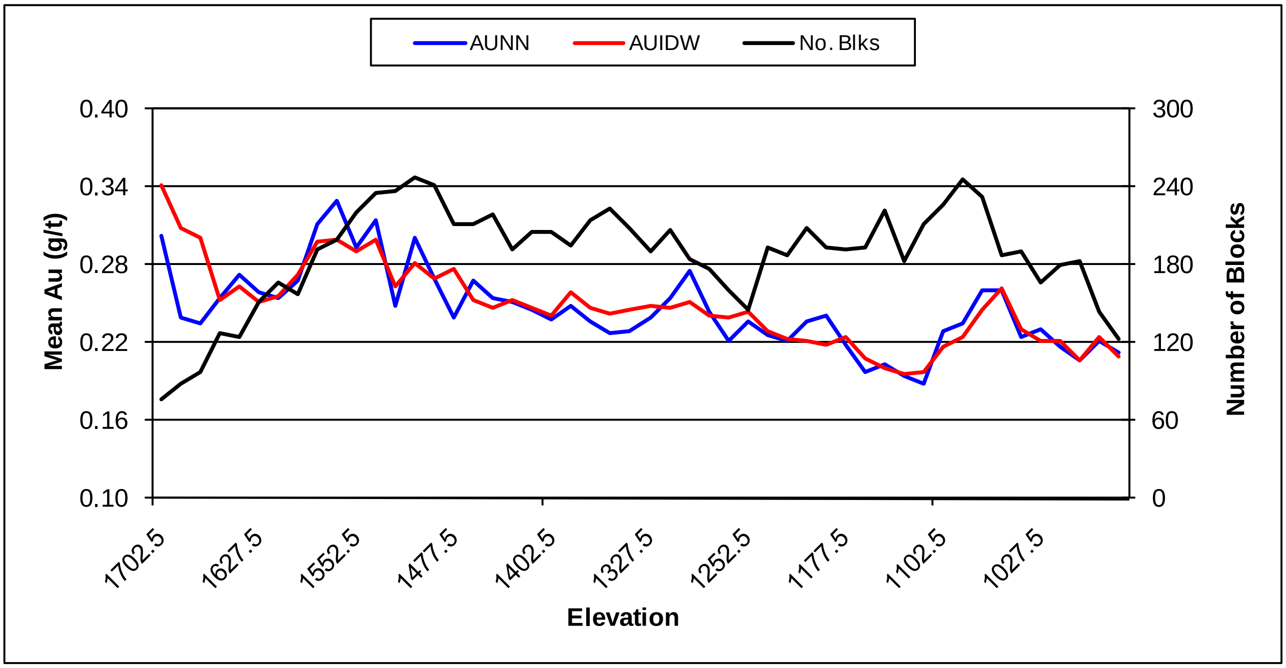
Iron Cap Zone						
Metal	Indicated			Inferred		
	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.4034	0.4031	0.1%	0.3119	0.3201	-2.6%
Copper (%)	0.1871	0.1866	0.3%	0.1652	0.1648	0.2%
Silver (g/t)	4.9637	4.8528	2.3%	3.2468	3.1480	3.1%
Molybdenum (ppm)	43.3	43.0	0.7%	48.7	50.8	-4.1%

The results shown in Table 17-24 show that the inverse distance weighted (IDW) models compare very well with the nearest neighbor grades for the Measured+Indicated category (only the Mitchell zone has Measured Resources). There are wider differences in mean grades for Inferred material which is based on less drilling hence lower confidence levels in those estimates.

Possible local biases in the estimate of block grades were examined by preparing a set of “swath plots” for gold and copper. These plots compare mean estimated inverse distance gold and copper grades (AUIDW and CUIDW) with nearest neighbor gold and copper (AUNN and CUNN) estimates by block model columns (eastings), rows (northings), and levels (elevation). Gold and Copper swath plots by elevation are shown in Figures 17-37 through 17-40 for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. These plots were drawn for Measured (Mitchell only) and Indicated Resources. The number of blocks by elevation are shown by the heavy black line and the units are read from the Y-axis on the right side of the plots.

In the author’s opinion, the swath plots shown in Figures 17-37 through 17-40 show a close comparison between the inverse distance and nearest neighbor estimates. There do not appear to be any severe local biases in the estimate of gold and copper. Based on visual and statistical checks, it is the opinion of the author that the Kerr, Sulphurets, Mitchell, and Iron Cap models are globally unbiased and represent reasonable estimates of insitu block grades.

Figure 17-37: Kerr Au-Cu Swath Plots by Elevation



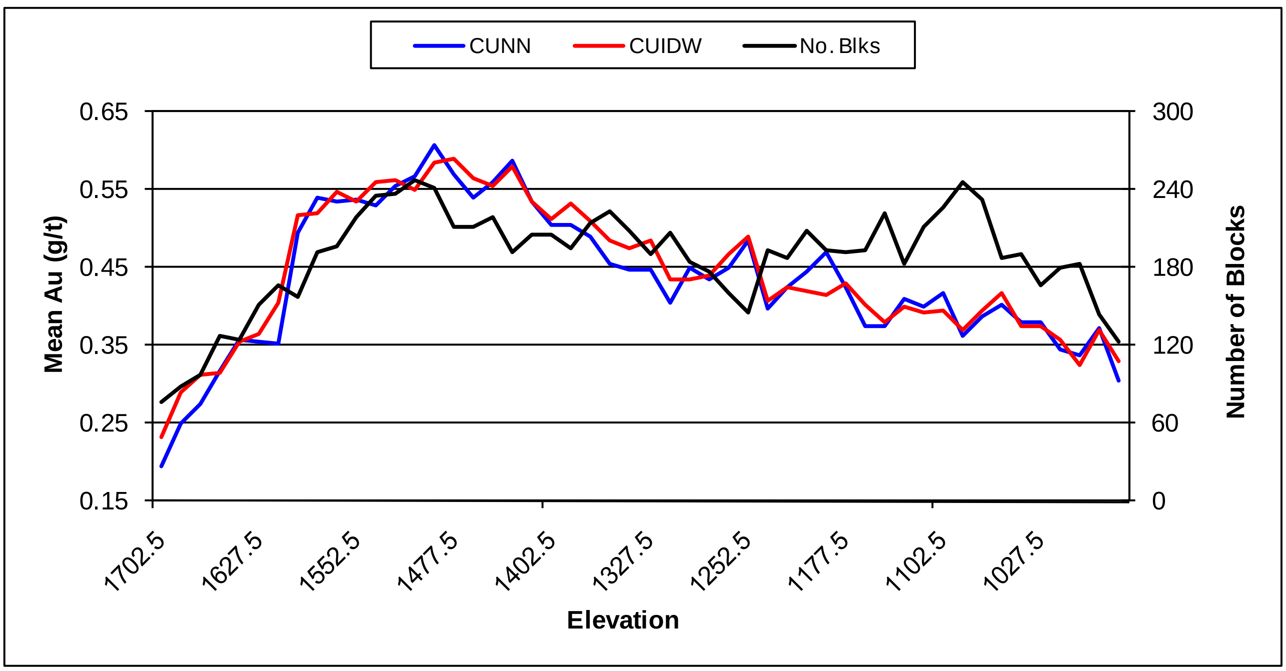
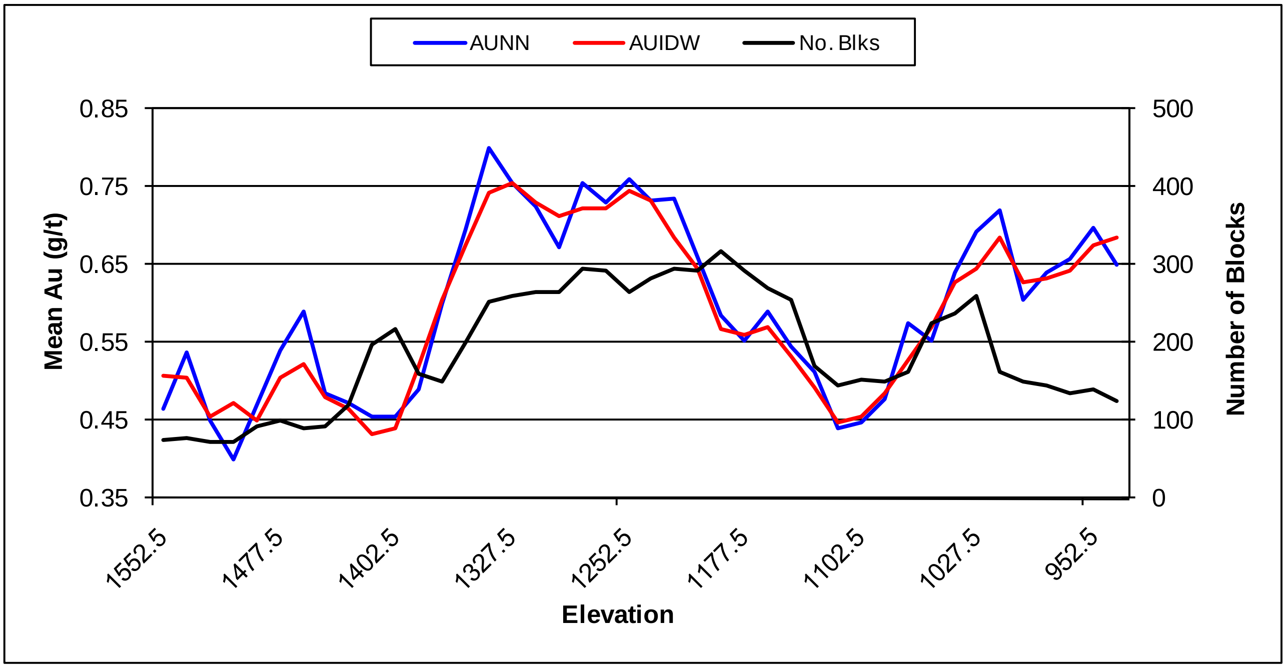


Figure 17-38: Sulphurets Au-Cu Swath Plots by Elevation



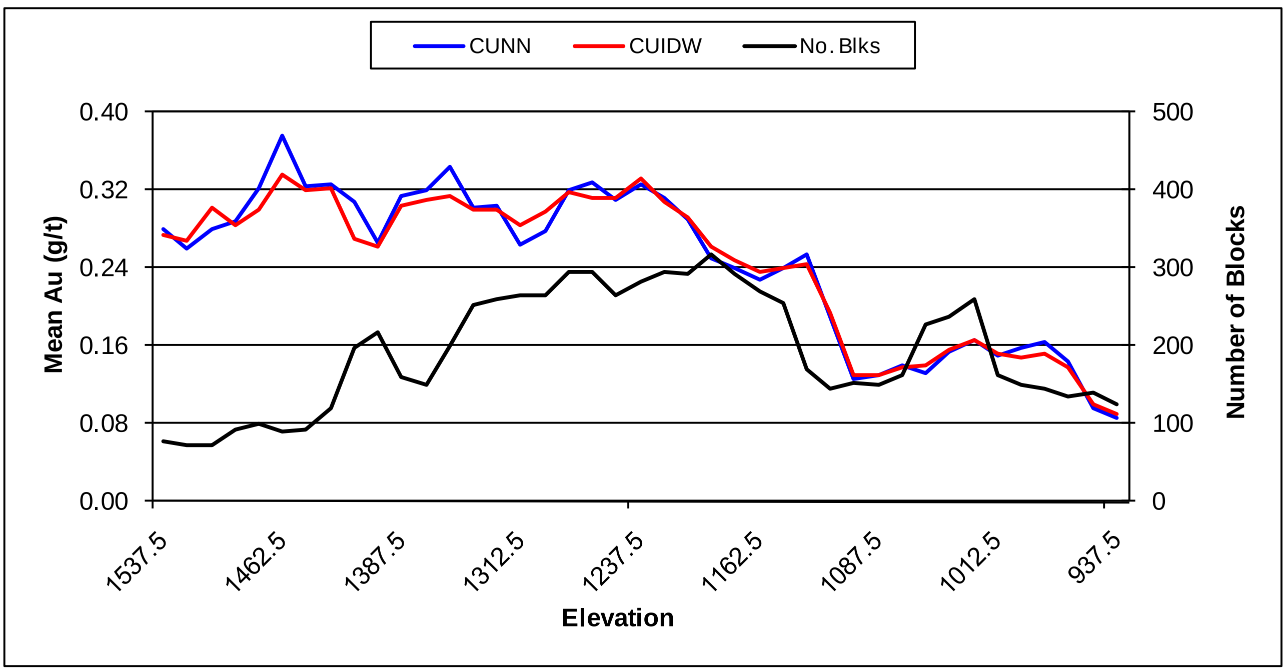
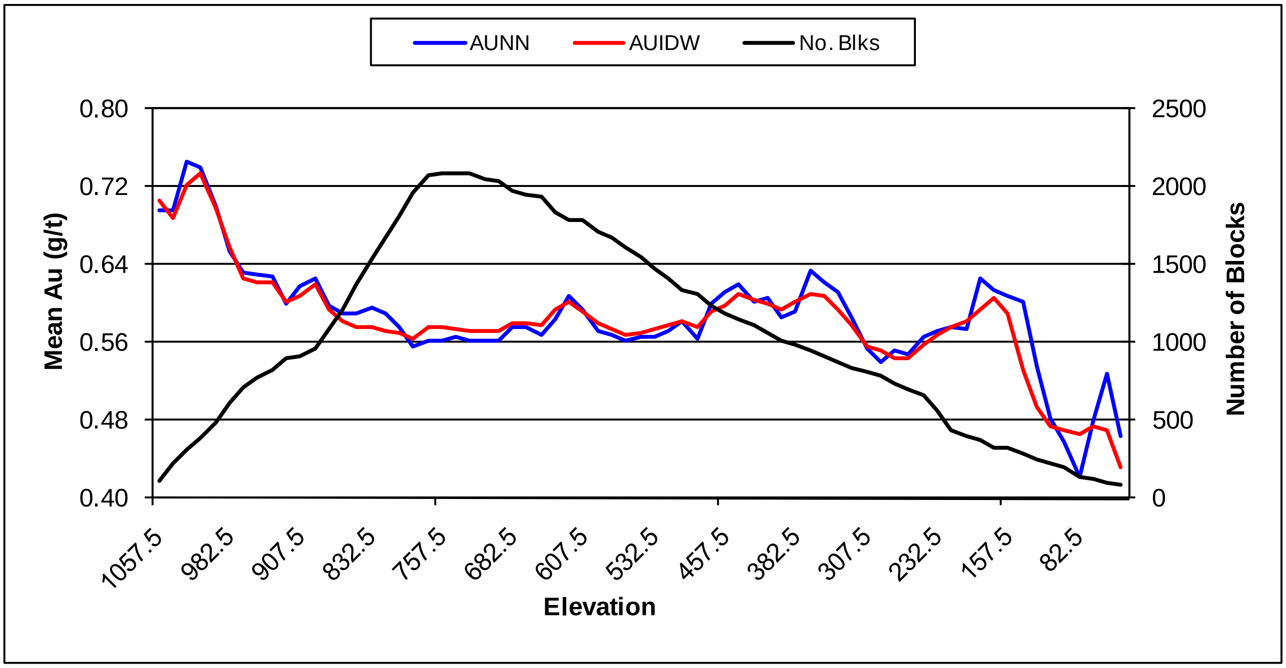


Figure 17-39: Mitchell Au-Cu Swath Plots by Elevation



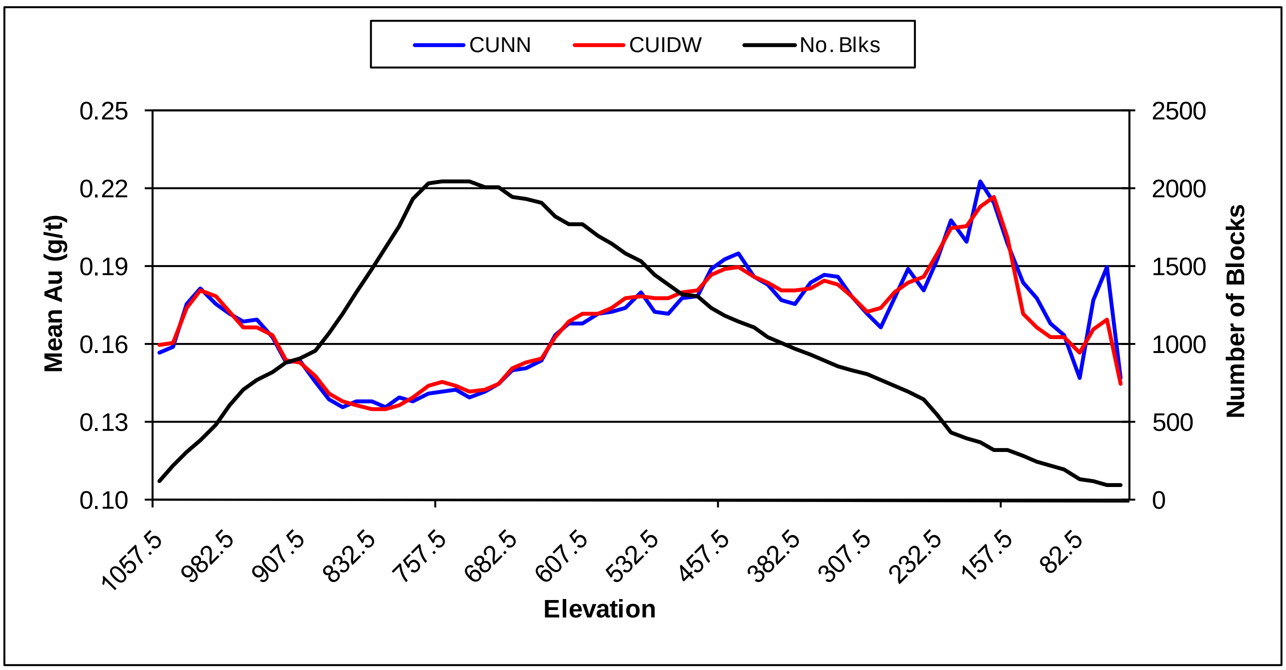
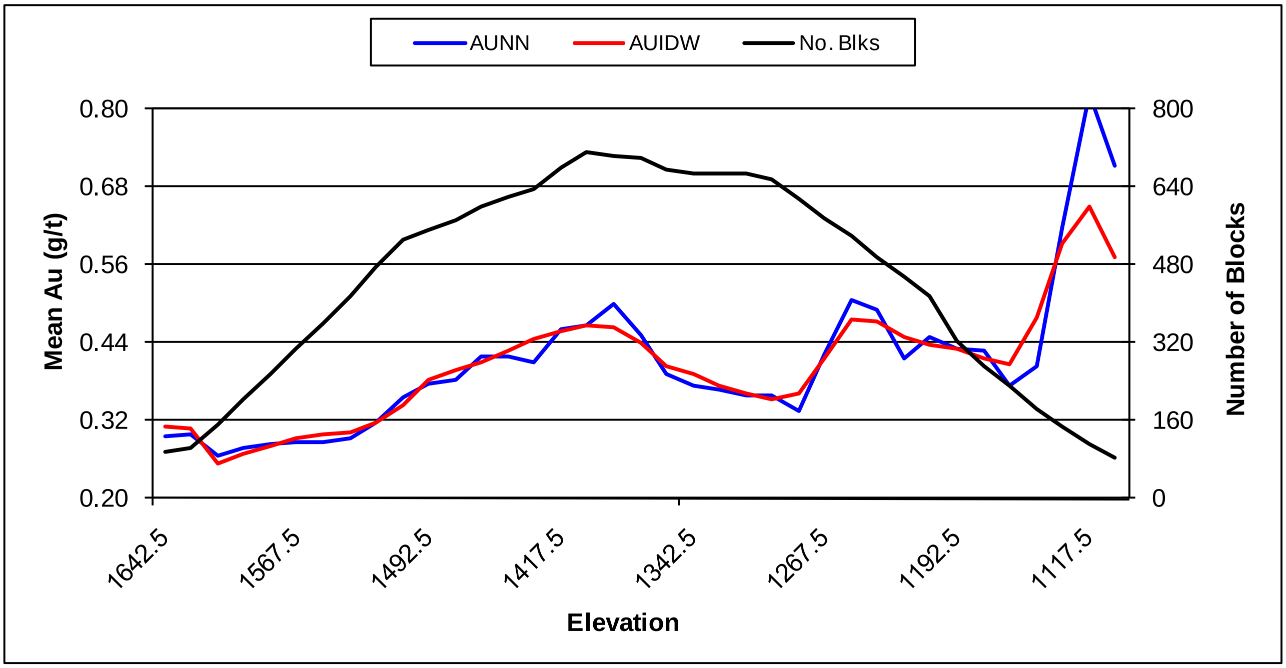
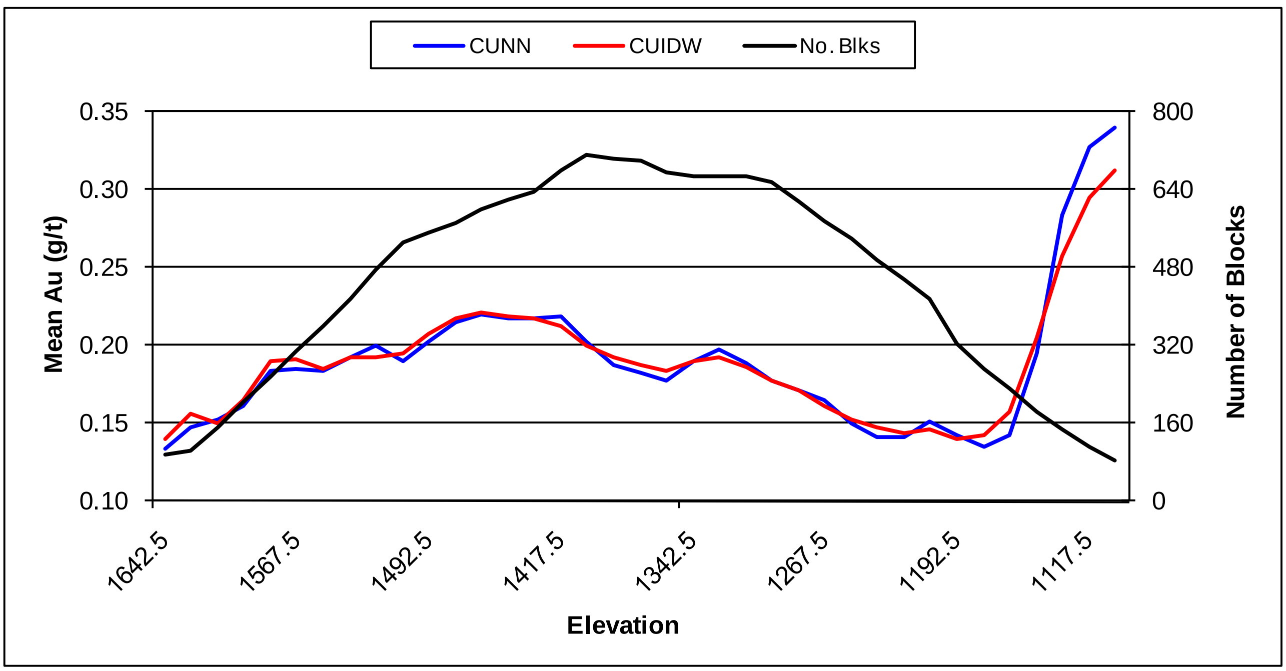


Figure 17-40: Iron Cap Au-Cu Swath Plots by Elevation





17.9 **Resource Classification**

The author classified Kerr, Sulphurets, Mitchell, and Iron Cap estimated block grades into Measured (Mitchell only), Indicated, and Inferred Mineral Resources using a combination of distance to data, a required number of drill holes, and manually constructed shapes that represent "mineralized continuity".

To define mineralized continuity, RMI created probabilistic (indicator) gold equivalent models for each mineralized zone using a 0.5 g/t gold equivalent cutoff. Blocks with an estimated probability in excess of 50% of being above a 0.50 g/t gold equivalent cutoff were used as a guide in drawing mid-bench polygons that defined mineralized continuity. The indicator probability model required that at least three drill holes were used to estimate block probabilities using a 150m spherical search strategy.

Blocks for all four mineralized zones were initially coded with the mineralized continuity polygons and were considered to be Indicated Resources (code = 2). A default code of 5 was assigned to all other blocks. Then criteria such as distance to the closest drill hole and a minimum number of drill holes used to estimate the block grade were tested to see if the block was to remain as an Indicated Resource. If the criteria were not met the Indicated blocks were re-assigned to Inferred (code = 3). Table 17-25 summarizes the criteria that were used to establish Indicated Resources.

Table 17-25: Indicated Resource Criteria

Mineralized Zone	Block Location	Minimun No. Holes	Distance to Closest Composite (m)
Kerr	Inside mineralized continuity shape	≥ 2	≤ 75
Sulphurets	Inside mineralized continuity shape	≥ 2	≤ 75
Mitchell	Inside mineralized continuity shape & below MTF	≥ 2	≤ 125
Iron Cap	Inside mineralized continuity shape	≥ 2	≤ 75

Measured Mineral Resources (code = 1) were only assigned to the Mitchell zone if 1) the blocks were located inside of the mineralized continuity shape and 2) they were estimated by two or more holes with the closest being within 50 meters or one hole within 17 meters of the block.

Inferred Mineral Resources were assigned to any unclassified blocks (i.e. code = 5) if the distance to drilling data and the minimum number of holes used to estimate block grades were met. Table 17-26 summarizes the criteria used to establish Inferred Resources.

Table 17-26: Inferred Resource Criteria

Mineralized Zone	Block Location	Minimun No. Holes	Distance to Closest Composite (m)
Kerr	Outside mineralized continuity shape	≥ 2	≤ 37.5
	Inside/outside mineralized continuity shape	≥ 1	≤ 20
Sulphurets	Above STF	≥ 2	≤ 37.5
	Above STF	≥ 1	≤ 25
	Below STF, inside mineralized continuity shape	≥ 1	≤ 50
	Below STF, outside mineralized continuity shape	≥ 2	≤ 50
	Below STF, outside mineralized continuity shape	≥ 1	≤ 25
Mitchell	Above MTF, inside mineralized continuity shape	≥ 1	≤ 75
	Above MTF, outside mineralized continuity shape	≥ 1	≤ 50
	Below MTF, inside mineralized continuity shape	≥ 2	≤ 175
	Below MTF, outside mineralized continuity shape	≥ 2	≤ 75
	Below MTF, outside mineralized continuity shape	≥ 1	≤ 50
Iron Cap	Inside/outside mineralized continuity shape	≥ 2	≤ 125
	Inside/outside mineralized continuity shape	≥ 1	≤ 75

Note:
STF = Sulphurets Thrust Fault
MTF = Mitchell Thrust Fault

17.10 **Summary of Mineral Resources**

Mineral Resources were tabulated for the Kerr, Sulphurets, Mitchell, and Iron Cap zones using a gold equivalent cutoff grade. This equivalent grade was calculated based on assumed metal prices and recoveries. A gold price of US $650 per ounce and a copper price of US $2.00 per pound were used to calculate the gold equivalent grade. Gold and copper recoveries of 70% and 85%, respectively were also used to calculate gold equivalency using the following expression:

Gold Equivalent Grade (AuEQ) = Au (g/t) + (Cu (%) * (((Cu price/453.5924)/Au price/31.1035)) * (Cu recovery/Au recovery))*10000

The metal prices and recoveries are the same as those used in past KSM gold equivalent calculations and were selected so that direct comparisons could be made with previous estimates. RMI notes that some apparent discrepancies in the calculation of contained metal may occur due to the rounding of tonnes and grades.

Mineral Resources are summarized in Table 17-27 at a gold equivalent cutoff grade of 0.50 g/t, which has been selected for disclosing Mineral Resources. This cutoff grade is above a "break-even" cutoff grade given today's metal prices and was used for direct comparisons with previous KSM resource estimates. Mineral Resources for the Kerr, Sulphurets, Mitchell, and Iron Cap zones are tabulated in Tables 17-28 through 17-31, respectively at a number of gold equivalent cutoff grades.). Note that the KSM resources shown in Tables 17-27 through 17-30 are inclusive of Mineral Reserves that were disclosed in 2010 (Wardrop, 2010). No reserves have ever been declared for Iron Cap.

Table 17-27: Summary of KSM Mineral Resources

Measured Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	677,600	0.64	13,943	0.17	2,539	3.2	69,713	58	86.6
Total	677,600	0.64	13,943	0.17	2,539	3.2	69,713	58	86.6

Indicated Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	1,069,500	0.59	20,287	0.17	4,007	3.2	110,033	60	141.4
Sulphurets	199,300	0.63	4,037	0.26	1,142	0.7	4,485	59	25.9
Kerr	241,200	0.25	1,939	0.47	2,499	1.2	9,306	n/a	n/a
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	1,871,700	0.52	31,380	0.23	9,322	3.1	186,620	57	204.8

Measured Plus Indicated Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	1,747,100	0.61	34,230	0.17	6,546	3.2	179,746	59	228.0
Sulphurets	199,300	0.63	4,037	0.26	1,142	0.7	4,485	59	25.9
Kerr	241,200	0.25	1,939	0.47	2,499	1.2	9,306	n/a	n/a
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	2,549,300	0.55	45,323	0.21	11,861	3.1	256,333	57	291.4

Inferred Resources									
Zone	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
Mitchell	551,000	0.43	7,617	0.14	1,700	3.1	54,917	47	57.1
Sulphurets	160,500	0.53	2,735	0.16	566	1.1	5,676	34	12.0
Kerr	91,500	0.23	677	0.30	605	0.7	2,059	n/a	n/a
Iron Cap	297,300	0.36	3,441	0.20	1,310	3.9	37,278	60	39.3
Total	1,100,300	0.41	14,470	0.17	4,181	2.8	99,930	49	108.4

Note: Mineral Resources which are not Mineral Reserves do no have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as tc economic and legal feasibility. It cannot be assume that all or any part of an Inferred Resource will ever be up to a higher category.

Table 17-28: Kerr Mineral Resources

Kerr Indicated Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	255,600	0.25	2,054	0.45	2,535	1.2	9,861	n/a	n/a
0.30	254,200	0.25	2,043	0.45	2,521	1.2	9,807	n/a	n/a
0.35	252,700	0.25	2,031	0.45	2,506	1.2	9,749	n/a	n/a
0.40	249,900	0.25	2,009	0.46	2,534	1.2	9,641	n/a	n/a
0.45	246,200	0.25	1,979	0.46	2,496	1.2	9,499	n/a	n/a
0.50	**241,200**	**0.25**	**1,939**	**0.47**	**2,499**	**1.2**	**9,306**	**n/a**	**n/a**
0.55	235,800	0.26	1,971	0.48	2,495	1.2	9,097	n/a	n/a
0.60	230,100	0.26	1,923	0.49	2,485	1.2	8,877	n/a	n/a
0.65	222,900	0.26	1,863	0.50	2,456	1.2	8,600	n/a	n/a
0.70	215,900	0.26	1,805	0.51	2,427	1.2	8,330	n/a	n/a
0.75	208,100	0.26	1,740	0.52	2,385	1.2	8,029	n/a	n/a

Kerr Inferred Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	150,700	0.20	969	0.22	731	0.7	3,392	n/a	n/a
0.30	135,200	0.21	913	0.23	685	0.7	3,043	n/a	n/a
0.35	122,300	0.21	826	0.25	674	0.7	2,752	n/a	n/a
0.40	110,900	0.22	784	0.27	660	0.7	2,496	n/a	n/a
0.45	100,300	0.23	742	0.28	619	0.7	2,257	n/a	n/a
0.50	**91,500**	**0.23**	**677**	**0.30**	**605**	**0.7**	**2,059**	**n/a**	**n/a**
0.55	83,900	0.24	647	0.31	573	0.7	1,888	n/a	n/a
0.60	76,400	0.25	614	0.33	556	0.7	1,719	n/a	n/a
0.65	68,600	0.25	551	0.35	529	0.8	1,764	n/a	n/a
0.70	62,000	0.26	518	0.36	492	0.8	1,595	n/a	n/a
0.75	55,500	0.27	482	0.38	465	0.8	1,427	n/a	n/a

Note: Mineral Resources which are not Mineral Reserves do no have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to economic and legal feasibility. It cannot be assume that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 17-29: Sulphurets Mineral Resources

Sulphurets Indicated Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	212,900	0.61	4,175	0.25	1,173	0.7	4,791	57	26.7
0.30	212,300	0.61	4,164	0.25	1,170	0.7	4,778	57	26.7
0.35	211,200	0.61	4,142	0.25	1,164	0.7	4,753	57	26.5
0.40	209,300	0.62	4,172	0.25	1,153	0.7	4,710	57	26.3
0.45	204,300	0.62	4,072	0.25	1,126	0.7	4,598	58	26.1
0.50	**199,300**	**0.63**	**4,037**	**0.26**	**1,142**	**0.7**	**4,485**	**59**	**25.9**
0.55	192,600	0.64	3,963	0.26	1,104	0.7	4,335	61	25.9
0.60	184,800	0.65	3,862	0.27	1,100	0.7	4,159	62	25.3
0.65	177,700	0.66	3,771	0.28	1,097	0.7	3,999	64	25.1
0.70	170,600	0.67	3,675	0.29	1,090	0.7	3,839	66	24.8
0.75	161,500	0.68	3,531	0.30	1,068	0.7	3,635	68	24.2

Sulphurets Inferred Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	260,000	0.41	3,427	0.12	688	1.0	8,359	26	14.9
0.30	242,300	0.43	3,350	0.13	694	1.0	7,790	27	14.4
0.35	218,700	0.46	3,234	0.13	627	1.1	7,734	29	14.0
0.40	200,000	0.48	3,086	0.14	617	1.1	7,073	31	13.7
0.45	180,700	0.51	2,963	0.15	597	1.1	6,391	32	12.7
0.50	**160,500**	**0.53**	**2,735**	**0.16**	**566**	**1.1**	**5,676**	**34**	**12.0**
0.55	140,600	0.56	2,531	0.17	527	1.1	4,972	37	11.5
0.60	126,800	0.59	2,405	0.18	503	1.1	4,484	39	10.9
0.65	112,500	0.61	2,206	0.19	471	1.1	3,979	41	10.2
0.70	101,400	0.63	2,054	0.20	447	1.1	3,586	44	9.8
0.75	89,700	0.65	1,875	0.21	415	1.1	3,172	47	9.3

Note: Mineral Resources which are not Mineral Reserves do no have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to economic and legal feasibility. It cannot be assume that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 17-30: Mitchell Mineral Resources

Mitchell Measured Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	725,900	0.61	14,236	0.16	2,560	3.1	72,348	60	96.0
0.30	724,100	0.62	14,434	0.16	2,553	3.1	72,169	60	95.8
0.35	719,600	0.62	14,344	0.17	2,696	3.1	71,721	60	95.2
0.40	710,200	0.62	14,157	0.17	2,661	3.1	70,784	60	93.9
0.45	695,200	0.63	14,081	0.17	2,605	3.1	69,289	59	90.4
0.50	**677,600**	**0.64**	**13,943**	**0.17**	**2,539**	**3.2**	**69,713**	**58**	**86.6**
0.55	656,300	0.65	13,715	0.17	2,459	3.2	67,522	57	82.4
0.60	623,800	0.67	13,437	0.18	2,475	3.3	66,184	56	77.0
0.65	590,000	0.68	12,899	0.18	2,341	3.3	62,597	54	70.2
0.70	558,300	0.70	12,565	0.19	2,338	3.4	61,029	53	65.2
0.75	525,300	0.71	11,991	0.19	2,200	3.4	57,422	51	59.0

Mitchell Indicated Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	1,141,100	0.57	20,912	0.16	4,024	3.1	113,730	61	153.4
0.30	1,137,600	0.57	20,848	0.16	4,012	3.1	113,381	61	152.9
0.35	1,127,900	0.57	20,670	0.16	3,977	3.1	112,415	61	151.6
0.40	1,115,400	0.57	20,441	0.16	3,933	3.2	114,755	60	147.5
0.45	1,095,800	0.58	20,434	0.16	3,864	3.2	112,738	60	144.9
0.50	**1,069,500**	**0.59**	**20,287**	**0.17**	**4,007**	**3.2**	**110,033**	**60**	**141.4**
0.55	1,036,100	0.59	19,654	0.17	3,882	3.2	106,596	59	134.7
0.60	987,100	0.61	19,359	0.17	3,698	3.3	104,729	57	124.0
0.65	927,000	0.62	18,478	0.18	3,678	3.4	101,333	56	114.4
0.70	866,600	0.64	17,832	0.18	3,438	3.4	94,730	54	103.1
0.75	804,100	0.65	16,804	0.19	3,367	3.5	90,483	52	92.2

Mitchell Inferred Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	974,600	0.34	10,654	0.10	2,148	2.7	84,602	49	105.3
0.30	893,500	0.36	10,342	0.11	2,166	2.8	80,435	49	96.5
0.35	810,400	0.38	9,901	0.11	1,965	2.8	72,954	48	85.7
0.40	728,400	0.39	9,133	0.12	1,926	2.9	67,914	47	75.5
0.45	632,500	0.41	8,337	0.13	1,812	3.0	61,006	48	66.9
0.50	**551,000**	**0.43**	**7,617**	**0.14**	**1,700**	**3.1**	**54,917**	**47**	**57.1**
0.55	489,600	0.45	7,083	0.14	1,511	3.2	50,371	47	50.7
0.60	431,200	0.47	6,516	0.15	1,426	3.3	45,749	45	42.8
0.65	374,500	0.48	5,779	0.16	1,321	3.4	40,938	43	35.5
0.70	329,200	0.50	5,292	0.16	1,161	3.5	37,044	42	30.5
0.75	279,800	0.52	4,678	0.17	1,048	3.5	31,485	41	25.3

Note: Mineral Resources which are not Mineral Reserves do no have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to economic and legal feasibility. It cannot be assume that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 17-31: Iron Cap Mineral Resources

Iron Cap Indicated Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	419,200	0.41	5,526	0.19	1,755	5.0	67,388	44	40.7
0.30	413,000	0.41	5,444	0.19	1,729	5.1	67,719	44	40.1
0.35	403,700	0.42	5,451	0.19	1,691	5.1	66,194	45	40
0.40	391,300	0.42	5,284	0.20	1,725	5.2	65,419	45	38.8
0.45	376,800	0.43	5,209	0.20	1,661	5.4	65,418	46	38.2
0.50	**361,700**	**0.44**	**5,117**	**0.21**	**1,674**	**5.4**	**62,796**	**47**	**37.5**
0.55	344,800	0.45	4,989	0.21	1,596	5.5	60,971	46	35
0.60	325,400	0.46	4,812	0.22	1,578	5.6	58,586	45	32.3
0.65	304,200	0.48	4,695	0.22	1,475	5.7	55,747	43	28.8
0.70	279,800	0.49	4,408	0.23	1,418	5.8	52,175	41	25.3
0.75	250,300	0.52	4,185	0.23	1,269	5.9	47,479	38	21

Iron Cap Inferred Resources									
AuEQ Cutoff (g/t)	Tonnes (000)	Gold (g/t)	Gold (000 of ounces)	Cu (%)	Copper (millions of lbs)	Silver (g/t)	Silver (000 of ounces)	Moly (ppm)	Moly (millions of lbs)
0.25	373,700	0.33	3,965	0.17	1,400	3.4	40,850	51	42
0.30	365,900	0.33	3,882	0.18	1,452	3.5	41,174	52	41.9
0.35	353,300	0.34	3,862	0.18	1,402	3.6	40,892	53	41.3
0.40	339,100	0.34	3,707	0.18	1,345	3.6	39,248	55	41.1
0.45	318,800	0.35	3,587	0.19	1,335	3.8	38,949	57	40.1
0.50	**297,300**	**0.36**	**3,441**	**0.20**	**1,310**	**3.9**	**37,278**	**60**	**39.3**
0.55	273,800	0.37	3,257	0.20	1,207	3.9	34,331	63	38
0.60	244,800	0.39	3,069	0.21	1,133	4.0	31,482	64	34.5
0.65	224,400	0.41	2,958	0.22	1,088	4.0	28,858	62	30.7
0.70	195,300	0.43	2,700	0.22	947	4.0	25,116	60	25.8
0.75	169,600	0.46	2,508	0.23	860	4.0	21,811	60	22.4

Note: Mineral Resources which are not Mineral Reserves do no have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to economic and legal feasibility. It cannot be assume that all or any part of an Inferred Resource will ever be upgraded to a higher category.

17.11 **Conceptual Pit Results**

The Mineral Resources summarized in Table 17-27 were tabulated as “global resources” using a gold equivalent cutoff grade that is higher than a conceptual "break-even" cutoff grade using current metal prices. As a test to determine “reasonable expectation of economic viability”, the author generated a number of conceptual pits for each mineralized zone using the floating cone algorithm. Measured, Indicated, and

Inferred Mineral Resources were used in all cases. Five different metal prices and two different constant pit slope angles were used to generate a total of 10 conceptual pits. Mining and processing costs were kept constant for all 10 cases. Table 17-32 summarizes the key parameters that were used to generate the conceptual pits.

Table 17-32: Conceptual Pit Parameters

Conceptual Pit Number	Au Price (US $/oz)	Cu Price (US $/lb)	Ag Price (US $/oz)	Mo Price (US $/lb)	Au Rec. (%)	Cu Rec. (%)	Ag Rec. (%)	Mo Rec. (%)	Mining Cost (US $/tonne)	Processing Cost (US $/tonne)	Slope Angle (deg)
1	$700	$2.50	$10.00	$15.00	72%	84%	69%	32%	$1.60	$6.96	45
2	$850	$2.75	$12.50	$17.50	72%	84%	69%	32%	$1.60	$6.96	45
3	$1,000	$3.00	$15.00	$20.00	72%	84%	69%	32%	$1.60	$6.96	45
4	$1,150	$3.25	$17.50	$22.50	72%	84%	69%	32%	$1.60	$6.96	45
5	$1,300	$3.50	$20.00	$25.00	72%	84%	69%	32%	$1.60	$6.96	45
6	$700	$2.50	$10.00	$15.00	72%	84%	69%	32%	$1.60	$6.96	40
7	$850	$2.75	$12.50	$17.50	72%	84%	69%	32%	$1.60	$6.96	40
8	$1,000	$3.00	$15.00	$20.00	72%	84%	69%	32%	$1.60	$6.96	40
9	$1,150	$3.25	$17.50	$22.50	72%	84%	69%	32%	$1.60	$6.96	40
10	$1,300	$3.50	$20.00	$25.00	72%	84%	69%	32%	$1.60	$6.96	40

Mineral Resources for all 10 conceptual pits are tabulated for the Kerr, Sulphurets, Mitchell, and Iron Cap zones using a 0.50 g/t Au equivalent cutoff grade in Tables 17-33 through 17-36, respectively. For reference, the officially stated Mineral Resources for each zone are shown at the bottom of the tables and are highlighted in yellow. Rounding of tonnes and grade may result in contained metal in the conceptual pits to be greater than the global resource.

Table 17-33: Kerr Conceptual Pit Results

Conceptual Pit No.	Indicated Resources					Inferred Resources				
	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)
1	232,260	0.26	1,942	0.48	2,457	64,711	0.25	520	0.29	414
2	235,512	0.26	1,969	0.47	2,440	68,727	0.25	552	0.29	439
3	236,970	0.26	1,981	0.47	2,455	70,541	0.25	567	0.29	451
4	237,847	0.25	1,912	0.47	2,464	73,200	0.24	565	0.29	468
5	238,639	0.25	1,918	0.47	2,472	75,250	0.24	581	0.29	481
6	218,375	0.26	1,825	0.48	2,310	57,391	0.26	480	0.28	354
7	223,507	0.26	1,868	0.48	2,365	61,047	0.25	491	0.28	377
8	228,781	0.26	1,912	0.47	2,370	64,296	0.25	517	0.28	397
9	232,212	0.26	1,941	0.47	2,405	67,156	0.25	540	0.28	414
10	234,548	0.26	1,961	0.47	2,430	70,615	0.24	545	0.28	436
Official Resource	**241,200**	**0.25**	**1,939**	**0.47**	**2,499**	**91,500**	**0.23**	**677**	**0.30**	**605**

Table 17-34: Sulphurets Conceptual Pit Results

Conceptual Pit No.	Indicated Resources					Inferred Resources				
	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)
1	195,589	0.64	4,025	0.26	1,121	112,531	0.55	1,990	0.15	372
2	197,693	0.63	4,004	0.26	1,133	125,121	0.54	2,172	0.15	414
3	198,602	0.63	4,023	0.26	1,138	131,249	0.54	2,279	0.15	434
4	198,939	0.63	4,030	0.26	1,140	134,521	0.54	2,335	0.15	445
5	199,017	0.63	4,031	0.26	1,140	137,092	0.54	2,380	0.15	453
6	188,136	0.64	3,871	0.26	1,078	104,060	0.55	1,840	0.15	344
7	196,914	0.63	3,988	0.26	1,128	119,511	0.54	2,075	0.15	395
8	198,004	0.63	4,011	0.26	1,135	126,000	0.54	2,188	0.15	417
9	198,602	0.63	4,023	0.26	1,138	131,168	0.54	2,277	0.15	434
10	198,783	0.63	4,026	0.26	1,139	134,283	0.54	2,331	0.15	444
Official Resource	**199,300**	**0.63**	**4,037**	**0.26**	**1,142**	**160,500**	**0.53**	**2,735**	**0.16**	**566**

Table 17-35: Mitchell Conceptual Pit Results

Conceptual Pit No.	Measured + Indicated Resources					Inferred Resources				
	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)
1	1,547,361	0.62	30,854	0.17	5,797	206,302	0.35	2,321	0.14	637
2	1,664,972	0.61	32,846	0.17	6,238	285,440	0.39	3,579	0.14	881
3	1,707,138	0.61	33,454	0.17	6,396	346,630	0.41	4,569	0.14	1,070
4	1,718,826	0.61	33,680	0.17	6,440	370,073	0.42	4,997	0.14	1,142
5	1,729,390	0.61	33,886	0.17	6,480	396,124	0.42	5,349	0.14	1,222
6	1,254,208	0.64	25,885	0.17	4,699	142,952	0.31	1,425	0.14	441
7	1,470,582	0.62	29,544	0.17	5,510	200,140	0.35	2,252	0.14	618
8	1,625,208	0.61	32,077	0.17	6,090	273,298	0.38	3,339	0.14	843
9	1,655,486	0.61	32,663	0.17	6,203	298,195	0.39	3,739	0.14	920
10	1,685,744	0.61	33,039	0.17	6,316	335,866	0.41	4,427	0.13	962
Official Resource	**1,747,100**	**0.61**	**34,230**	**0.17**	**6,546**	**551,000**	**0.43**	**7,617**	**0.14**	**1,700**

Table 17-36: Iron Cap Conceptual Pit Results

Conceptual Pit No.	Indicated Resources					Inferred Resources				
	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)	Tonnes (000)	Au (g/t)	Au Ozs (000)	Cu (%)	Cu Lbs (M)
1	358,260	0.44	5,068	0.21	1,658	225,157	0.35	2,534	0.2	992
2	360,931	0.44	5,106	0.21	1,671	247,813	0.35	2,789	0.19	1,038
3	361,625	0.44	5,116	0.21	1,674	258,962	0.36	2,997	0.19	1,084
4	361,651	0.44	5,116	0.21	1,674	267,978	0.36	3,102	0.19	1,122
5	361,676	0.44	5,116	0.21	1,674	272,602	0.36	3,155	0.19	1,142
6	352,634	0.44	4,988	0.21	1,632	215,678	0.34	2,358	0.2	951
7	360,084	0.44	5,094	0.21	1,667	245,476	0.35	2,762	0.2	1,082
8	361,163	0.44	5,109	0.21	1,672	254,132	0.35	2,860	0.19	1,064
9	361,548	0.44	5,115	0.21	1,673	264,484	0.35	2,976	0.19	1,108
10	361,651	0.44	5,116	0.21	1,674	268,312	0.35	3,019	0.2	1,183
Official Resource	**361,700**	**0.44**	**5,117**	**0.21**	**1,674**	**297,300**	**0.36**	**3,441**	**0.20**	**1,310**

RMI notes that conceptual pits based upon a gold price of at least US $1,000 per ounce and a copper price of US $3.00 per pound capture nearly all of the Indicated Mineral Resources and where applicable, Measured Resources, for each zone. Conceptual pits using those same metal prices captured 98% to 100% of the contained Measured+Indicated gold and copper metal relative to the global resource inventory. Those conceptual pits captured 85%, 85%, 66%, and 90% of the global Inferred Resources for the Kerr, Sulphurets, Mitchell, and Iron Cap, respectively. Similar percentages of contained Inferred copper metal were captured by the conceptual pits. RMI notes that Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred resources have a high degree of uncertainty as to their existence, and great uncertainty as to economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Ongoing work will refine mining and processing costs for future conceptual pits that will be developed for an updated Prefeasibility Study, which is due to be completed during the second quarter of 2011.

17.12 Risks and Uncertainties

- **Resource** - In RMI's opinion, there is little risk associated with the insitu Mineral Resources which are the subject of this report. These estimated resources are based on drilling data that have been verified by RMI and are supported by adequate QA/QC results. Diamond drilling has shown that mineralization tends to be fairly continuous and widespread, especially within the Mitchell zone. Gold and copper variograms suggest long ranges of mineralized continuity along preferential orientations. The estimated block grades have been demonstrated to be globally unbiased and provide a reasonable estimate of local grades. Back testing previous block models with newly obtained infill drilling results have been favorable. The resources which are the subject of this report were not confined to a conceptual pit. RMI used the same cutoff grade that has been used for past resource estimates for comparison purposes. That cutoff grade of 0.50 g/t gold equivalent is higher than a cutoff grade calculated using current prices or the average price over the past several years. RMI did generate a number of conceptual pits for each mineralized zone and compared resources captured by those pits versus the global inventory using the same cutoff grade. The "base case" conceptual pits captured nearly all the Measured (Mitchell only) and Indicated Mineral Resources for all four zones. The conceptual pits captured less Inferred material than the global inventory, especially for the Mitchell zone. Inferred material by its very nature is speculative and may never be upgraded into higher categories.

- **Mining** - Interim and final pit slope angles for each zone are currently being analyzed by several consulting groups. The south and north ultimate high walls of the Mitchell pit present a significant risk due to their overall heights, which are in excess of 1,500 meters. According to Moose Mountain Technical Services, the geotechnical design has been completed to a higher level of detail than is typical for a prefeasibility level study. However, walls of this height have not been built to date. Moose Mountain Technical Services also point out that for the first seven years of mill feed the Mitchell high wall height is less than 1,000m high, for which there is precedence. The current plan also shows that the high wall will be around 1,200m high for the first 16 years of mill feed. Another potential mining issue surrounds glacial ice. Currently a small portion of the Mitchell Glacier is located inside of the "ultimate" pit. However, the glacier has been retreating at a rate of approximately 30m/year. At that rate of melt back coupled with mine scheduling it is likely that no ice will need to be mined. Glacial melt water will need to be diverted from the pit and various diversion plans are being analyzed. The ice field above the Iron Cap zone is more problematic as more ice would have to be contended with than at Mitchell. Various mine planning scenarios are currently being studied. One scenario would call for Iron Cap to be the last KSM zone that is mined. That plan would call for mining ice above the deposit and placing it in the mined out Mitchell pit. Other solutions are being analyzed to deal with this issue including the potential of mining the zone using block caving methods.

- **Processing** - Metal recoveries for Sulphurets and Mitchell appear to be higher than those for the Kerr and Iron Cap zones. If ongoing test work results in showing lower recoveries a portions of the KSM resource could be reduced. Contracts for accepting concentrates from KSM will need to be secured.

- **Permitting** - At this juncture the authors are not aware of any fatal flaws associated with obtaining the various permits needed to construct and operate a mine at this site. However, permitting of any large undeveloped project represents an ongoing risk. RMI has held discussions with Mr. Clem Pelletier, CEO of Rescan Environmental Services Ltd. Mr. Pelletier has indicated that at this stage of the project there is no indication that the project cannot be permitted or that Seabridge will not gain its social license to operate, including the cooperation of the local Aboriginal peoples. However, Mr. Pelletier pointed out that a significant amount of work will need to be completed in order to obtain all required permits.

- **Capital Costs** - In the last KSM Prefeasibility Study (Wardrop, 2010), the capital cost estimate to develop this project was 3.3 billion dollars. This is a significant cost. Based on current estimates and economic studies, the project is not particularly sensitive to capital costs. However, if capital costs were to dramatically increase, the economic return of the project could be adversely affected.

- **Metal Prices** - Metal prices have been at record highs over the past few years, particularly gold. However, if prices were to dramatically fall, the overall project economics could be seriously impaired.

18.0 OTHER RELEVANT DATA AND INFORMATION

Seabridge has retained Rescan, a leading environmental services company located in Vancouver, B.C., to collect baseline data and prepare all of the documents necessary for obtaining various permits.

The following information was excerpted from the April 2010 Preliminary Feasibility Study (Wardrop, 2010) and updated by Seabridge:

The KSM Project is located in the mountainous terrain of northwestern BC, approximately 940 km northwest of Vancouver and approximately 65 km northwest of Stewart, as shown in Figure 18-1. The proposed project area lies approximately 20 km southeast of Barrick Gold's Eskay Creek Mine and within 30 km of the BC-Alaska border. At the present time, access to the property is via helicopter.

The area is rugged, remote, and undeveloped. The widely varying terrain hosts a broad range of ecosystems. Its rivers are home to all five species of Pacific salmon as well as trout and Dolly Varden char. Black and grizzly bears frequent the forests and moose and migratory birds can be found in the wetlands. Mountain goats are common in the alpine areas.

Extensive baseline studies, required to undertake an evaluation of the potential environmental impacts associated with the proposed KSM Project, were initiated in April 2008 following issuance of the Section 10 order from the BC Environmental Assessment Office (BCEAO). Three field seasons of baseline data collection have been completed as of December 2010 and specific programs will continue thru 2011.

Licensing and Permitting

Mining projects in British Columbia are subject to regulation under federal and provincial legislation to protect workers and the environment. This section discusses the principal licences and permits required for the KSM Project. The KSM Project is undergoing a joint harmonized environmental assessment with the Province of British Columbia and the Government of Canada. Figure 18-2 outlines the approval schedule for the Project up to the issuance of high level federal and provincial approvals in principle.

British Columbia Environmental Assessment Act Process

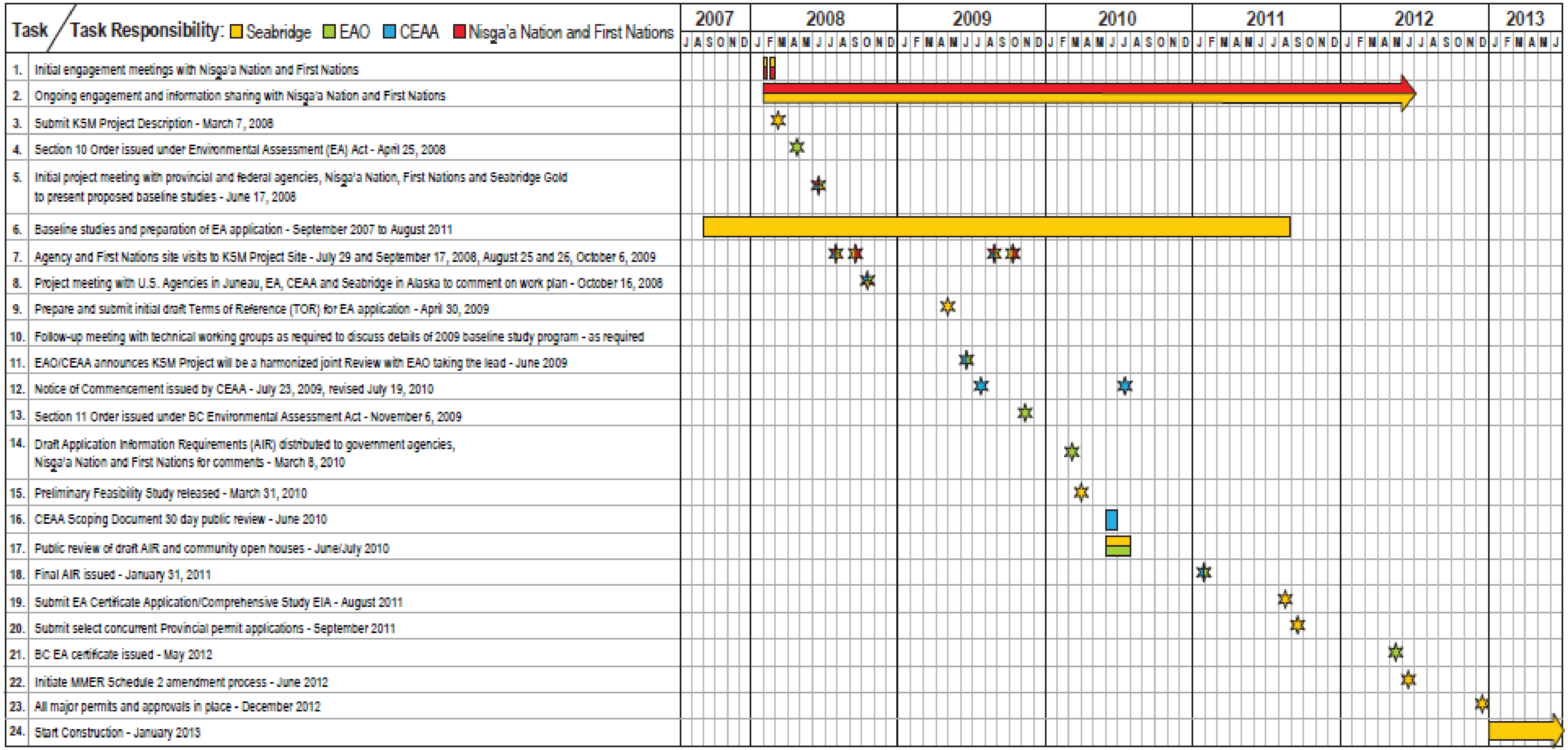
The British Columbia Environmental Assessment Act (BCEAA) requires that certain large-scale project proposals undergo an environmental assessment and obtain an Environmental Assessment Certificate before they can proceed. Proposed mining developments that exceed the threshold criteria laid out in the Reviewable Project Regulations are required under the Act to obtain an Environmental Assessment Certificate from the Ministers of Environmental and Energy, Mines and Petroleum Resources before the issuance of any permits to construct or operate. The KSM Project will be developed *with a production rate greater than the Reviewable Project Regulation threshold of 75,000 t/a and therefore requires an Environmental Assessment Certificate.*

The BC Environmental Assessment Office issued the Section 11 Order for the KSM Project in November 2009, which outlined the procedural issues associated with the planned environmental assessment for the KSM Project. This order also identified the Aboriginal groups with which Seabridge must engage, as the Project may potentially impact their rights and interests in the area.

The Application Information Requirements, or the Terms of Reference, which delineates the requirements for the environmental assessment application document, was approved by the BC Government in January 2011.

Figure 18-1: Project Location Map

Figure 18-2: KSM Project Schedule



Canadian Environmental Assessment Act Process

The proposed KSM Project will require federal approvals, such as for the tailing facility in a wetland environment occupied by fish, or other activities that will require authorizations under Section 35(2) of the Fisheries Act to alter or disturb fish habitat. These approvals will trigger the Canadian Environmental Assessment Act (CEAA) environmental review process.

According to the Comprehensive Study List Regulations, the environmental assessment of the KSM Project must proceed by way of a comprehensive study because, among other reasons, it involves the proposed construction of "a metal mill with an ore capacity of 4,000 tonnes/day or more" and "a gold mine with an ore production capacity of 600 tonnes/day." The federal government confirmed in July 2009 that the KSMP Project will undergo a comprehensive review.

Authorizations Required

The federal and provincial licences, permits, and approvals required to construct, operate, decommission and close the KSM Project are summarized in the following sections. The lists of government approvals represent the major permits, licences, approvals, consents and material authorizations which are required to occupy, use, construct and operate the KSM Project. The list cannot be considered comprehensive due to the complexity of government regulatory processes which evolve over time and the large number of minor permits, licences, approvals, consents and authorizations and potential amendments which will be required throughout the life of the mine.

British Columbia Authorizations, Licenses and Permits

Provincial permitting, licensing and approval processes (statutory permit processes) may proceed concurrently with the BCEAA review or may, at the proponent's option, follow the issuance of the Environmental Assessment Certificate. At this time, Seabridge is evaluating the applicability of seeking concurrent approvals under the BCEAA process for the KSM Project.. However, no statutory permit approvals may be issued before an Environmental Assessment Certificate is obtained. Statutory permit approval processes are normally more specific than the environmental assessment level of review, and for example, will require detailed and possibly final engineering design information for certain permits such as the tailing impoundment structures and others.

Table 18-1 presents a list of provincial authorizations, licences, and permits required to develop the KSM Project. The list includes the major permits and is not intended to be comprehensive.

Table 18-1:
List of BC Authorizations, Licences, and Permits Required to Develop the KSM Project

BC Government Permits and Licenses	Enabling Legislation
Environmental Assessment Certificate	*BC Environmental Assessment Act*
Permit Approving Work System & Reclamation Program (Minesite – Initial Development)	*Mines Act*
Amendment to Permit Approving Work System & Reclamation Program (Pre-production)	*Mines Act*
Amendment to Permit Approving Work System & Reclamation Program (Bonding)	*Mines Act*
Amendment to Permit Approving Work System & Reclamation Program (Mine Plan - Production)	*Mines Act*
Approvals to Construct & Operate Tailings Impoundment Dam	*Mines Act*
Permit Approving Work System & Reclamation Program (Gravel Pit/Wash Plant/Rock Borrow Pit)	*Mines Act*
Water Licence – Notice of Intention (Application)	*Water Act*
Water Licence – Storage & Diversion	*Water Act*
Water Licence – Use	*Water Act*
Licence to Cut – Minesite/Tailings Impoundment	*Forest Act*
Licence to Cut – Gravel Pits and Borrow Areas	*Forest Act*
Licence to Cut – Access Road	*Forest Act*
Licence to Cut – Transmission Line	*Forest Act*
Special Use Permit – Plant Access Road, Extension of Eskay	*Forest Act*
Road Use Permit – Eskay Road	*Forest Act*
Licence of Occupation – Borrow/Gravel Pits	*Land Act*
Licence of Occupation/Statutory Right of Way – Transmission Line	*Land Act*
Pipeline Permit – Diesel Pipeline	*Pipeline Act*
Surface Lease – Minesite Facilities	*Land Act*
Waste Management Permit – Effluent (Tailings & Sewage)	*Environmental Management Act*
Waste Management Permit – Air (Crushers, concentrator)	*Environmental Management Act*
Waste Management Permit – Refuse	*Environmental Management Act*
Camp Operation Permits (Drinking Water, Sewage Disposal, Sanitation and Food Handling)	*Health Act/Environmental Management Act*
Special Waste Generator Permit (Waste Oil)	*Environmental Management Act (Special Waste Regulations)*

Federal Approvals and Authorizations

Federal approvals include an authorization from the federal Minister of Environment approving the combined Application/Comprehensive Study Report for the KSM Project. Authorizations for major stream crossing will be required from the Department of Fisheries and Oceans under the Fisheries Act. Approvals for water crossings will also be required under the Navigable Waters Protection Act. An explosive factory licence will be required under the Explosives Act. The Metal Mining Effluent Regulation under the Fisheries Act, and administered by Environment Canada, will require a Schedule II amendment because the area proposed for the tailing impoundment facility contains fish habitat. Other federal requirements such as those in respect of radio communication will need licences. Table 18-2 lists some of the federal approvals required.

Table 18-2:
List of Federal Approvals and Licences Required to Develop the KSM Project

Federal Government Approvals & Licenses	**Enabling Legislation**
CEAA Approval	*Canadian Environmental Assessment*
Metal Mining Effluent Regulations (MMER)	*Fisheries Act* /Environment Canada
Fish Habitat Compensation Agreement	*Fisheries Act*
Section 35(2) Authorization	*Fisheries Act*
Navigable Water: Stream Crossings	*Navigable Waters Protection Act*
Explosives Factory Licence	*Explosives Act*
Ammonium Nitrate Storage Facilities	*Canada Transportation Act*
Radio Licences	*Radio Communication Act*
Radioisotope Licence (Nuclear Density	*Atomic Energy Control Act*
Dam Licence	*International River Improvements Act*

19.0 INTERPRETATION AND CONCLUSIONS

19.1 Data Collection and Results

In 2010, Seabridge drilled 118 core holes totaling about 30,000 meters at KSM with the primary objective of upgrading resources to a level required for a reserve definition. A significant proportion of the 2010 drilling program was focused on drilling the Iron Cap zone. Approximately 60 holes totaling about 8,500 meters were dedicated to geotechnical studies. The results from the geotechnical drill holes are being evaluated by various consultants and will be used in various mine planning and mine/plan infrastructure studies that will be in discussed in an updated prefeasibility study that is scheduled to be completed during the second quarter of 2010.

The geologic interpretation of the Kerr, Sulphurets, Mitchell, and Iron Cap zones was updated by Seabridge personnel incorporating the new drilling information. In general the overall interpretation of the deposits has not changed. A combination of alteration, and grade envelopes were developed for each zone and were used as the primary control for estimating block grades.

The author combined the 2010 drilling data with previously collected data to update the KSM resource estimate. Various exploratory data analyses were completed by RMI to establish modeling populations, grade capping limits, and spatial relationships. Gold, silver, copper, and molybdenum grades were estimated by inverse distance and nearest neighbor methods. The estimated grades were validated by visual and statistical methods and in the opinion of RMI, are globally unbiased and locally consistent with the current drill hole data.

The estimated block grades were classified into Measured (Mitchell only), Indicated and Inferred Mineral Resources using several criteria including mineralized continuity shapes, distance to data, and number of holes used in the estimate. Mineral Resources were tabulated and are summarized in Table 1-1.

Substantial metallurgical test work results indicate that the mineral samples from the four mineralized zones are amenable to the flotation-cyanidation process. The process consists of:

- Copper-gold-molybdenum bulk rougher flotation followed by gold-bearing pyrite flotation
- Regrinding the resulting bulk rougher concentrate followed by three stages of cleaner flotation to produce a copper-gold-molybdenum bulk cleaner flotation concentrate
- Molybdenum separation of the bulk cleaner flotation concentrate to produce a

molybdenum concentrate and a copper/gold concentrate containing associated silver

- Cyanide leaching of the gold-bearing pyrite flotation concentrate and the scavenger cleaner tailing to further recover gold and silver values as doré bullion.

The samples from the Mitchell and Sulphurets zones produced better metallurgical results with the chosen flotation circuit and cyanide leach extraction when compared to metallurgical results obtained from samples collected from the Iron Cap and Kerr zones.

19.2 **Data Density and Reliability**

In RMI's opinion, the drill hole data density within the Kerr, Sulphurets, Mitchell, and Iron Cap zones is sufficient to estimate and classify Mineral Resources. This is based on recognized mineralized continuity and supported by spatial analyses (i.e. variography). The author also believes that the data that were used to estimate Mineral Resources are reliable based on QA/QC results (blanks, standards, and check assays) and an audit of the electronic database.

19.3 **2010 Project Objectives**

It is the opinion of the author that Seabridge Gold successfully completed their stated 2010 project objectives by upgrading significant portions of the Sulphurets Inferred Mineral Resources to an Indicated category. The 2010 drilling program was very successful in outlining a substantial resource at the Iron Cap zone. Significant progress has also been made regarding metallurgy, mine planning exercises, along with a host of various permitting activities.

20.0 RECOMMENDATIONS

20.1 Resource Definition

- A modest drilling campaign of 10-15 core holes totaling around 3,000 meters could potentially upgrade currently defined Kerr Inferred Resources to Indicated. This program is estimated to cost about $1,000,000.

- There is potential to increase resources within the Sulphurets zone. The area with potential is located between the Canyon Zone, located at the southwest end of the deposit and the main zone of mineralization to the northeast. It is estimated that 30-35 holes would be required totaling about 10,000 meters. This program is estimated to be cost approximately $3,500,000.

- Infill drilling should be completed within key areas of the Iron Cap zone. Approximately 10-15 core holes totaling about 5,000 meters would increase the overall confidence level of the resource. This program is estimated to cost about $1,750,000.

- Additional bulk density determinations should be collected from future Iron Cap drill hole samples focusing on representative mineralized and unmineralized rocks. The cost for this program is nominal since the work will be carried out by staff personnel.

20.2 Mining

- Continue with geotechnical studies for determining possible pit slope angles for each of the four zones. Seabridge has been working several geotechnical consulting companies to determine appropriate pit slope angles. The author is unaware of the magnitude of costs associated with these activities.

20.3 Processing

- Additional metallurgical test work and mineralogical evaluations should be conducted to optimize process conditions and to establish design-related parameters for the next stage of study. The test work should include variability testing of samples from Sulphurets, Kerr and Iron Cap zones. The cost of the test work is estimated at $500,000.

- Further investigation of the separation between copper and molybdenum from the bulk concentrate should be included in the next study phase. The potential additional value of rhenium in the molybdenum concentrate should be evaluated at an estimated cost of $150,000.

- Further study should be conducted to optimize the proposed cyanide recovery and destruction methods. The cost is estimated to be approximately $100,000.

Test work to confirm the slurry pumping arrangement to deliver the ore slurry from mine site and plant site should be conducted to confirm the current preliminary design. The cost for this is estimated to be approximately $200,000.

20.4 Permitting

- Continue gathering environmental data and working on the various permits that will be required (see Tables 18-1 and 18-2). The authors are unaware of the costs estimated to complete these activities.

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22.0 DATE AND SIGNATURE PAGE

I, Michael J. Lechner, of Stites, Idaho do hereby certify:

1. That I am an independent consultant and owner of Resource Modeling Incorporated, an Arizona Corporation with it's office located at 124 Lazy J Drive, PO Box 295, Stites, ID 83552

2. That I am a registered professional geologist in the State of Arizona (#37753), a Certified Professional Geologist with the AIPG (#10690), and a P. Geo. in British Columbia (#155344).

3. That I am a graduate of the University of Montana (1979) with a Bachelor of Arts degree in Geology.

4. That I have practiced my profession continuously since 1977 amd have worked as an exploration geologist, mine geologist, Engineering Superintendent, resource modeler, and consultant on a wide variety of base and precious metal deposits throughout the world.

5. This certificate applies to technical report entitled "March 2011 Updated KSM Mineral Resources", with an effective date of March 29, 2011 (the "Technical Report").

6. 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 and professional registration (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. That I, Michael J. Lechner, performed various statistical and geostatistical analyses of the drill hole data and independently estimated gold resources for the Kerr, Sulphurets, Mitchell, and Iron Cap zones. Various descriptions of the site location, geologic setting, property history, and other historical information were prepared by Mr. Mike Savell and Mr. Tim Dodd from Seabridge Gold. I'm responsible for portions of Sections 1, 19, and 20. I'm responsible for all of Sections 2-15, and 17-18 of the Technical Report.

8. That as of the date of this certificate, I am not aware of any material fact or material change with regard to the property that would make the Technical report misleading.

9. That I have written this report as an independent consulting geologist and have no material interest, direct or indirect, in the property discussed in this report and have not had any prior involvement with this property prior to

working with Seabridge Gold.

10. I have read NI 43-101 and fully believe that this report has been written in complete compliance with that Instrument.

11. This Technical Report was prepared for Seabridge Gold Inc. by Mr. Michael Lechner, President of Resource Modeling Incorporated. The report is based almost exclusively on data that were provided to Resource Modeling Inc. by Seabridge Gold Inc. Resource Modeling Incorporated disclaims all liability for the underlying data and do not accept responsibility for the interpretations and representation made in this report where they were a result of erroneous, false, or misrepresented data. Resource Modeling Inc. disclaims any and all liability for representations or warranties, expressed or implied, contained in, or for omissions from, this report or any other written or oral communications transmitted or made available to any interested party when done without written permission or when they are inconsistent with the conclusions and statements of this report.

Signed and Sealed

"Michael J. Lechner"

March 29, 2011

I, Jiahhui (John) Huang, of Burnaby, BC, do hereby certify:

1. I am a Senior Metallurgist with Wardrop Engineering Inc. with a business address at #555-800 W. Hastings St., Vancouver, BC, V6B 1M1.

2. This certificate applies to the technical report entitled “March 2011 Updated KSM Mineral Resources”, with an effective date of March 29, 2011 (the “Technical Report”).

3. I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000).

4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30898).

5. My relevant experience includes over 28 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores.

6. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

7. My most recent personal inspection of the Property was September 16, 2008.

8. I am responsible for all of Section 16 for matters pertaining to metallurgical testing review in the Technical Report and portions of Sections 1, 19, and 20.

9. I am independent of Seabridge Gold Inc. as defined by Section 1.4 of the Instrument.

10. I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008”, dated December 22, 2008, the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009” dated September 8, 2009, and the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study” dated March 31, 2010.

11. I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 29th day of March, 2011 at Vancouver, BC

*"Original document signed and sealed by
Jianhui (John) Huang, P.Eng."*

Jianhui (John) Huang, P.Eng.
Senior Metallurgist
Wardrop Engineering Inc.

23.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

Not applicable.