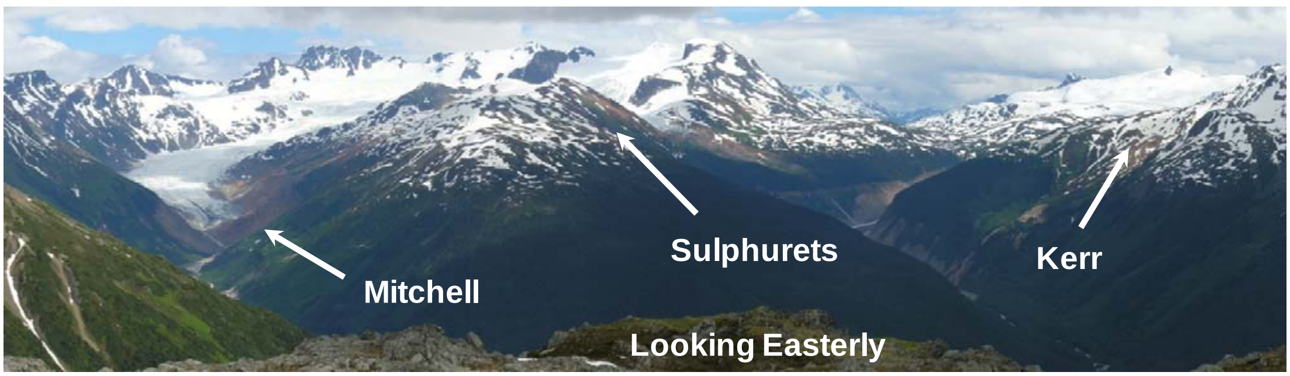
January 2010 Updated KSM Mineral Resources



Prepared for

Seabridge Gold Inc.
106 Front Street East
Suite 400
Toronto, Ontario Canada
M5A 1E1

Prepared by

Michael J. Lechner, P. Geo #155344
Resource Modeling Incorporated
124 Lazy J Drive, PO Box 295
Stites, ID 83552

January 25, 2010

Table of Contents

List of Figures ....Page

List of Tables ....Page

1.0 SUMMARY

1.1 Location and Ownership

The Kerr-Sulphurets-Mitchell (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 three 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 property consists of 46 contiguous mineral claims and 19 contiguous placer claims that cover an area of about 12,951 hectares. The Mitchell zone is located within a block of two contiguous claim blocks that make up the Kerr and Sulphurets (or Sulphside) properties. 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. 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, and Mt. Milligan.

The Mitchell Zone is underlain by foliated, schistose, volcanic and volcaniclastic rocks that are exposed below the shallow north dipping Mitchell thrust fault. These rocks tend to be highly altered (primarily phyllic and propylitic) and characterized by intense sericitization, abundant pyrite with numerous quartz vein stockworks and sheeted quartz veins that are often deformed and flattened. Towards the west end of the property, the extent and strength of phyllic alteration diminishes and chlorite (and/or green sericite) ± magnetite alteration becomes more dominant. Within the core of the deposit, pyrite content ranges between 5 to 20% and typically occurs as fine disseminations.

Gold and copper mineralization tends to be relatively low-grade but dispersed over a very large area and appears to be related to hydrothermal activity associated with Early Jurassic hypabyssal porphyritic intrusions. 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 core of the deposit, gold and copper grades tend to correlate well with one another. Preliminary work indicates that gold is intimately associated with chalcopyrite. Copper/gold ratios tend to be slightly higher near the phyllic-propylitic alteration contact zones. In low-grade areas, which tend to flank the higher-grade core, copper/gold ratios tend to be highest. In general, within the currently drilled area, gold and copper grades tend to be remarkably consistent between drill holes, which is consistent with a large, stable hydrothermal system with a low thermal gradient.

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, and 2009 field seasons. Since entering into the district, Seabridge has drilled 146 diamond core holes totaling about 54,426 meters.

The author has reviewed all of the available drilling data for all three zones. Based on the result of those reviews, it is the opinion of the author that the Kerr, Sulphurets, and Mitchell zones represent large copper-gold resources.

The author updated the estimate of Mineral Resources for the Kerr, Sulphurets, and Mitchell zones by creating three-dimensional block models. Gold and copper grades were estimated using 15-meter-long drill hole composites by inverse distance and nearest neighbor methods. The author validated the estimated block grades using visual and statistical methods. It is the author's opinion that the grade models are globally unbiased and represents a reasonable estimate of in situ resources. The author classified a portion of the estimated blocks 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, and Mitchell 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.

Table 1-1: Summary of KSM Mineral Resources

Zone	Measured Mineral Resources					Indicated Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)
Kerr	No Measured Resources					237,500	0.26	0.48	1,985	2,513
Sulphurets	No Measured Resources					159,000	0.63	0.28	3,221	981
Mitchell	659,700	0.64	0.17	13,574	2,472	1,080,900	0.58	0.17	20,156	4,050
Total	**659,700**	**0.64**	**0.17**	**13,574**	**2,472**	**1,477,400**	**0.53**	**0.23**	**25,362**	**7,544**

Zone	Measured + Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)
Kerr	237,500	0.26	0.48	1,985	2,513	76,100	0.20	0.30	489	503
Sulphurets	159,000	0.63	0.28	3,221	981	144,000	0.50	0.16	2,317	511
Mitchell	1,740,600	0.60	0.17	33,730	6,522	537,000	0.44	0.14	7,597	1,657
Total	**2,137,100**	**0.57**	**0.21**	**38,936**	**10,015**	**757,100**	**0.43**	**0.16**	**10,403**	**2,671**

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). Work is currently underway in preparing a Pre-feasibility Study (PFS) using the updated Mineral Resources that are the subject of this report. The list of consultants involved in updating the PEA and working on the PFS include:

- Resource Modeling Inc. (RMI)
- Moose Mountain Technical Services (MMTS)
- TJS Mining-Met Services Inc. (TJS)
- G & T Metallurgical Services Ltd. (G&T)
- WN Brazier Associates Inc. (Brazier)
- Klohn Crippen Berger Ltd. (KCBL)
- Bosche Ventures Ltd. (BVL)
- McElhanney Consulting Services, Ltd. (McElhanney)
- BGC Engineering Inc. (BGC)
- Rescan Environmental Services Ltd. (Rescan)

1.4 **Conclusions**

Seabridge Gold's 2006 through 2009 drilling programs have confirmed the presence of a large, disseminated, gold-copper system known as the Mitchell zone, with average grades in the order of 0.8 g/t Au and 0.2% Cu. The geology, dimensions and metal distribution of the Mitchell deposit are consistent with those of a gold-enriched, low-grade copper porphyry model. There are no apparent significant hard grade boundaries except the Mitchell thrust fault which separates more permissive lower plate units from upper plate lithologies with less extensive mineralization. The Mitchell deposit still remains somewhat open in the down-dip direction although the 2009 drilling campaign has demonstrated local limits to mineralization. The deposit also appears to somewhat open to the south and southeast although grades tend to be lower in those vectors. Pit optimization studies should be carried out to determine what type of economics may be required to chase the mineralized system to depth. RMI's preliminary conceptual pits (see Section 17-11) mined a significant portion of the updated resources using all resource categories. After new costs and recoveries have been established a series of pit shells need to be generated, using only Measured and/or Indicated Mineral Resources for "feed blocks".

The Sulphurets zone, while smaller than the Mitchell zone, represents an attractive target due to its proximity to the Mitchell zone (and possible shared revenues), higher copper grades, and near surface exposures. Additional drilling will be required to close off the deposit and to upgrade the current Inferred Mineral Resources to higher confidence categories. Drilling in 2009 helped to confirm mineralization within the "Main Copper" zone, which had been identified in the 1990's by Placer Dome and earlier companies. Additional drilling will be required to upgrade the currently classified Inferred material to higher categories.

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 a beneficial for blending purposes.

1.5 Recommendations

- The Sulphurets deposit remains open along strike from the "Canyon Zone" at the southwest end of the deposit northeasterly towards the main zone of mineralization. Drilling should target permissive geometry along strike or down-dip from existing gold intercepts. This program should be carefully designed with contingencies for dropping or adding holes based on the drilling results. The cost for such a program depends on the number of meters drilled but could easily exceed several million dollars.

- Additional drilling should be completed within the Main Copper Zone at Sulphurets to determine its limits and to upgrade currently defined Inferred Mineral Resources. The cost for such a program depends on the number of meters drilled but could range between $500,000 and $1,000,000.

- Seabridge drilled 7 holes into the Kerr deposit in 2009 and in general confirmed prior Placer Dome results. Additional drilling is recommended to 1) provide new samples for metallurgical testwork, and 2) examine core recovery/geotechnical issues that were reported with the early 1990 drilling. This drilling should be conducted with triple tube core barrels and an adequate mud program to enhance core recovery. The "rubble breccia" zones (anhydrite/gypsum) that were identified by Placer Dome should be targeted. The cost for such a program depends on the number of meters drilled but could easily exceed several million dollars.

- If possible, test the continuity of mineralization between the Mitchell and Iron Cap deposits by drilling methods. Little is understood about the Iron Cap zone other than quartz-sericite-pyrite alteration is more intense than at Mitchell and there appears to be more base metal mineralization, particularly in narrow veins. The 2005 Falconbridge holes intersected low-grade gold mineralization near the surface. Offset holes from existing known mineralization should be designed to aid in determining the possible geometry of mineralization and its possible relationship to the nearby Mitchell deposit. The cost for such a program depends on the number of meters drilled.

- Continue with geotechnical studies for determining possible pit slope angles for the Mitchell deposit. Similar studies will be required for the Sulphurets and Kerr deposit. Seabridge has been working with a geotechnical consulting company to recommend pit slope angles and other infrastructure geotechnical design parameters. The author is unaware of the magnitude of costs associated with these activities.

- Continue ongoing metallurgical studies to determine potential recoveries and process flowsheets. Seabridge has contracted TJS Mining-Met Services Inc. to manage and direct a number of metallurgical consulting firms to complete these activities. The author is unaware of the costs associated with these activities.

- 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.

2.0 INTRODUCTION

The Kerr-Sulphurets-Mitchell (KSM) copper-gold project is currently 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, and quality assurance/quality control (QA/QC) protocols. The author'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.

Seabridge Gold provided the author 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. Timothy Dodd, Mr. Mike Savell, 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. James Smolik regarding metallurgical testing.

Mr. Michael J. Lechner, President of Resource Modeling Inc. 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.

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 the author using data that were provided to him by Seabridge Gold. Over 95% of the drilling data for the Mitchell deposit has been collected by Seabridge Gold. The author has personally verified the assay data that have been collected from Seabridge's 2006, 2007, 2008, and 2009 field seasons. A significant portion of the Sulphurets data and Kerr drilling data were collected by other companies prior to Seabridge’s acquisition of the property.

In preparing this document the author 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, 2009).

The author requested and received several metallurgical reports that were generated by consulting groups that were contracted by Seabridge Gold. These groups include G & T Metallurgical Services, Ltd. and SGS Lakefield Mineral Services (Wardrop, 2009).

To the best of the author’s knowledge there are no environmental liabilities or other liens against the property. In 2003, an environmental evaluation of the Kerr-Sulphurets property was undertaken by Stantec Engineering for Falconbridge. A reclamation program addressing surface disturbances resulting from historical exploration work was recommended by Stantec and undertaken in 2004 by Falconbridge. This reclamation work was deemed to be satisfactory by the Ministry of Energy, Mines, and Petroleum Resources. The Stantec study noted that there are extensive areas of naturally occurring sulfide minerals (mostly pyrite) which have been exposed by erosion and glaciation. Natural oxidation of sulfide minerals results in acidic drainage with elevated metal content. This has been occurring over a geological (quaternary) time scale.

Seabridge has contracted Rescan, a leading Canadian environmental permitting company to direct and manage all aspects of permitting the KSM project. This work is currently underway and the author is unaware of any negative initial findings by Rescan.

This report was prepared for Seabridge by the author and is based in part by information not within the control of either Seabridge or the author, although the majority of the Mitchell Creek data were only recently 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 is unaware of any existing technical data other than those that were provided to him by Seabridge. 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.

4.0 PROJECT DESCRIPTION AND LOCATION

The Mitchell property, along with the Kerr and Sulphurets deposits, is located within a package of 46 contiguous mineral claims and 19 contiguous placer claims that are summarized in Table 4-1 and Table 4-2, respectively (Savell, 2009). The mineral claims cover an area of approximately 12,951 hectares while the placer claims cover about 4,554 hectares. It should be noted that most of the placer claims lie “over the top” of the mineral claims and secure the rights to minerals contained in unconsolidated overburden. Seabridge also owns 53 contiguous mineral claims totaling 21,478 hectares (Seabee Property) that are located about 19 kilometers northeast of the Kerr-Sulphurets-Mitchell property and are summarized in Table 4-3. The Seabee property covers locations of proposed processing facilities and tailings storage areas.

In 2009 Seabridge added an additional 8,975 hectares through the purchase of 22 mineral claims from Max Minerals Ltd.

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 subject to 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 Kerr-Sulphurets-Mitchell mineral claims purchased from Placer Dome in 2000 were converted from 58 legacy claims to B.C.’s new Mineral Titles Online (MTO) system in 2005. Eleven legacy placer claims were converted in 2005 to nine cell placer claims. Ten cell placer claims were added in 2008 to the property and are contiguous with the converted legacy placer claims. 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 Kerr-Sulphurets-Mitchell). 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 project consists 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.

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 mineral claims in the Mitchell and Sulphurets drainages to 2018. The Kerr-Sulphurets placer claims have been kept in good standing by paying fees in lieu of completing assessment work. Assessment work was completed on most of the Seabee claims in 2008 with that work filed in February 2009 which advanced expiry dates to 2012. 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 schedule of 2010 maintenance costs for the KSM and Seabee properties is in Table 4-4.

Table 4-1: KSM Claim Group Mineral Claims

Tenure No.	Claim Name	Cells or Units	Area (hectares)	TRIM Map No.	Mining Division	Expiry Date
516236	ICE 4	17	303.3	104B059	Skeena	June 30, 2018
516237	ICE 2	4	71.4	104B059	Skeena	June 30, 2018
516238	OK #1	35	624.5	104B059	Skeena	December 10, 2018
516239	OK #2	30	535.5	104B059	Skeena	December 10, 2018
516240	ICE 1	6	107.0	104B059	Skeena	June 30, 2018
516241	IRON CAP 4	8	142.7	104B059	Skeena	June 30, 2018
516242	IRON CAP 6	1	71.4	104B059	Skeena	September 23, 2018
516245	XRAY 1	20	356.9	104B059	Skeena	October 12, 2018
516248	TEDRAY NO. 1	8	142.7	104B059	Skeena	August 26, 2018
516251	TEDRAY NO. 6	18	321.3	104B059	Skeena	August 26, 2018
516252	ED NO. 1	7	125.0	104B059	Skeena	August 26, 2018
516253	ED NO. 2	10	178.6	104B059	Skeena	August 26, 2018
516254	TEDRAY NO. 9	16	285.8	104B059	Skeena	August 26, 2018
516255	TEDRAY 15	12	214.3	104B049	Skeena	September 23, 2018
516256	TEDRAY NO. 11	3	53.6	104B049	Skeena	August 26, 2018
516258	TEDRAY 16	6	178.6	104B059	Skeena	November 3, 2018
516259	TEDRAY 17	10	107.2	104B049	Skeena	November 3, 2018
516260	TEDRAY 18	6	107.2	104B049	Skeena	November 3, 2018
516261	KERR 41	26	464.6	104B049	Skeena	December 20, 2018
516262	KERR 10	19	339.5	104B049	Skeena	December 17, 2018
516263	KERR 15	36	643.9	104B049	Skeena	December 17, 2018
516264	KERR 99	22	393.3	104B049	Skeena	October 30, 2018
516266	KERR 8	10	178.8	104B049	Skeena	December 17, 2018
516267	KERR 9	1	250.2	104B049	Skeena	December 17, 2018
516268	KERR 12	18	321.8	104B049	Skeena	December 17, 2018
516269	TEDRAY 13	6	107.2	104B049	Skeena	August 26, 2018
254756	ARBEE #35	1	25.0	104B059	Skeena	June 16, 2018
254757	ARBEE #39	1	25.0	104B059	Skeena	June 16, 2018
254758	ARBEE #54	1	25.0	104B059	Skeena	June 16, 2018
254759	ARBEE #55	1	25.0	104B059	Skeena	June 16, 2018
394782	BJ 7	25	500.0	104B059	Skeena	December 11, 2010
394783	BJ 8	25	500.0	104B059	Skeena	December 11, 2010
394784	BJ 9	20	400.0	104B059	Skeena	December 11, 2010
394792	BJ 16	25	500.0	104B059	Skeena	December 11, 2010
394793	BJ 17	20	400.0	104B059	Skeena	December 11, 2010
394795	BJ 19	25	500.0	104B059	Skeena	December 11, 2010
394796	BJ 20	18.75	375.0	104B059	Skeena	December 11, 2010
394799	BJ 23	25	500.0	104B059	Skeena	December 11, 2010
394800	BJ 24	15	300.0	104B059	Skeena	December 11, 2010
394801	BJ 25	25	500.0	104B059	Skeena	December 11, 2010
394802	BJ 26	12.5	250.0	104B059	Skeena	December 11, 2010
394803	BJ 27	10	200.0	104B059	Skeena	December 11, 2010
394804	BJ 28	5	100.0	104B059	Skeena	December 11, 2010
394805	BJ 29	15	300.0	104B049	Skeena	December 11, 2010
394806	BJ 30	20	400.0	104B049	Skeena	December 11, 2010
394807	BJ 31	25	500.0	104B049	Skeena	December 11, 2010

Table 4-2: KSM Claim Group Placer Claims

Tenure No.	Claim Name	Cells or Units	Area (hectares)	TRIM Map No.	Mining Division	Expiry Date
516323	PLACER CLAIM	6	107.2	104B049	SKEENA	September 30, 2010
516325	PLACER CLAIM	7	125.0	104B049	SKEENA	September 30, 2010
516328	PLACER CLAIM	4	71.5	104B049	SKEENA	September 28, 2010
516330	PLACER CLAIM	6	107.2	104B049	SKEENA	September 28, 2010
516332	PLACER CLAIM	6	107.2	104B049	SKEENA	September 28, 2010
516333	PLACER CLAIM	5	89.3	104B049	SKEENA	September 28, 2010
516375	PLACER CLAIM	7	125.0	104B049	SKEENA	September 30, 2010
516676	PLACER CLAIM	1	17.9	104B059	SKEENA	September 30, 2010
516677	PLACER CLAIM	1	17.9	104B059	SKEENA	July 11, 2010
576658	KERR PL1	22	446.9	104B049	SKEENA	February 20, 2010
576659	KERR PL2	22	446.6	104B049	SKEENA	February 20, 2010
576660	KERR PL3	22	446.4	104B059	SKEENA	February 20, 2010
576661	KERR PL4	22	446.2	104B059	SKEENA	February 20, 2010
576662	KERR PL5	22	446.0	104B059	SKEENA	February 20, 2010
576663	KERR PL6	22	446.0	104B059	SKEENA	February 20, 2010
576664	KERR PL7	7	142.7	104B059	SKEENA	February 20, 2010
576665	KERR PL8	16	321.4	104B059	SKEENA	February 20, 2010
576666	KERR PL9	14	285.7	104B059	SKEENA	February 20, 2010
576667	KERR PL10	18	357.4	104B049	SKEENA	February 20, 2010
694483	KSM P1	20	357.4	104B049	SKEENA	January 5, 2011
694543	KSM P2	23	410.5	104B059	SKEENA	January 5, 2011
694683	KSM P3	24	427.9	104B059	SKEENA	January 5, 2011

Table 4-3: Seabee Property Mineral Claims

Tenure No.	Claim Name	Cells or Units	Area (hectares)	TRIM Map No.	Mining Division	Expiry Date
566467	BRIDGE1	25	445.8	104A052	Skeena	February 8, 2012
566468	BRIDGE2	25	445.6	104A052	Skeena	February 8, 2012
566469	BRIDGE3	24	427.8	104A052	Skeena	February 8, 2012
566470	BRIDGE4	24	428.0	104A052	Skeena	February 8, 2012
566471	BRIDGE5	25	445.7	104A052	Skeena	February 8, 2012
566472	BRIDGE6	25	445.6	104A052	Skeena	February 8, 2012
566473	BRIDGE7	24	427.9	104A052	Skeena	February 8, 2012
566474	BRIDGE8	24	427.8	104A052	Skeena	February 8, 2012
566475	BRIDGE9	24	427.6	104A052	Skeena	February 8, 2012
566476	BRIDGE10	25	445.5	104A052/053	Skeena	February 8, 2012
566477	BRIDGE11	17	302.9	104A052/053	Skeena	February 8, 2012
566478	BRIDGE12	24	427.4	104A061	Skeena	February 8, 2012
566479	BRIDGE13	25	445.2	104A061	Skeena	February 8, 2012
566481	BRIDGE14	25	445.1	104A061	Skeena	February 8, 2012
566482	BRIDGE15	25	444.8	104A061	Skeena	February 8, 2012
566484	BRIDGE16	25	444.6	104A061	Skeena	February 8, 2012
566485	BRIDGE17	24	426.7	104A061	Skeena	February 8, 2012
566487	BRIDGE18	25	444.7	104A061	Skeena	February 8, 2012
566488	BRIDGE19	25	444.8	104A061	Skeena	February 8, 2012
566489	BRIDGE20	25	445.0	104A061	Skeena	February 8, 2012
566490	BRIDGE21	24	427.3	104A061	Skeena	February 8, 2012
566491	BRIDGE22	25	445.2	104A061	Skeena	February 8, 2012
566492	BRIDGE23	24	427.3	104A061/104B070	Skeena	February 8, 2012
566493	BRIDGE24	24	427.9	104A052	Skeena	February 8, 2012
566494	BRIDGE25	24	427.9	104A052/053	Skeena	February 8, 2012
566495	BRIDGE26	25	444.9	104A061/104B070	Skeena	February 8, 2012
566496	BRIDGE27	22	391.3	104B070	Skeena	February 8, 2012
566497	BRIDGE28	25	444.5	104A061/104B070	Skeena	February 8, 2012
566567	BRIDGE29	24	427.5	104A052/062	Skeena	February 8, 2012
571582	SEABEE1	23	408.8	104A061	Skeena	February 8, 2012
571583	SEABEE2	21	373.1	104A061	Skeena	February 8, 2012
571584	SEABEE3	25	444.1	104A061,071	Skeena	February 8, 2012
571585	SEABEE4	24	426.1	104A071	Skeena	February 8, 2012
571586	SEABEE5	21	372.6	104A071	Skeena	February 8, 2012
571587	SEABEE6	9	159.6	104A071	Skeena	February 8, 2012
573813	SEABEE7	12	213.3	104A071	Skeena	February 8, 2012
575633	SEA 1	25	445.2	104A051	Skeena	February 8, 2012
575635	SEA 2	25	445.3	104A061	Skeena	February 8, 2012
575636	SEA 3	25	445.4	104A061	Skeena	February 8, 2012
575638	SEA 4	25	445.4	104A061	Skeena	February 8, 2012
575639	SEA 5	25	445.3	104A061	Skeena	February 8, 2012
575642	SEA 6	25	445.1	104A051	Skeena	February 8, 2012
575643	SEA 7	12	213.4	104A051	Skeena	February 8, 2012
575645	SEA 8	24	427.1	104A051	Skeena	February 8, 2012
575646	SEA 9	2	35.6	104B070	Skeena	February 8, 2012
603133	SEABEE 8	24	426.6	104B070	Skeena	April 21, 2010
603134	SEABEE 9	3	53.4	104B070	Skeena	April 21, 2010
401548	TINA 1	25	500.0	104B070	Skeena	February 28, 2010
401549	TINA 2	25	500.0	104B070	Skeena	February 28, 2010
401550	TINA 3	25	500.0	104B070	Skeena	February 28, 2010
401551	TINA 4	25	500.0	104B070	Skeena	February 28, 2010
401552	TINA 5	25	500.0	104B070	Skeena	February 28, 2010
401553	TINA 6	13	250.0	104B070	Skeena	February 28, 2010

Table 4-4: 2010 Mining Claim Maintenance Costs

Claim Group/Property	No. of Tenures	Area (HA)	Next Expiry or Due Year	Required Work or Cash in 2010 (C$)		
				February	April	July
KSM Placer Claims	10	3,785.38	2010	$45,424.56		
KSM Placer Claims	1	17.86	2010			$214.30
SEABEE	2	479.94	2010		$2,111.73	
SEABEE-TINA	6	2,750.00	2010	$23,100.00		
Total	19	7,033.18	n/a	$68,524.56	$2,111.73	$214.30

The KSM Project is located on provincial Crown land. The three 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 Mass 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 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 2 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.

The 2009 drilling program proposal (Notice of Work and Reclamation Program) was submitted to the Ministry of Energy, Mines and Petroleum Resources on March 19, 2009. Approval was received by Mines Act Permit No. MX-1-571 granted June 1, 2009 by the Ministry of Energy, Mines, and Petroleum Resources, and Free Use Permit No. 18281 granted April 23, 2009 by the Ministry of Forests. Seabridge's exploration programs are conducted under the permits issued in 2009.

Figure 4-1: General Location Map

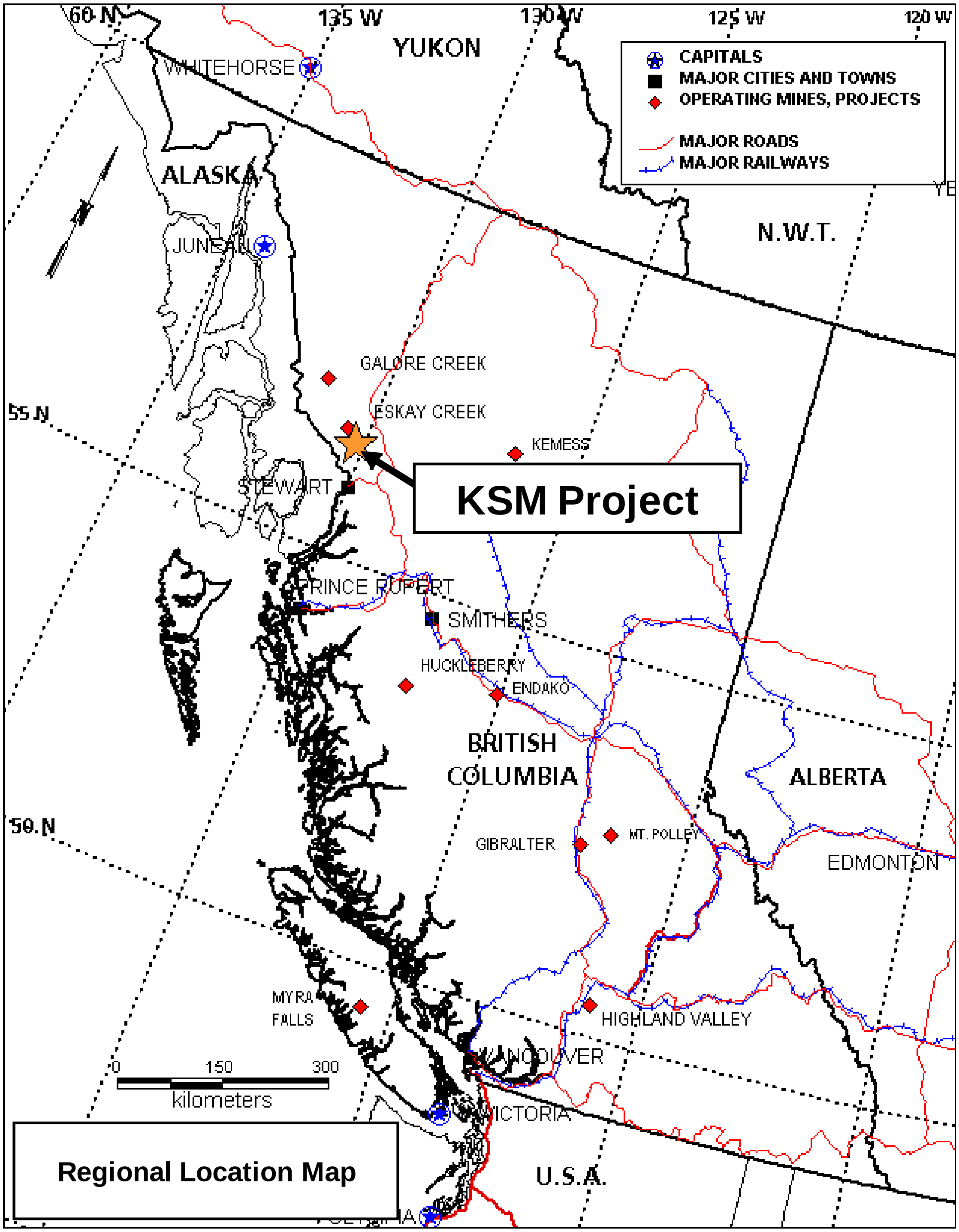
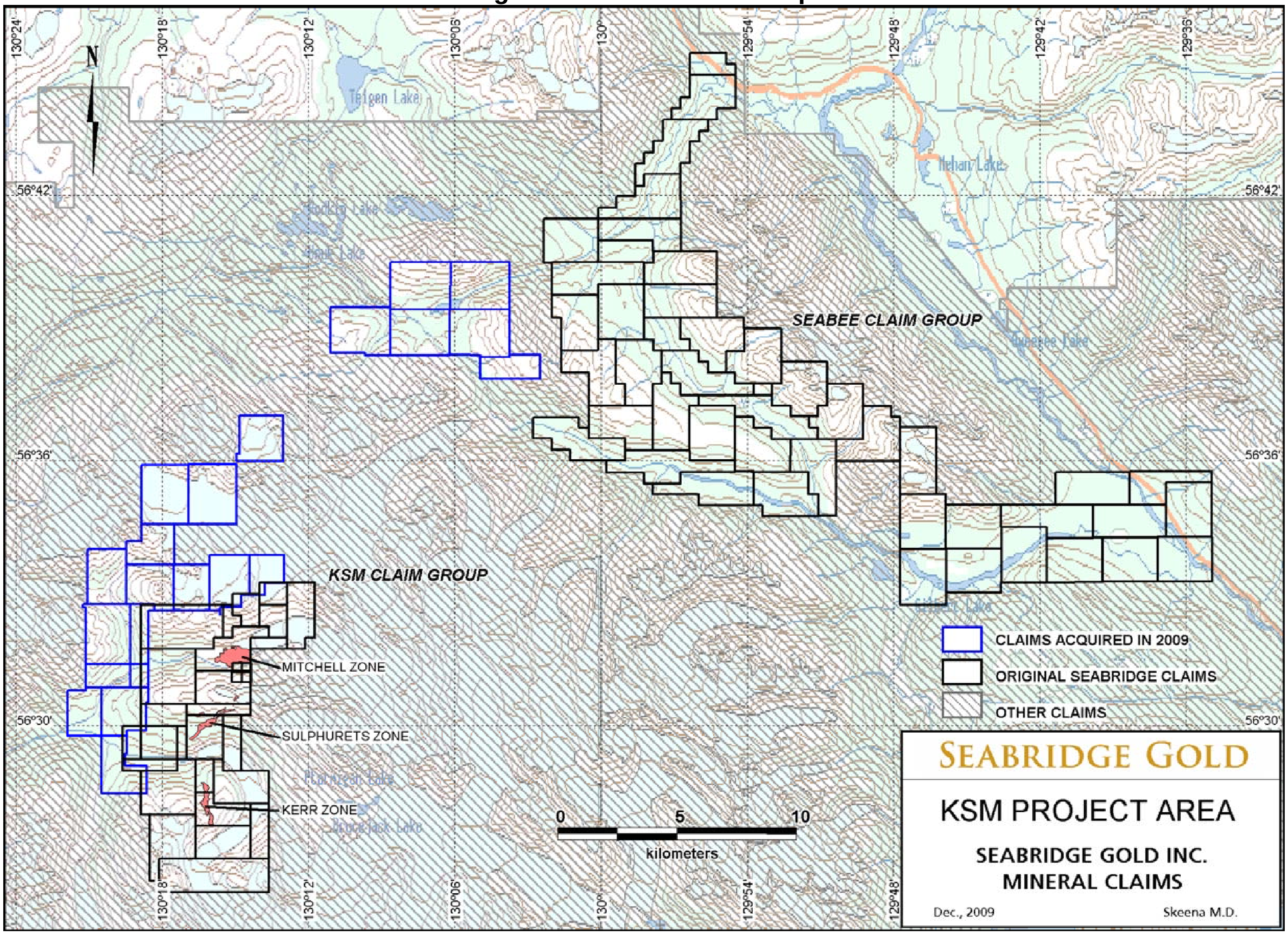


Figure 4-2: KSM Claim Map



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. For the 2006 drilling program, an Astar 350B2 was chartered from Mustang Helicopters out of Red Deer, Alberta. 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. The Eskay Creek Mine produces its own diesel generated power. 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 property."

Table 6-1: Exploration Summary of the Kerr Property

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

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

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

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, and other
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.

6.2 Historical Resource Estimates

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

6.3 History of Production

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

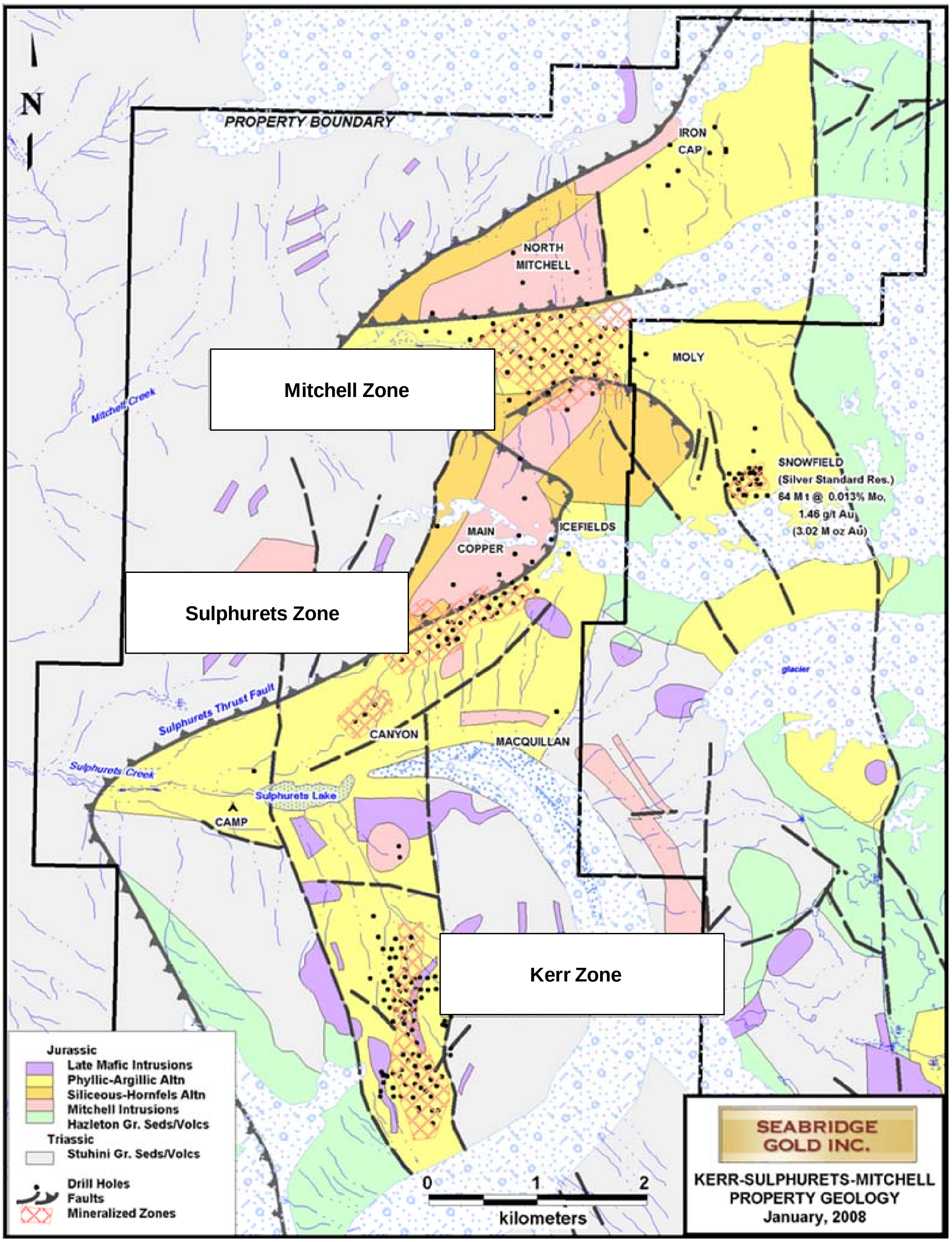
Early Jurassic sub-volcanic intrusive complexes are common in the Stikinia terrane, and several host well-known precious and base metal rich hydrothermal systems. These include copper-gold porphyry deposits such as Galore Creek, Red Chris, Kemess, Mt. Milligan, and Kerr-Sulphurets. In addition, there 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/copper mineralized zones.

Figure 7-1: Generalized Geologic Map



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 26,000 meters of core drilling in 144 drill holes spaced at intervals of 50 to 100 meters by six previous operators between 1987 and 1991. In 2009, Seabridge drilled an additional 877 meters in four core holes. These drill holes were within or immediately down dip of the previously estimated resource. Geological and assay results were consistent with existing models for 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.

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. The line of section for the two cross sections is shown on the plan map (Figure 9-1) in blue.

Figure 9-1: Kerr Geology Plan Map

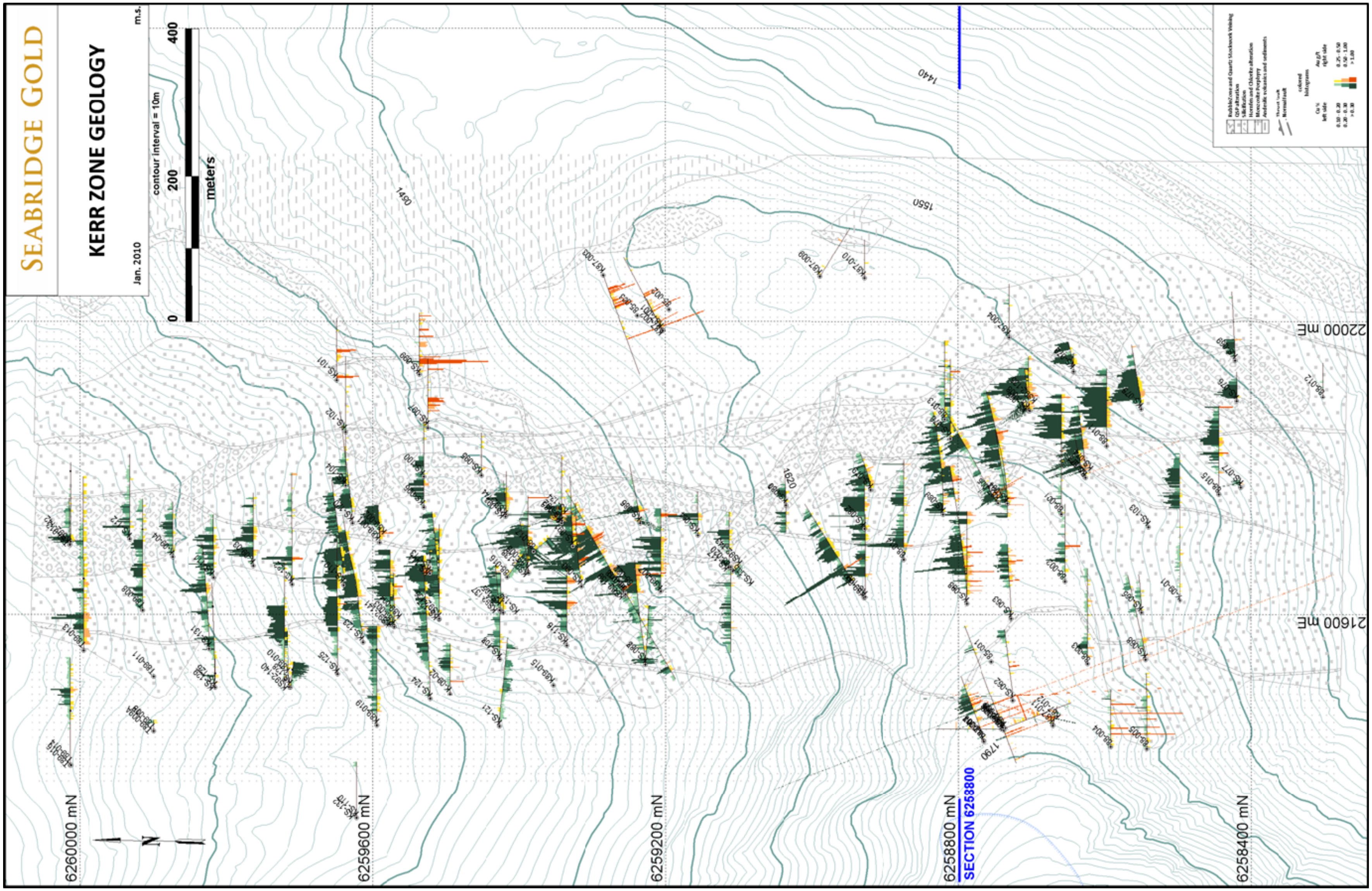


Figure 9-2: Kerr Geologic Section 6,258,800 N (Au)

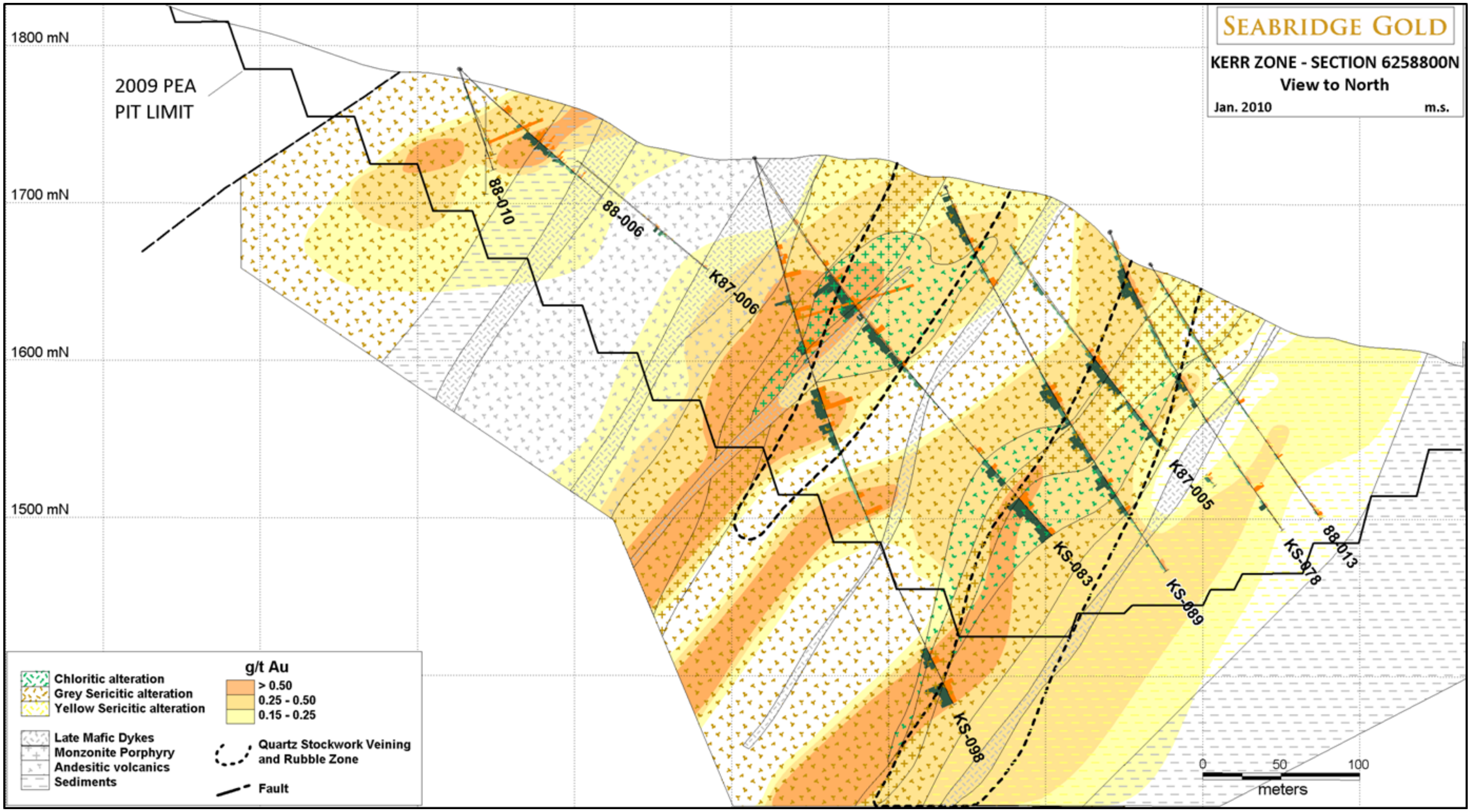
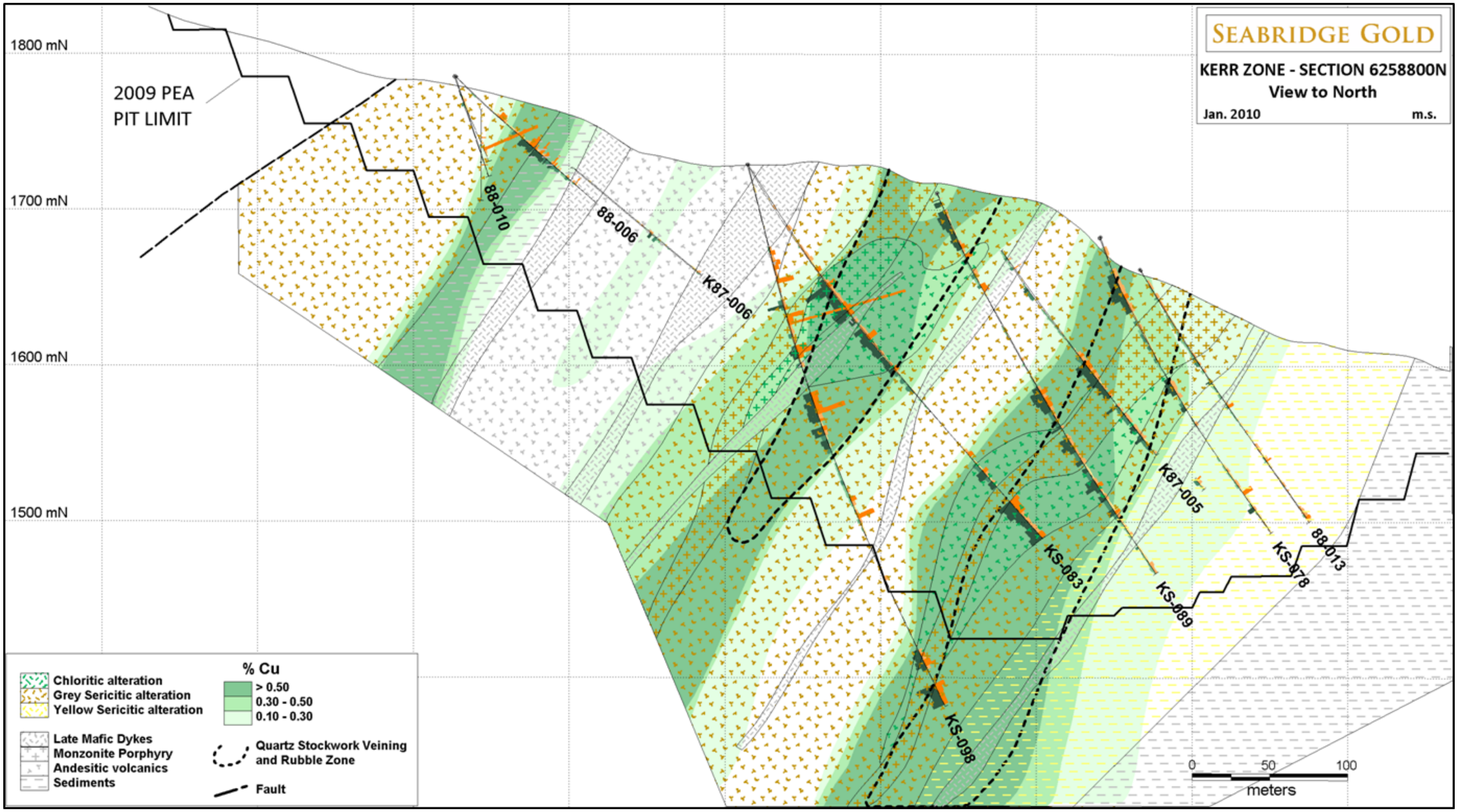


Figure 9-3: Kerr Geologic Section 6,258,800 N (Cu)



9.2 Sulphurets Zone

The Sulphurets deposit has been delineated by over 15,200 meters of core drilling in 65 drill holes spaced at intervals of 50 to 100 meters; in total six different operators drilled the project between 1968 and 2006.

In 2009, Seabridge drilled an additional 3,081 meters in seven core holes. These drill holes were within or immediately down dip of the previously estimated resource. 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.

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. The line of section for the two cross sections is shown on the plan map (Figure 9-4) in blue.

Figure 9-4: Sulphurets Geology Plan Map

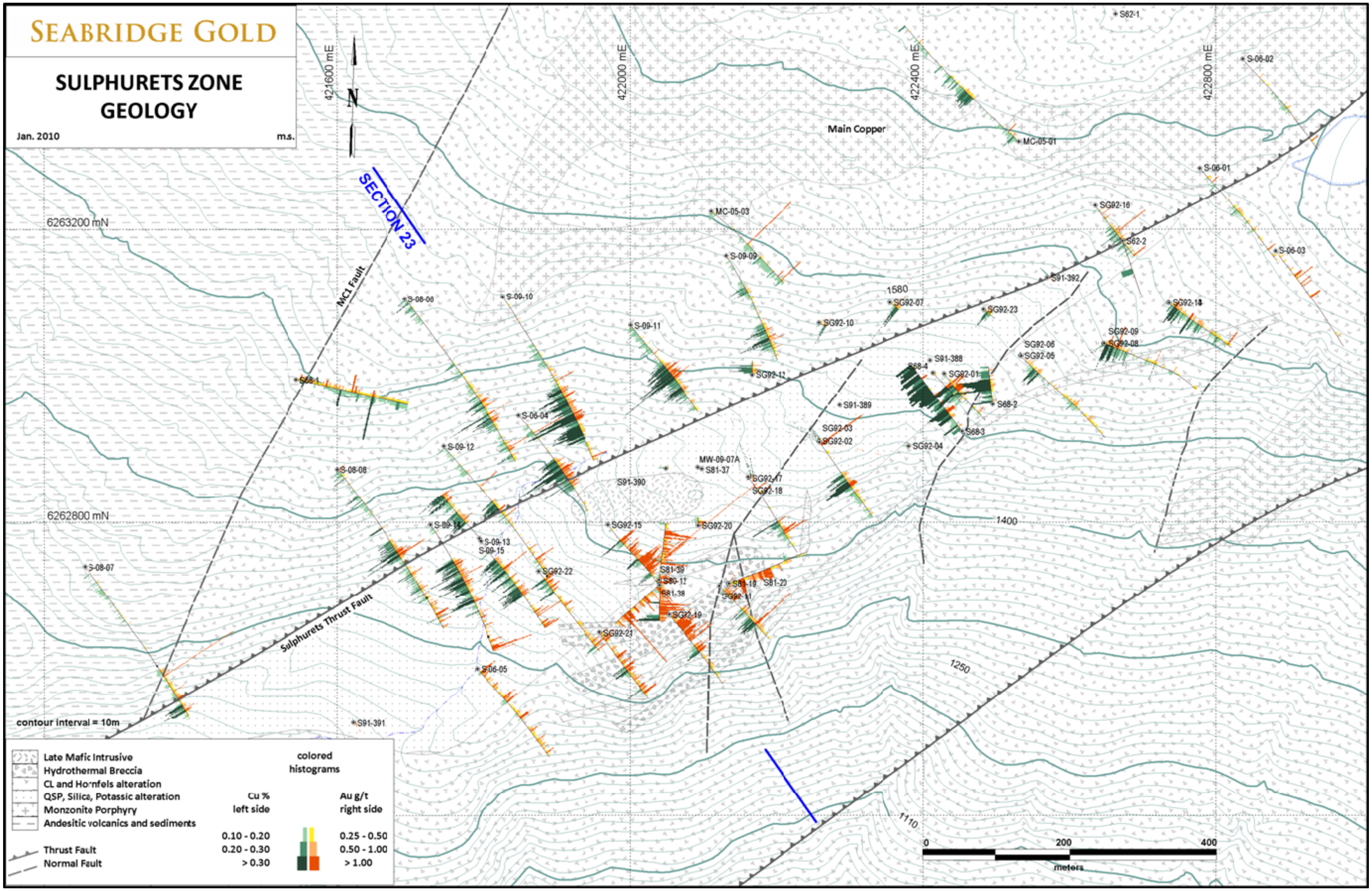


Figure 9-5: Sulphurets Geologic Section 23 (Au)

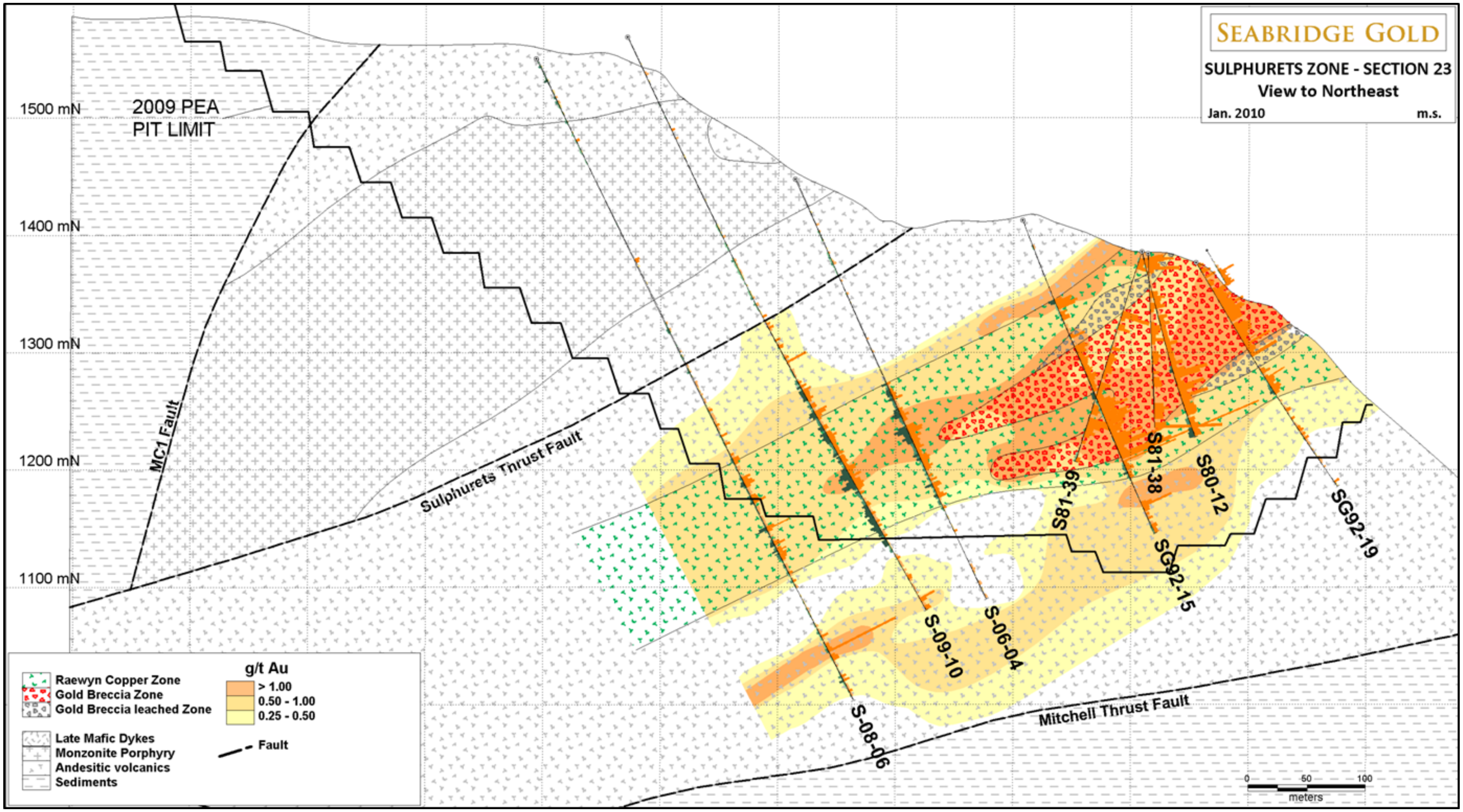
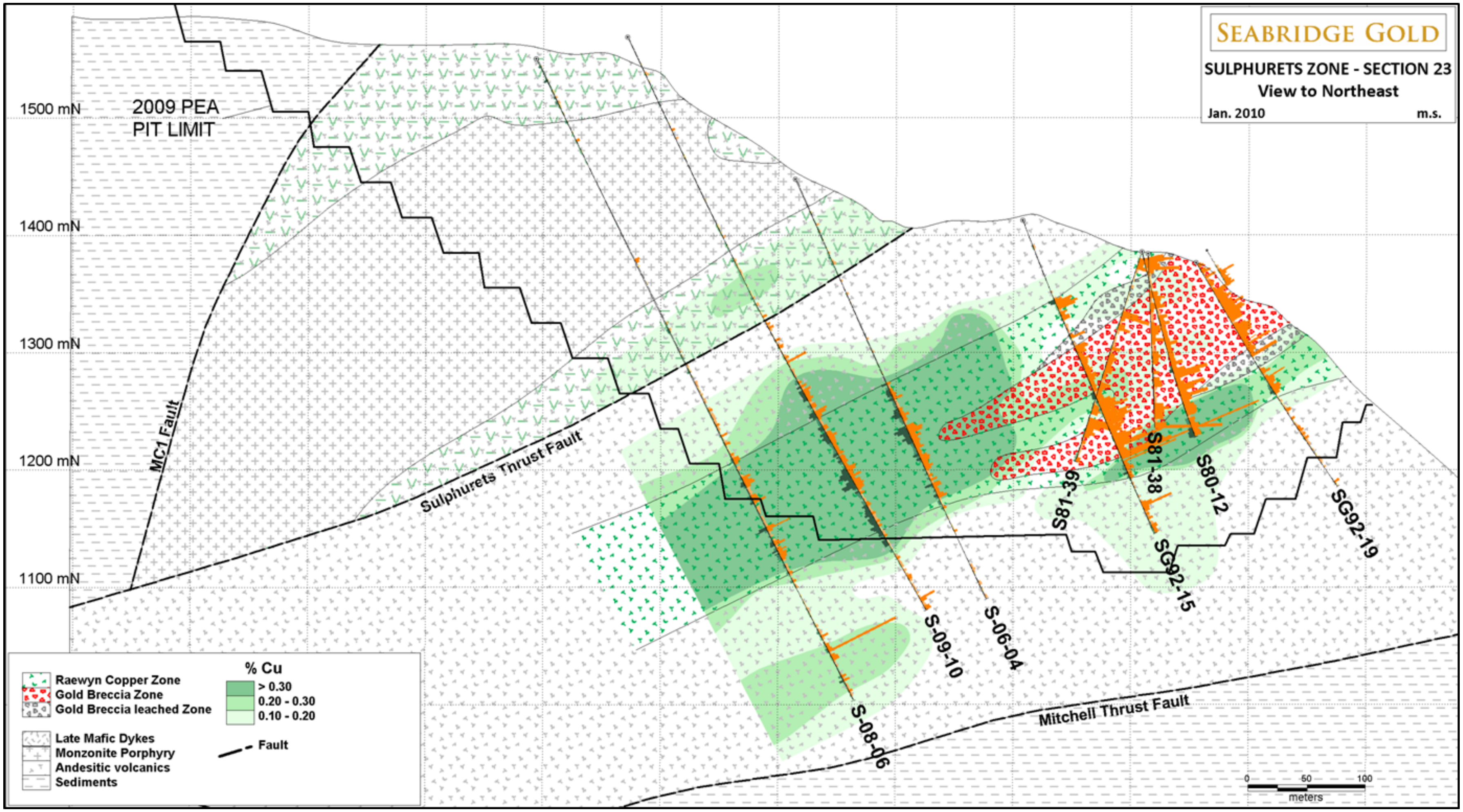


Figure 9-6: Sulphurets Geologic Section 23 (Cu)



9.3 **Mitchell Zone**

Twenty-five core holes were drilled by Seabridge in 2009 within the Mitchell zone totaling about 7,800 meters. The majority of these holes were drilled along the northern and southern flanks of the mineralized zone in order to: 1) test the north dipping extension of the system 2) upgrade Inferred material to Indicated and 3) test upper plate mineralization (material above the Mitchell thrust fault). 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 1,000 meters by 500 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 Creek 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 Mitchell Creek, 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 Creek 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. The line of section for the two cross sections is shown on the plan map (Figure 9-7) in blue.

Figure 9-7: Mitchell Geology Plan Map

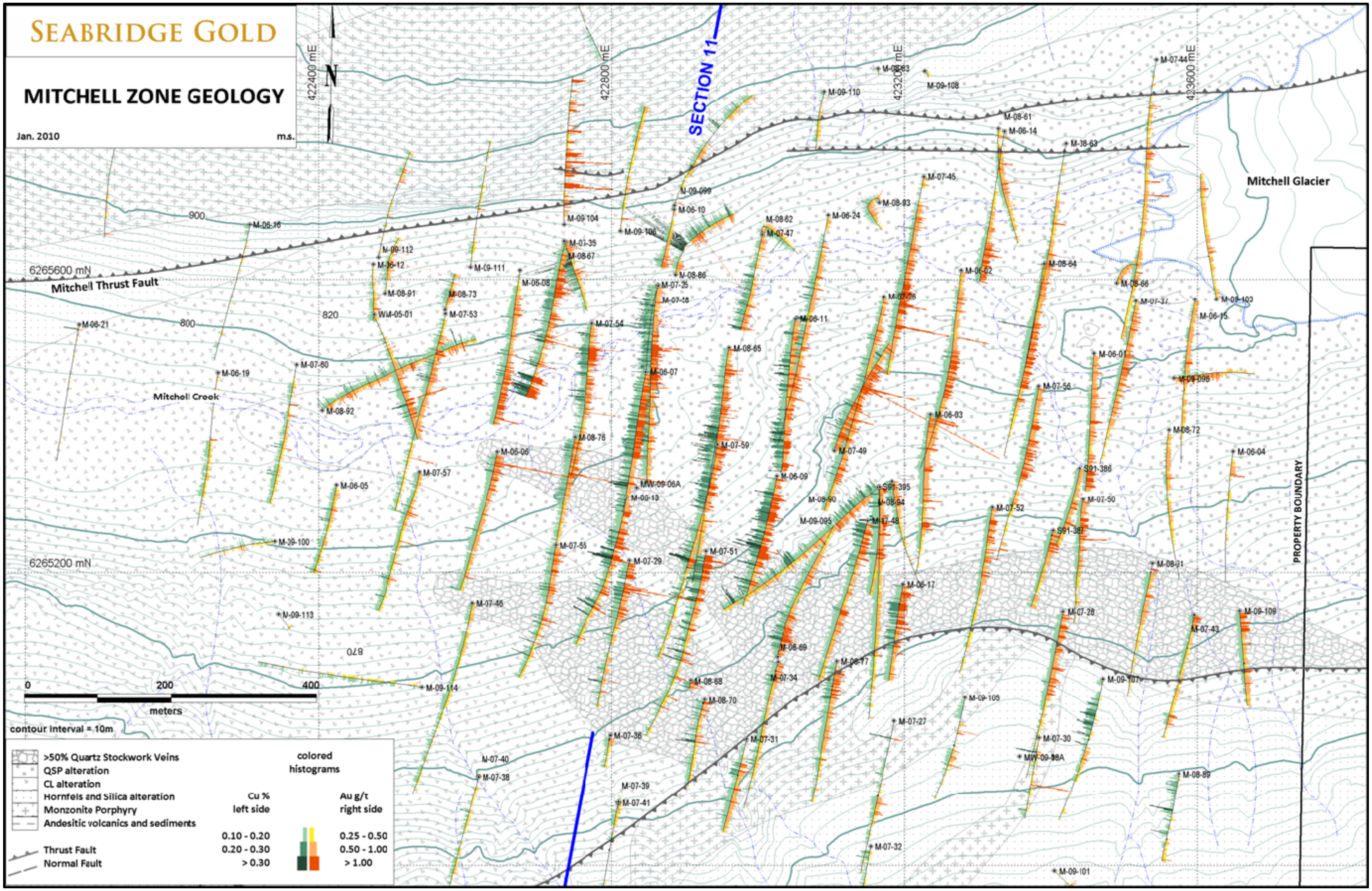


Figure 9-8: Mitchell Geologic Section 11 (Au)

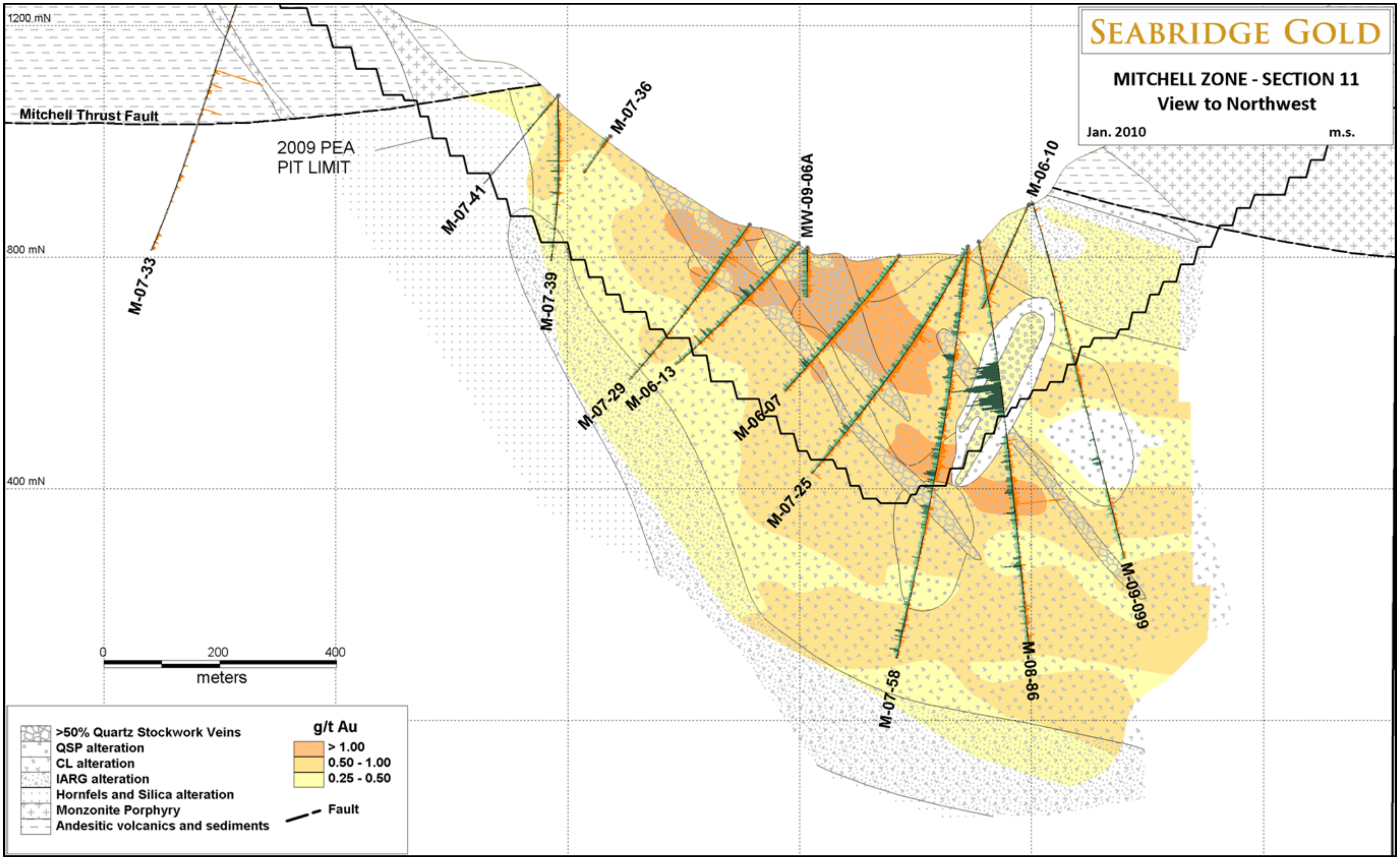
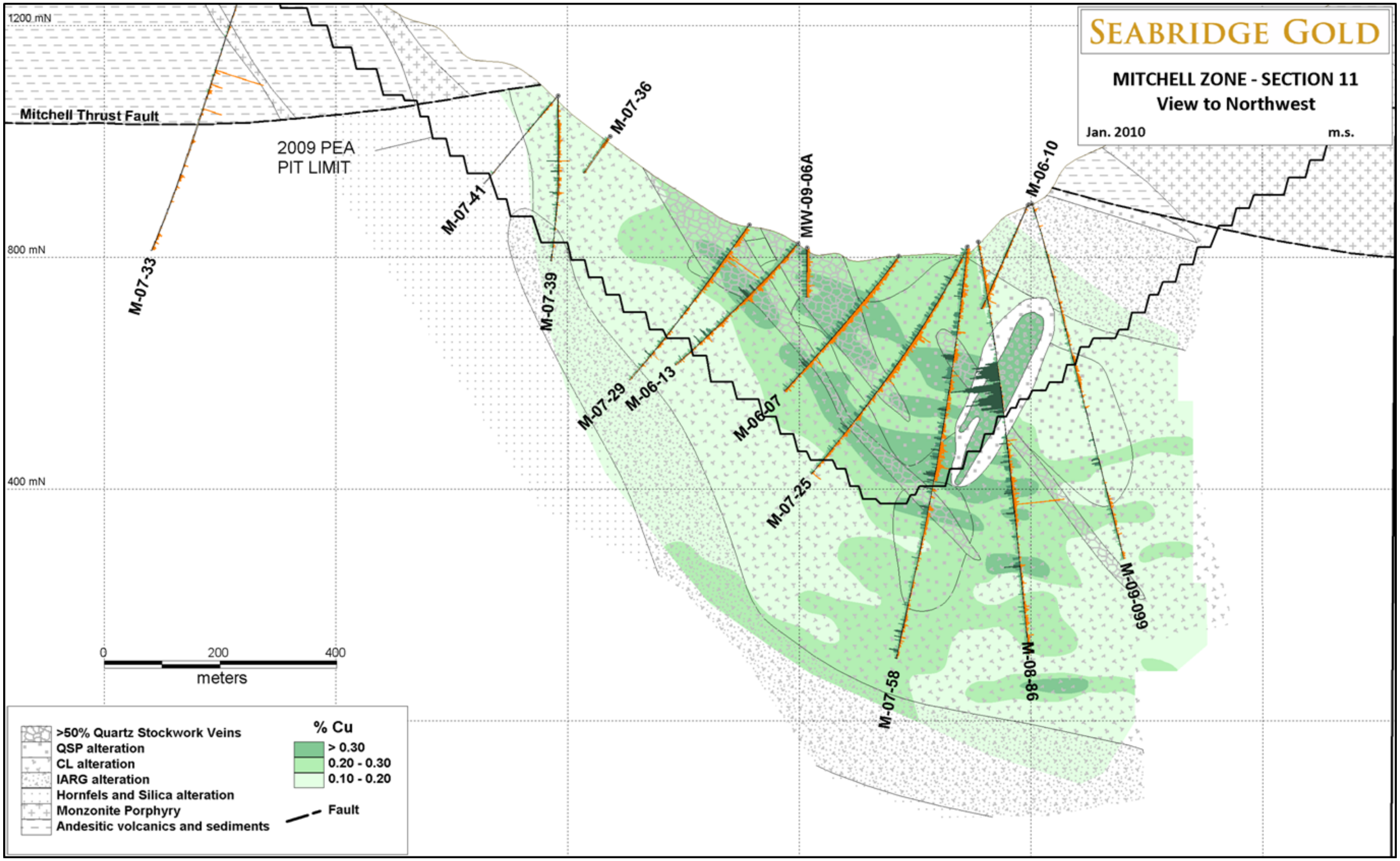


Figure 9-9: Mitchell Geologic Section 11 (Cu)



- **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 9-1 summarizes and briefly describes the alteration and lithologic characteristics associated with the various ore types for the Mitchell zone.

Table 9-1: 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).

Tables 9-2 and 9-3 summarize ore types for the Sulphurets and Kerr zones, respectively.

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

Table 9-3: 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

10.0 EXPLORATION

This section describes Seabridge's 2009 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 **2009 KSM Exploration Program**

Seabridge's 2009 exploration efforts centered primarily around infill drilling within the Mitchell, Sulphurets, and Kerr deposits in order to upgrade resource categories within current pit designs to at least an indicated level. The Kerr, Sulphurets, and Mitchell drilling programs are tabulated in Tables 11-1 through 11-3, respectively. At Mitchell, nine of the holes totaling 3,754.6 were drilled using larger diameter HQ tools to allow detailed geotechnical data collection including down-hole digital photography. In addition, 18 shallow, large diameter holes were drilled at various locations throughout the property for geotechnical purposes.

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 Labs, a commercial laboratory located in Kamloops B. C. 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 2009 Exploration Program**

The previous geologic interpretation of the Mitchell deposit was updated using the 2009 core hole data and surface mapping data. The author notes the updated geologic interpretation remains virtually unchanged from the previous interpretation (see Section 7, 8, and 9). The drilling, sampling, and assay procedures employed for the 2009 exploration program were adopted from the previous year and are discussed in Sections 11 and 12, respectively.

10.3 **Interpretation of Exploration Data**

The author combined the 2009 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 Mitchell in 2009 were either directly carried out by Seabridge's geologic staff or directly supervised by Seabridge personnel.

11.0 DRILLING

This section describes Seabridge's 2009 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, and Lechner 2009).

11.1 **2009 Drilling Campaign**

Seabridge Gold completed a helicopter supported diamond drilling program at Mitchell Creek in 2009 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 6,170 diamond core samples were collected from the 2009 Kerr, Sulphurets, and Mitchell drilling program and analyzed by Eco Tech Lab out of Kamloops, B.C. for gold, copper and a suite of other elements. In addition, 471 quality control samples (blanks, standards and duplicates) were submitted with the core samples. From these core and control samples, 545 pulps (9%) were selected and analyzed by ALS Chemex Laboratory in Vancouver, B.C. as per the QAQC protocol.

Seven holes were drilled at Kerr in 2009 totaling about 1,150 meters. These holes were primarily drilled in the northern and southern ends of the deposit to infill areas of wider spaced drilling and to confirm earlier drill results. Eleven holes were drilled within the Sulphurets zone totaling about 3,500 meters. The majority of these holes were drilled along the western flank of the west dipping mineralized zone below the Sulphurets thrust fault in order to upgrade Inferred resources to an Indicated category. Several of holes also tested the Main Copper zone, which is located immediately above the Sulphurets thrust fault. Twenty-five core holes were drilled within the Mitchell zone totaling about 7,800 meters. The majority of these holes were drilled along the northern and southern flanks of the mineralized zone to 1) test the north dipping extension of the system 2) upgrade Inferred material to Indicated and 3) test upper plate mineralization (material above the Mitchell thrust fault).

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 drill holes completed in 2009 were surveyed by Seabridge personnel using a standard uncorrected DGPS unit. Twelve of these holes were also surveyed by McGladrey & Associates, Professional Land Surveyors using control station corrected DGPS and processed using the CSRS PPP Service. Table 14-2 summarizes the differences between the two surveys. The differences are considered acceptable. In some cases, the McGladrey survey was only able to locate the anchor steel used to secure the drill, which may be on the order of 1 meter from the hole collar location.

Tables 11-1 through 11-3 summarize drill hole data that were used for estimating Mineral Resources for the Kerr, Sulphurets, and Mitchell zones, respectively by the company and year in which the holes were drilled.

Table 11-1: 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	19.3%
Newhawk Gold	1988	T88-nnn	2	115.21	0.4%
Sulphurets Gold	1989	K89-nnn, T89-nnn	20	4,365.35	15.8%
Placer Dome	1992	KS-nnn, KS92-nnn	83	16,413.57	59.5%
Seabridge	2009	K-09-nn, MW-09-nna	7	1,158.75	4.2%
Total	n/a	n/a	151	27,567.34	100.0%

Table 11-2: 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	5.4%
Esso Resources	1980, 1981	S80-nn, S81-nn	14	2,275.23	12.1%
Newhawk Gold Mines	1991	S91-nn	7	1,306.30	7.0%
Placer Dome	1992	SG92-nn	23	5,577.34	29.8%
Falconbridge	2005, 2006	MC-05-nn, MQ-05-nn, IF-05-nn	7	1,648.09	8.8%
Seabridge Gold	2006, 2008, 2009	S-06-nn, S-08-nn, S-09-nn, MW-09-nna	19	6,918.89	36.9%
Total	n/a	n/a	76	18,741.87	100.0%

Table 11-3: Mitchell Drill Hole Summary By Company

Company	Year Drilled	Hole Pre-fix	No. Holes	No. Meters	% of Total
Newhawk	1991	S91-nnn	4	647.30	1.3%
Falconbridge	2005	NM-05-nn, WM-05-nn	4	1,197.29	2.5%
Seabridge	2006 - 2009	M-06-nnn thru M-09-nnn, MW-09-nna	120	46,384.38	96.2%
Total	n/a	n/a	128	48,228.97	100.0%

Figure 11-1 is a drill hole collar map for the Kerr area showing the collar location and hole trace in red. The surface trace of 0.3% copper is shown in green and the September 2009 PEA pit is shown in blue. Figure 11-2 shows the drill hole locations/traces for the Sulphurets and Mitchell deposits. The surface trace of plus 0.25 g/t gold is shown in green and the September 2009 PEA pits shown in blue. Cross section reference lines are also shown in Figure 11-2.

Figure 11-1: Kerr Drill Hole Locations

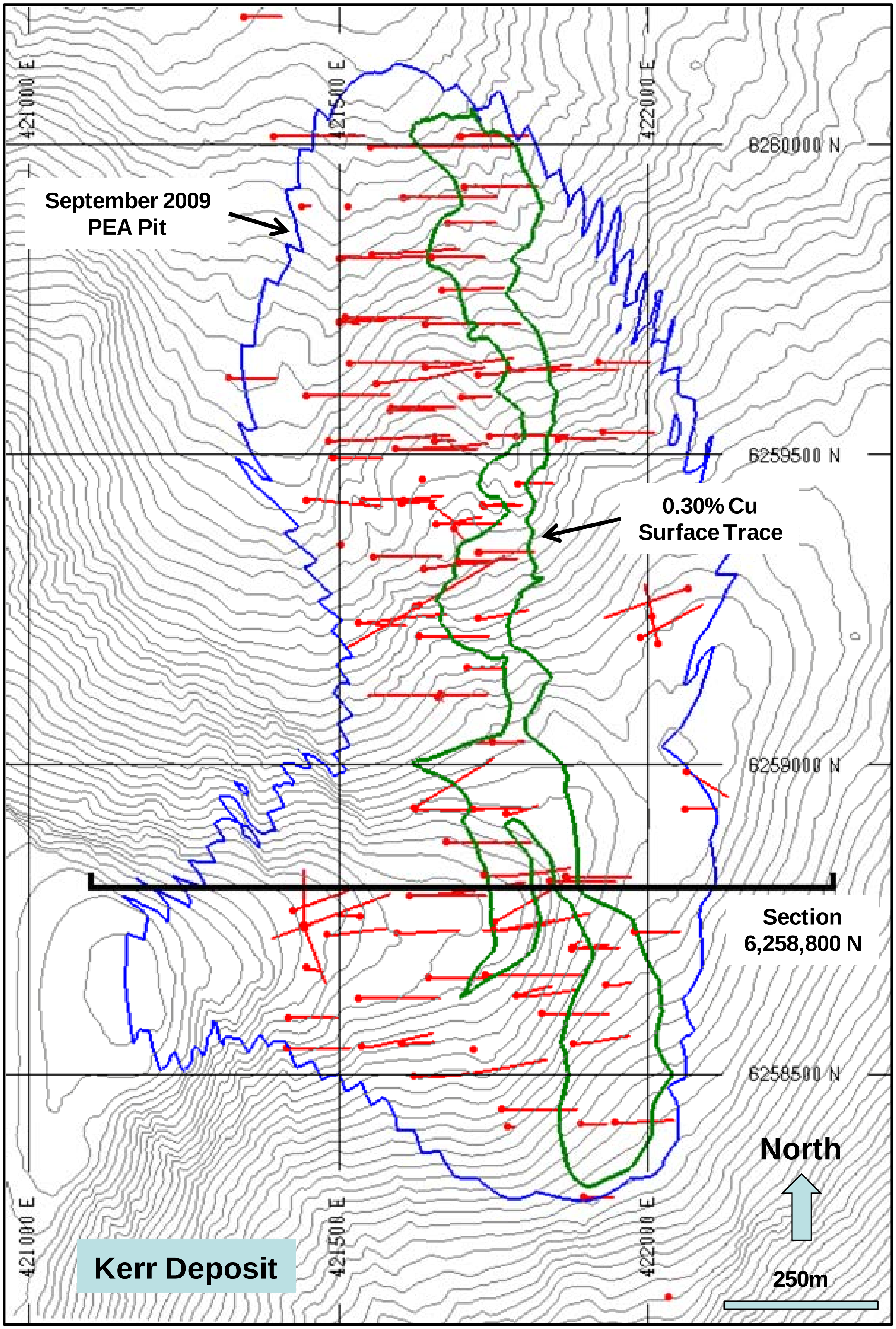
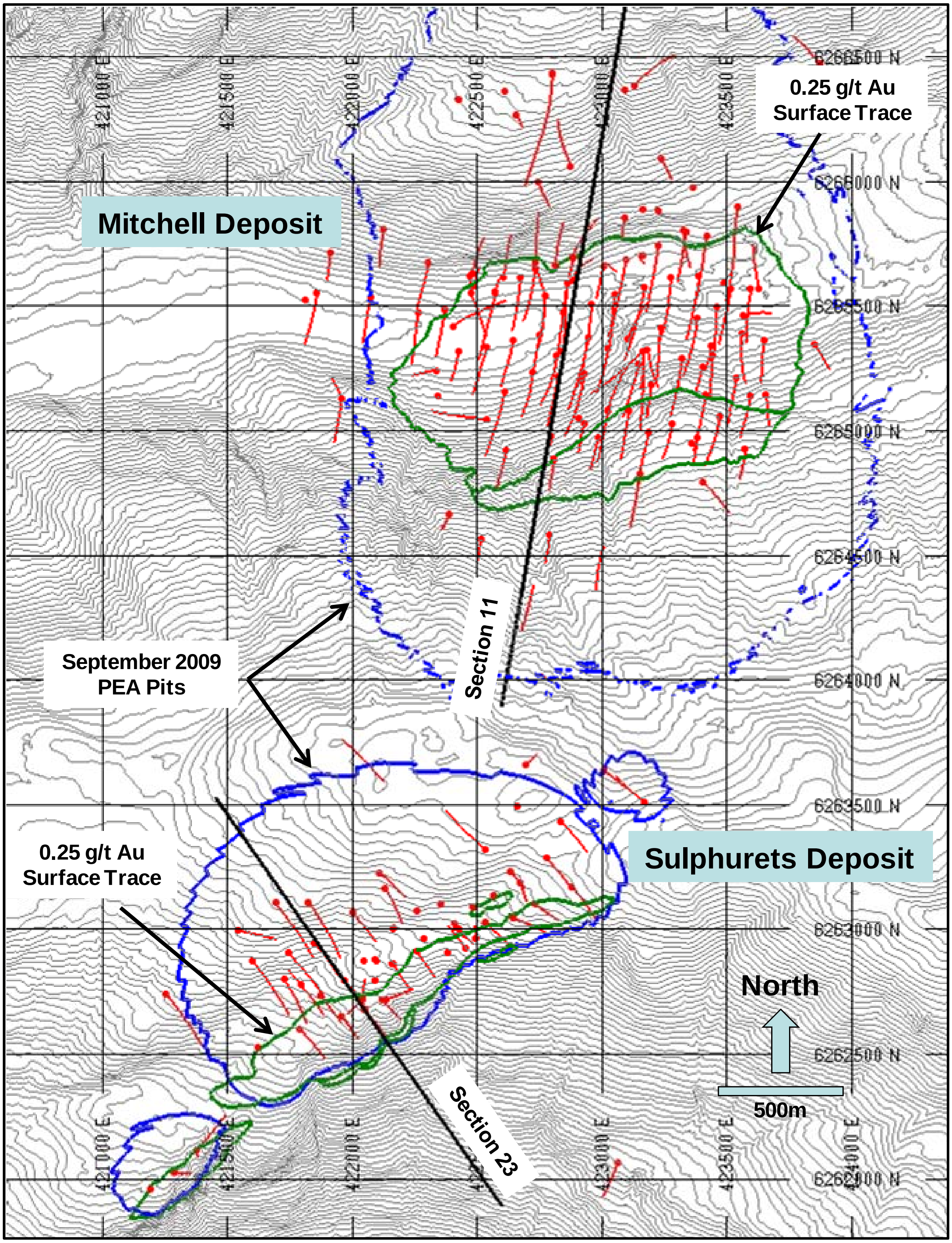


Figure 11-2: Sulphurets/Mitchell Drill Hole Locations



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 2009:

"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 Creek, 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 mineralization at Mitchell Creek 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.

Similarly 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.

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, and Lechner 2009).

12.1 Sample Length

The 2009 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 6,170 samples that were collected, 15 percent were less than 2-meters-long, 80% were exactly 2-meters-long and 5% were longer than 2-meters. In 2009, approximately 96% of drilled meterage was assayed. Approximately 85% of the 2009 drill holes used NQ drilling tools and the remainder with HQ tools. The 43 holes that were drilled in 2009 and used for resource estimation averaged about 291 meters in length. After completing the 2009 drilling campaign, the Kerr deposit has been drilled on roughly 50 to 75m centers over an area which measures about 1500m in the north-south direction and 250m in the east-west direction. The Sulphurets zone has been drilled to about 75m to 100m centers over an area measuring about 1000m (northeast-southwest) by 250m (northwest-southeast). The Mitchell zone has been drilled to roughly 75-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.

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 RQD for the 2009 drilling was about 69 percent and core recovery averaged about 95%. The frequency of natural breaks averaged about 12 per meter. Poorer recoveries and RQD's were obtained from the Kerr zone where the average core recovery was about 82% due to the highly altered and fractured nature of the material. Larger diameter drilling tools (HQ and HQ3) were used at Kerr to enhance core recovery. The average RQD for the 2009 Kerr drilling was about 53%. Core recovery for the 2009 drilling at Sulphurets and Mitchell in 2009 was 97% and 96%, 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. Sample quality was definitely poorer at Kerr where about 15% of the samples had recovery rates less than 50%. This is in contrast to Sulphurets and Mitchell where only 1% and 3% of the 2009 drilling had recoveries less than 50%, respectively. As previously mentioned, the Kerr rocks are highly altered and sheared which undoubtedly lead to the poorer recoveries.

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 12-5 show relevant composited drill hole grades for the Kerr, Sulphurets, and Mitchell 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 12-5 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 Composite Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)
88-001	176.17	228.25	52.08	0.30	1.22
88-011	51.00	163.85	112.85	0.41	1.31
88-015	130.00	196.00	66.00	0.26	0.71
88-016	3.05	106.07	103.02	0.28	0.54
88-018	3.05	75.85	72.80	0.36	0.72
88-021	162.10	213.05	50.95	0.54	1.17
88-022	2.74	55.00	52.26	0.17	0.32
K-09-01	218.17	276.00	57.83	0.22	0.59
K-09-01	277.50	344.28	66.78	0.18	0.60
K-09-02	93.00	201.00	108.00	0.25	0.41
K87-005	10.30	62.90	52.60	0.41	0.50
K89-002	20.75	101.19	80.44	0.37	0.56
K89-003	58.00	136.40	78.40	0.28	0.61
K89-004	94.00	239.88	145.88	0.20	0.48
K89-005	53.85	127.70	73.85	0.32	0.79
K89-006	57.20	114.00	56.80	0.32	0.92
K89-006	120.00	187.22	67.22	0.30	0.66
K89-007	70.30	138.10	67.80	0.37	0.91
K89-010	101.00	178.05	77.05	0.20	0.80
K89-019	105.00	156.00	51.00	0.20	0.29
K89-019	159.00	361.49	202.49	0.13	0.40
KS-066	76.20	146.00	69.80	0.37	0.87
KS-067	12.30	87.00	74.70	0.27	0.37
KS-067	135.00	185.30	50.30	0.43	0.71
KS-067	188.70	256.30	67.60	0.39	0.66
KS-071	111.00	177.00	66.00	0.39	0.87
KS-073	16.80	125.50	108.70	0.32	1.16
KS-075	23.20	149.40	126.20	0.29	0.98
KS-076	8.60	90.00	81.40	0.16	0.69
KS-077	148.90	256.00	107.10	0.18	0.49
KS-081	70.00	143.30	73.30	0.22	0.41
KS-082	27.40	87.90	60.50	0.22	0.95
KS-086	26.30	77.40	51.10	0.53	1.05
KS-087	138.74	195.95	57.21	0.59	0.97
KS-088	102.80	169.60	66.80	0.15	0.29
KS-089	12.10	70.40	58.30	0.30	0.49
KS-089	173.60	255.00	81.40	0.29	0.56
KS-091	3.00	72.60	69.60	0.56	0.81
KS-094	21.90	75.80	53.90	0.26	0.48
KS-094	298.50	382.30	83.80	0.35	0.86
KS-104	36.30	87.50	51.20	0.15	0.33
KS-105	8.15	94.49	86.34	0.19	0.35
KS-106	57.30	128.20	70.90	0.19	0.66

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)
KS-107	57.91	114.40	56.49	0.16	0.32
KS-108	64.00	134.11	70.11	0.19	0.41
KS-109	69.00	179.00	110.00	0.28	0.57
KS-111	3.05	69.00	65.95	0.20	0.45
KS-112	5.18	75.80	70.62	0.18	0.44
KS-115	159.90	215.00	55.10	0.15	0.34
KS-116	24.38	87.00	62.62	0.20	0.37
KS-116	144.00	218.00	74.00	0.14	0.39
KS-116	239.30	302.05	62.75	0.15	0.28
KS-117	3.70	78.33	74.63	0.21	0.37
KS-119	136.00	231.66	95.66	0.20	0.48
KS-120	38.40	93.57	55.17	0.24	0.60
KS-121	89.70	162.46	72.76	0.18	0.32
KS-121	165.50	218.10	52.60	0.14	0.31
KS-122	197.00	251.00	54.00	0.18	0.27
KS-123	24.00	108.81	84.81	0.24	0.45
KS-123	124.05	238.50	114.45	0.35	0.70
KS-123	241.15	299.70	58.55	0.27	0.59
KS-124	79.00	253.00	174.00	0.22	0.40
KS-124	259.00	331.00	72.00	0.56	0.71
KS-125	130.15	212.25	82.10	0.33	0.69
KS-125	262.40	324.90	62.50	0.31	0.57
KS-126	80.60	141.80	61.20	0.20	0.35
KS-127	33.50	151.80	118.30	0.21	0.49
KS-127	158.80	209.40	50.60	0.25	0.52
KS-127	212.45	268.90	56.45	0.26	0.42
KS-128	149.96	297.40	147.44	0.23	0.63
KS-130	28.04	110.64	82.60	0.18	0.34
KS-131	43.00	105.00	62.00	0.19	0.34
KS-131	141.10	192.00	50.90	0.25	0.61
KS92-135	28.96	92.05	63.09	0.40	1.15
KS92-136	95.10	160.00	64.90	0.20	0.48
KS92-138	55.78	135.67	79.89	0.38	0.79
KS92-139	3.66	54.56	50.90	0.19	0.32
KS92-141	59.00	110.30	51.30	0.30	0.72
KS92-143	115.50	186.50	71.00	0.34	0.45
T89-008	4.57	76.00	71.43	0.15	0.36
T89-008	82.00	175.00	93.00	0.21	0.43
T89-011	105.00	210.00	105.00	0.21	0.38
T89-011	213.00	319.40	106.40	0.27	0.66
T89-013	15.24	90.00	74.76	0.57	0.47
T89-014	140.00	203.00	63.00	0.28	0.38
Average	**n/a**	**n/a**	**77.03**	**0.29**	**0.68**

Table 12-4: Relevant Sulphurets Drill Hole Composite Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)
MC-05-02	126.00	180.00	54.00	0.13	0.27
MQ-05-01	172.00	222.00	50.00	0.23	0.27
MW-09-07A	125.85	182.50	56.65	0.68	0.55
S-06-04	188.00	310.00	122.00	0.80	0.52
S-08-08	274.00	344.20	70.20	0.89	0.46
S-09-10	326.00	399.55	73.55	0.73	0.52
S-09-10	400.42	494.60	94.18	0.58	0.71
S-09-11	183.00	354.00	171.00	0.73	0.55
S-09-13	75.00	140.00	65.00	0.75	0.47
S-09-14	131.40	263.50	132.10	0.76	0.48
S-09-15	107.00	204.05	97.05	0.55	0.42
S-09-15	263.00	351.00	88.00	0.65	0.48
S68-1	156.67	214.27	57.60	0.38	0.18
S80-12	35.00	166.24	131.24	1.29	0.15
S81-23	4.80	62.08	57.28	1.37	0.14
S81-24	3.00	60.40	57.40	1.02	0.09

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)
S81-39	93.00	150.00	57.00	0.98	0.05
S91-388	52.90	104.30	51.40	0.55	0.50
S91-389	71.10	166.70	95.60	0.70	0.64
S91-391	97.90	182.40	84.50	0.67	0.11
S91-398	12.10	69.00	56.90	0.35	0.45
SG92-02	84.00	166.60	82.60	1.32	0.86
SG92-04	19.00	87.00	68.00	0.51	0.36
SG92-07	232.00	291.69	59.69	0.69	0.51
SG92-10	218.00	292.60	74.60	0.57	0.41
SG92-12	131.00	261.00	130.00	0.58	0.44
SG92-13	25.00	144.82	119.82	0.57	0.30
SG92-15	116.13	190.40	74.27	1.57	0.15
SG92-19	14.00	93.80	79.80	1.69	0.05
SG92-23	159.70	224.27	64.57	0.54	0.40
Average	**n/a**	**n/a**	**87.99**	**0.74**	**0.43**

Table 12-5: Relevant Mitchell Drill Hole Composite Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)
M-06-001	5.70	306.00	300.30	0.81	0.13
M-06-002	3.00	100.00	97.00	0.70	0.12
M-06-002	102.00	426.00	324.00	0.85	0.19
M-06-003	5.00	208.00	203.00	0.85	0.16
M-06-003	210.00	310.00	100.00	0.62	0.14
M-06-005	5.60	108.00	102.40	0.54	0.15
M-06-006	6.00	221.70	215.70	0.91	0.21
M-06-007	4.40	287.90	283.50	0.98	0.29
M-06-008	34.00	346.00	312.00	0.83	0.20
M-06-009	4.00	296.00	292.00	0.98	0.31
M-06-010	53.00	198.00	145.00	0.66	0.14
M-06-011	3.70	297.00	293.30	0.85	0.27
M-06-012	185.00	265.00	80.00	0.60	0.14
M-06-013	4.85	105.10	100.25	0.94	0.27
M-06-013	107.60	248.00	140.40	0.79	0.27
M-06-014	83.00	137.00	54.00	0.64	0.09
M-06-014	269.00	453.00	184.00	0.92	0.18
M-06-015	2.90	206.00	203.10	0.63	0.10
M-06-017	17.00	80.60	63.60	1.14	0.29
M-06-017	99.00	164.60	65.60	0.89	0.27
M-06-017	166.10	223.00	56.90	0.63	0.20
M-06-024	110.00	356.80	246.80	0.66	0.20
M-07-024E	358.80	597.25	238.45	0.83	0.25
M-07-025	9.00	465.00	456.00	0.84	0.27
M-07-026	24.00	376.20	352.20	0.82	0.19
M-07-026	377.90	472.72	94.82	0.71	0.18
M-07-027	179.85	231.00	51.15	0.67	0.18
M-07-028	74.37	237.00	162.63	0.79	0.17
M-07-029	53.30	163.80	110.50	1.21	0.26
M-07-031	76.00	214.00	138.00	0.69	0.21
M-07-034	42.00	126.79	84.79	0.70	0.22
M-07-034	190.00	246.00	56.00	0.48	0.14
M-07-034	248.00	298.00	50.00	0.52	0.15
M-07-034	300.00	368.00	68.00	0.49	0.15
M-07-035	72.00	144.00	72.00	0.80	0.18
M-07-035	146.00	484.00	338.00	1.03	0.25
M-07-035	516.00	574.30	58.30	0.79	0.28
M-07-037	6.00	143.25	137.25	0.73	0.11
M-07-037	143.85	309.00	165.15	0.85	0.13
M-07-039	41.00	116.00	75.00	0.73	0.20
M-07-043	126.00	232.00	106.00	0.50	0.11
M-07-044	336.00	402.00	66.00	0.67	0.06
M-07-044	466.00	553.00	87.00	0.57	0.11
M-07-045	128.00	300.00	172.00	0.78	0.16
M-07-045	300.90	630.00	329.10	1.05	0.21
M-07-046	66.00	123.00	57.00	0.48	0.15
M-07-047	9.75	89.00	79.25	0.68	0.13
M-07-047	198.00	410.15	212.15	0.61	0.27
M-07-048	6.40	58.80	52.40	0.92	0.34
M-07-048	164.00	342.40	178.40	0.72	0.18
M-07-048	343.80	394.39	50.59	0.47	0.15
M-07-049	0.00	396.85	396.85	1.12	0.22
M-07-050	3.05	123.45	120.40	0.96	0.17
M-07-050	151.25	237.00	85.75	0.62	0.13
M-07-051	28.50	146.25	117.75	0.92	0.30
M-07-051	147.40	259.70	112.30	0.66	0.23
M-07-052	13.70	96.05	82.35	0.82	0.16
M-07-052	98.30	210.31	112.01	0.74	0.17
M-07-053	124.00	442.00	318.00	0.75	0.17
M-07-054	3.50	446.00	442.50	0.92	0.27
M-07-054	470.00	544.28	74.28	0.55	0.18
M-07-054	600.00	670.45	70.45	0.58	0.19
M-07-055	6.10	101.57	95.47	0.94	0.22
M-07-055	121.60	177.65	56.05	0.88	0.26
M-07-056	4.57	257.50	252.93	0.88	0.16
M-07-057	61.20	171.00	109.80	0.59	0.17
M-07-057	173.00	250.75	77.75	0.51	0.13
M-07-057	257.20	308.00	50.80	0.46	0.15
M-07-058	4.50	146.00	141.50	0.81	0.26
M-07-058	176.00	288.00	112.00	0.80	0.32
M-07-058	297.65	432.00	134.35	0.99	0.35
M-07-058	448.00	565.00	117.00	0.63	0.24
M-07-058	567.00	720.00	153.00	0.49	0.22
M-07-059	2.50	152.25	149.75	0.94	0.29
M-07-059	155.60	285.00	129.40	0.61	0.25
M-07-060	175.00	263.00	88.00	0.48	0.14
M-07-060	285.00	338.00	53.00	0.46	0.17
M-08-061	273.00	599.30	326.30	0.71	0.13
M-08-061	640.00	690.00	50.00	0.44	0.11
M-08-062	32.00	100.75	68.75	0.65	0.14
M-08-062	313.00	571.13	258.13	0.54	0.22
M-08-062	572.02	745.00	172.98	0.55	0.20
M-08-063	290.00	569.00	279.00	0.66	0.13
M-08-064	23.00	345.00	322.00	0.84	0.17
M-08-065	4.00	380.10	376.10	0.96	0.29
M-08-065	384.55	482.00	97.45	0.54	0.19
M-08-065	488.00	592.60	104.60	0.59	0.19
M-08-066	94.00	435.00	341.00	0.66	0.11
M-08-067	78.00	372.00	294.00	0.64	0.26
M-08-067	384.00	469.00	85.00	0.48	0.62
M-08-067	471.00	714.00	243.00	0.69	0.30
M-08-069	1.20	79.00	77.80	0.85	0.24
M-08-069	81.00	586.00	505.00	0.78	0.22
M-08-070	172.00	225.00	53.00	0.68	0.24
M-08-070	251.00	320.00	69.00	0.39	0.17
M-08-071	99.00	154.00	55.00	0.66	0.13
M-08-072	80.50	139.00	58.50	0.63	0.11
M-08-073	91.00	374.00	283.00	0.78	0.20
M-08-073	390.00	442.00	52.00	0.47	0.16
M-08-076	9.58	238.78	229.20	0.98	0.25
M-08-076	320.09	408.00	87.91	0.49	0.14
M-08-077	15.00	133.40	118.40	0.91	0.26
M-08-077	135.05	271.00	135.95	0.65	0.19
M-08-079	313.00	399.00	86.00	0.19	0.45
M-08-086	43.74	120.00	76.26	0.71	0.17
M-08-086	200.00	306.00	106.00	0.22	1.22
M-08-086	336.00	544.14	208.14	0.89	0.28
M-08-086	545.12	718.00	172.88	0.60	0.23
M-08-090	2.50	169.90	167.40	0.96	0.28
M-08-090	173.24	597.00	423.76	0.55	0.23
M-08-091	124.00	408.00	284.00	0.67	0.16
M-08-092	53.00	314.00	261.00	0.68	0.18
M-08-092	316.00	418.00	102.00	0.73	0.20
M-08-093	119.00	645.00	526.00	0.65	0.19
M-08-094	2.50	195.00	192.50	0.80	0.23
M-08-094	197.00	339.00	142.00	0.56	0.17
M-09-095	110.53	205.65	95.12	0.84	0.22
M-09-096	3.50	191.00	187.50	0.80	0.09
M-09-099	177.00	257.00	80.00	0.52	0.13
M-09-099	259.00	337.50	78.50	0.66	0.15
M-09-099	510.50	599.50	89.00	0.52	0.18
M-09-106	222.00	274.00	52.00	0.50	0.13
M-09-107	74.00	148.00	74.00	0.22	0.31
M-09-107	175.00	233.00	58.00	0.53	0.13
M-09-108	223.00	273.00	50.00	0.41	0.10
M-09-109	2.10	201.00	198.90	0.71	0.15
MW-09-06A	3.95	87.40	83.45	1.03	0.30
S91-386	0.00	153.70	153.70	0.73	0.18
S91-387	0.00	60.30	60.30	0.91	0.19
S91-387	61.40	123.90	62.50	0.51	0.14
S91-395	0.00	114.10	114.10	0.74	0.28
S91-395	116.50	190.50	74.00	0.60	0.21
WM-05-01	81.50	282.89	201.39	0.80	0.19
Average	**n/a**	**n/a**	**156.50**	**0.82**	**0.20**

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section describes Seabridge's sample security, sample preparation, and analytical methods that were used in 2009 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).

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
Ag	0.2 ppm	0.0 ppm
Al	0.01%	10.00%
As	5 ppm	10,000 ppm
Ba	5 ppm	10,000 ppm
Bi	5 ppm	10,000 ppm
Ca	0.01%	10.00%
Cd	1 ppm	10,000 ppm
Co	1 ppm	10,000 ppm
Cr	1 ppm	10,000 ppm
Cu	1 ppm	10,000 ppm
Fe	0.01%	10.00%
La	10 ppm	10,000 ppm
Mg	0.01%	10.00%
Mn	1 ppm	10,000 ppm

Element	Lower	Upper
Mo	1 ppm	10,000 ppm
Na	0.01%	10.00%
Ni	1 ppm	10,000 ppm
P	10 ppm	10,000 ppm
Pb	2 ppm	10,000 ppm
Sb	5 ppm	10,000 ppm
Sn	20 ppm	10,000 ppm
Sr	1 ppm	10,000 ppm
Ti	0.01%	10.00%
U	10 ppm	10,000 ppm
V	1 ppm	10,000 ppm
Y	1 ppm	10,000 ppm
Zn	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.

Blanks were prepared from barren river gravels collected near Stewart and screened to remove fines and oversize to produce nominal 1" diameter pieces which were submitted into the 2009 sample stream at a frequency of about 1 blank for every 33 samples. Approximately 186 barren samples or "blanks" were submitted to Eco Tech, which were obtained from stream gravel collected near Stewart B.C. Figures 13-1 and 13-2 chart the performance of the gold and copper blanks for the 2009 drilling campaign.

Figure 13-1: 2009 Au Blank Performance

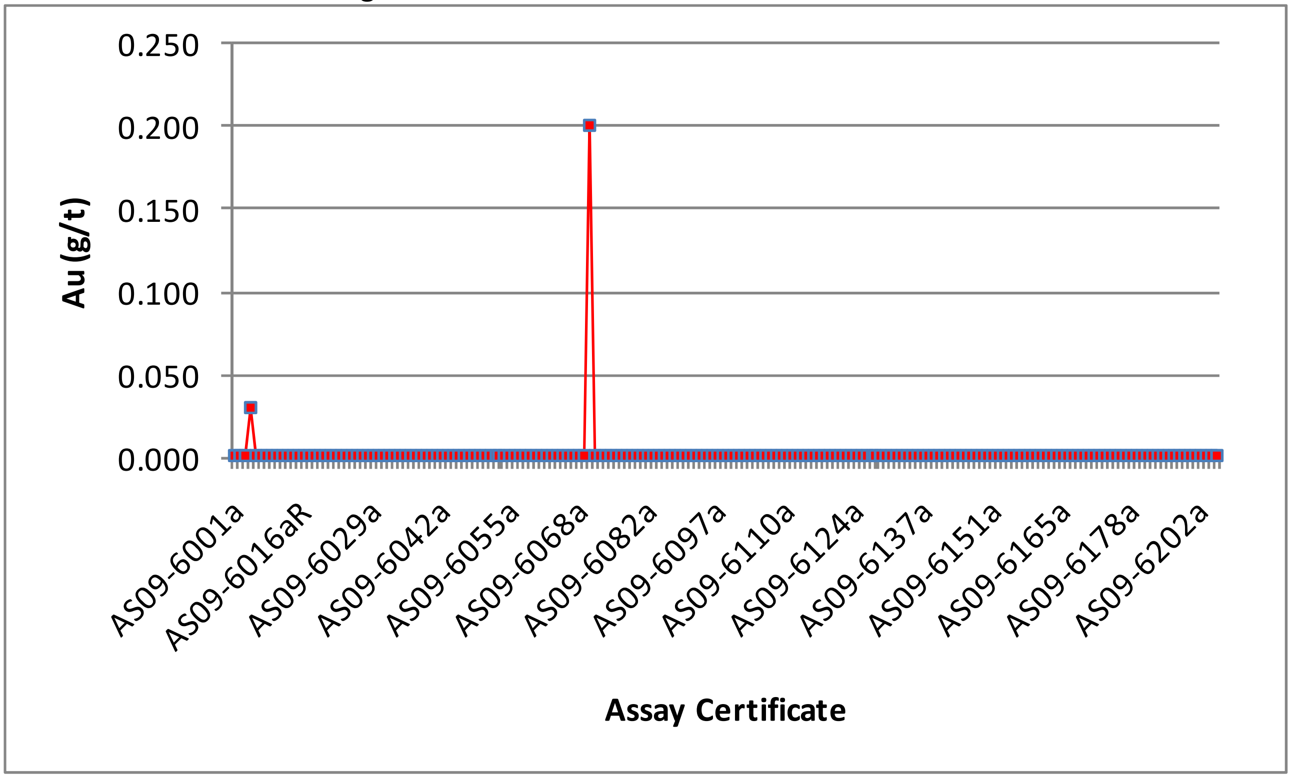
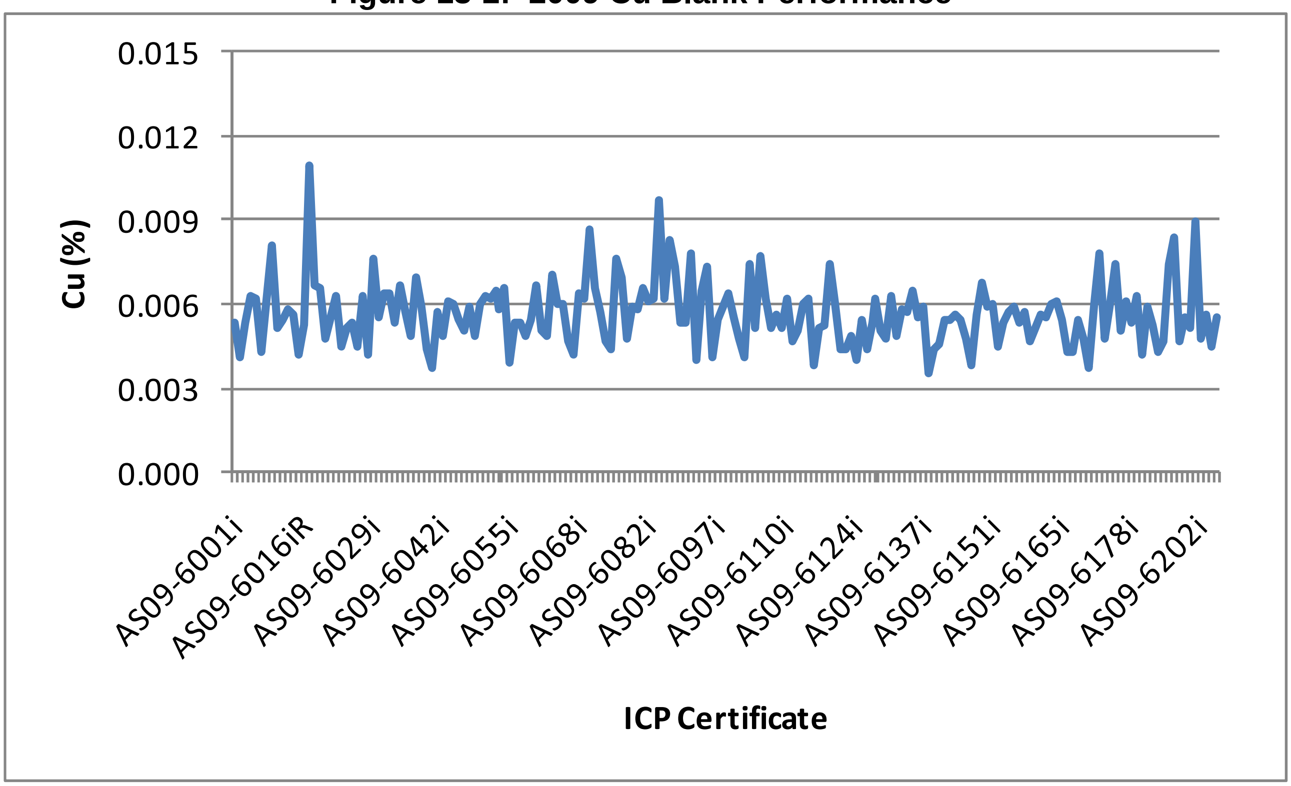


Figure 13-2: 2009 Cu Blank Performance



The pulp standards that were used by Seabridge for their 2009 drilling/sampling campaign were purchased from CDN Resource Laboratories Ltd. (CDN) out of Delta, B.C. Six CDN standards (CDN-CGS-13, CDN-CGS-18, CDN-CGS-19, CDN-CM-1, CDN-CM-4, and CDN-CM-5) were prepared from material that was collected from the Casino copper-gold-molybdenum porphyry property located in Yukon Territory. 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 190 SRM's were inserted into the 2009 sample stream or a frequency of about one SRM for every 32 samples or 3% of the total assay samples. Table 13-2 summarizes the SRM's that were used by Seabridge for their 2009 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: 2009 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-13	24	1.01	0.90	1.12	0.329	0.311	0.347	n/a	n/a	n/a
CGS-18	40	0.30	0.26	0.34	0.319	0.304	0.334	n/a	n/a	n/a
CGS-19	35	0.74	0.67	0.81	0.132	0.122	0.142	n/a	n/a	n/a
CM-1	34	1.85	1.69	2.01	0.853	0.833	0.873	0.076	0.068	0.084
CM-4	20	1.18	1.06	1.30	0.508	0.483	0.533	0.032	0.028	0.036
CM-5	37	0.29	0.25	0.34	0.319	0.299	0.339	0.050	0.045	0.055
Total	190	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 13-17.

Figure 13-3: 2009 Au Standard GCS-13 Performance

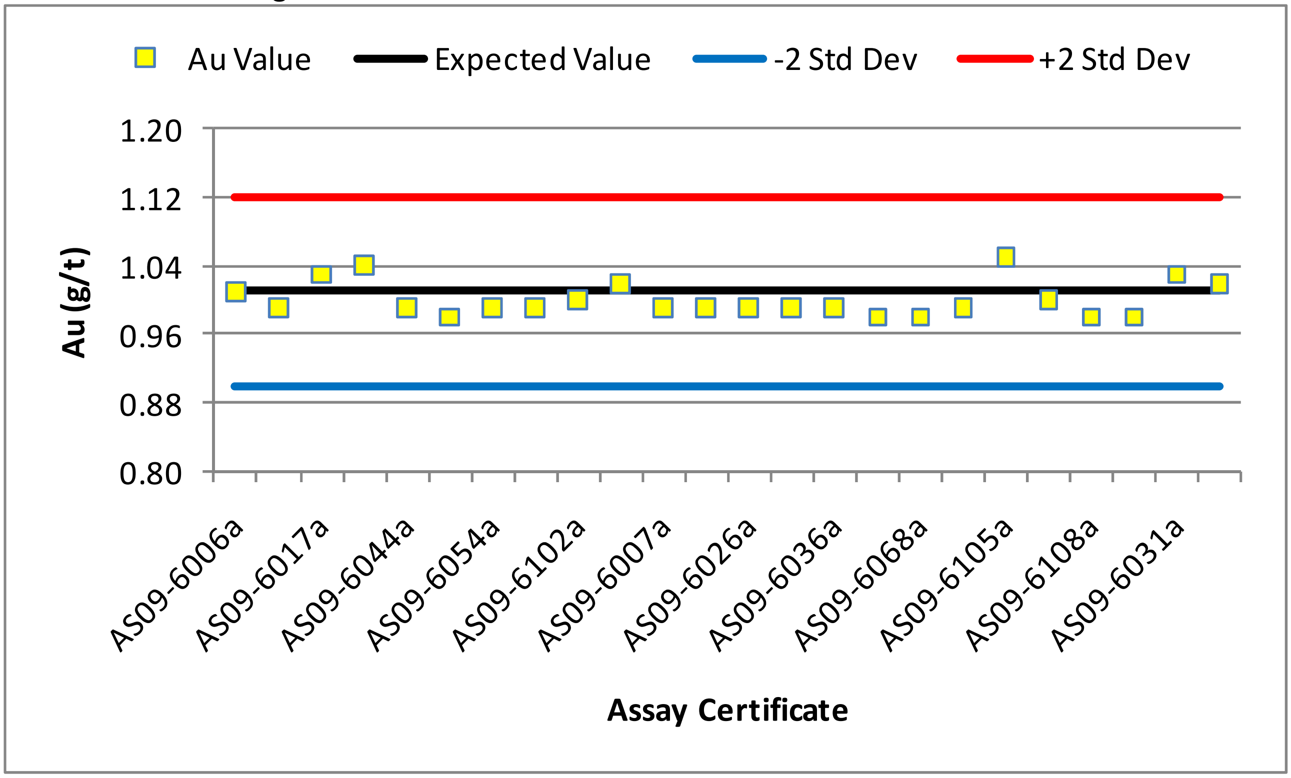


Figure 13-4: 2009 Cu Standard GCS-13 Performance

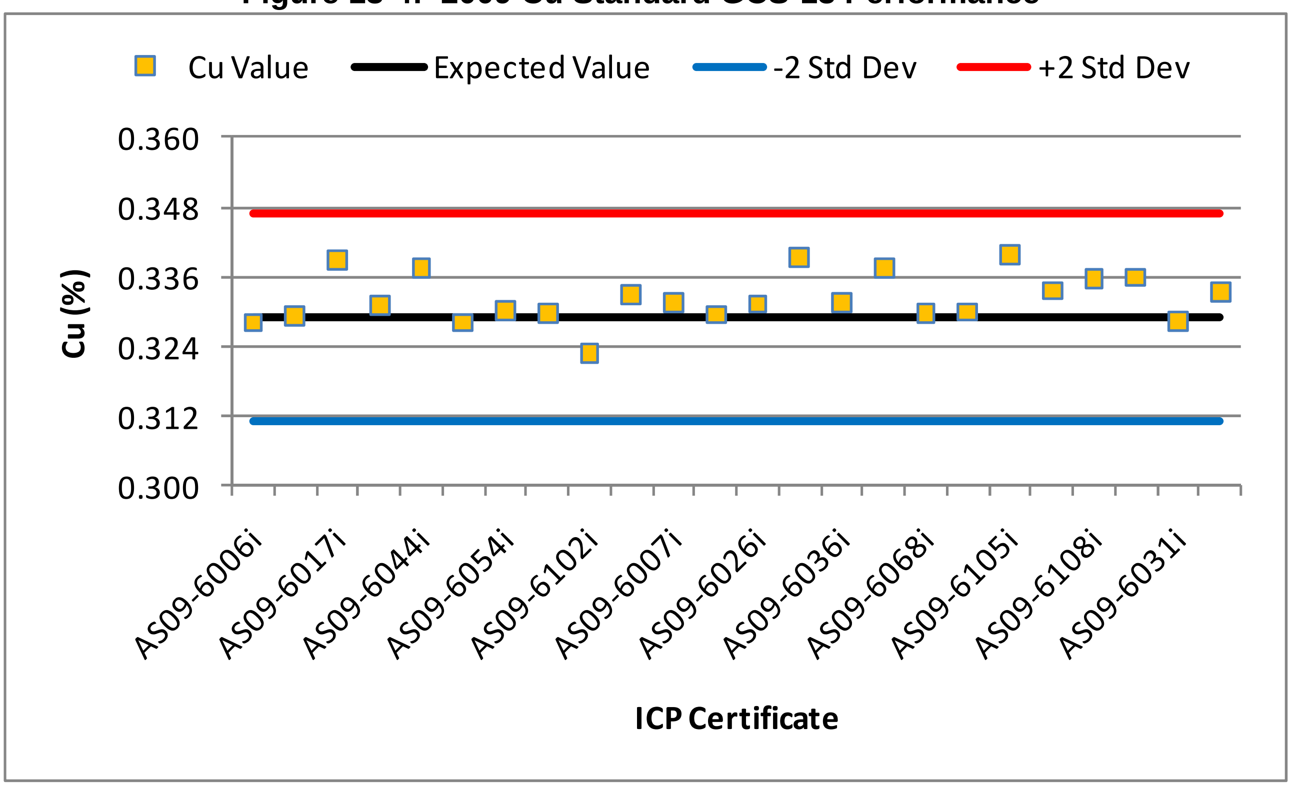


Figure 13-5: 2009 Au Standard GCS-18 Performance

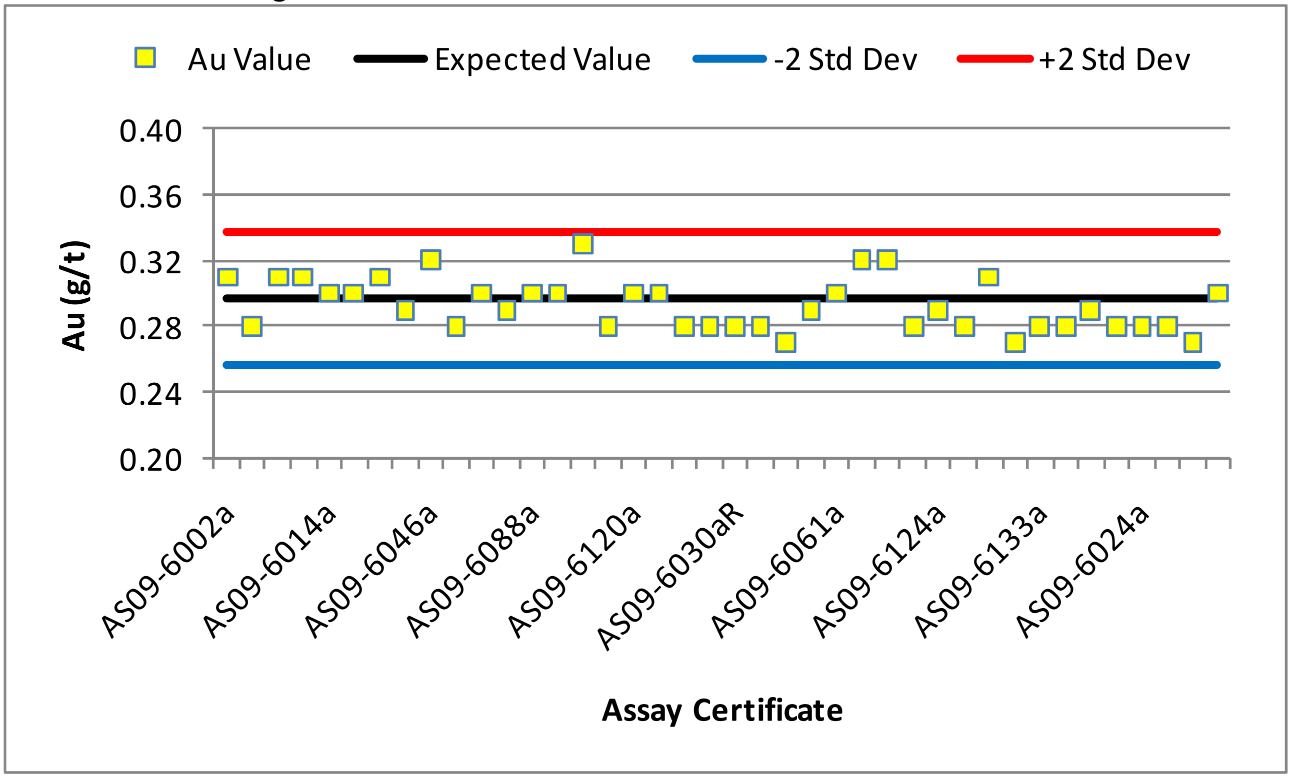


Figure 13-6: 2009 Cu Standard GCS-18 Performance

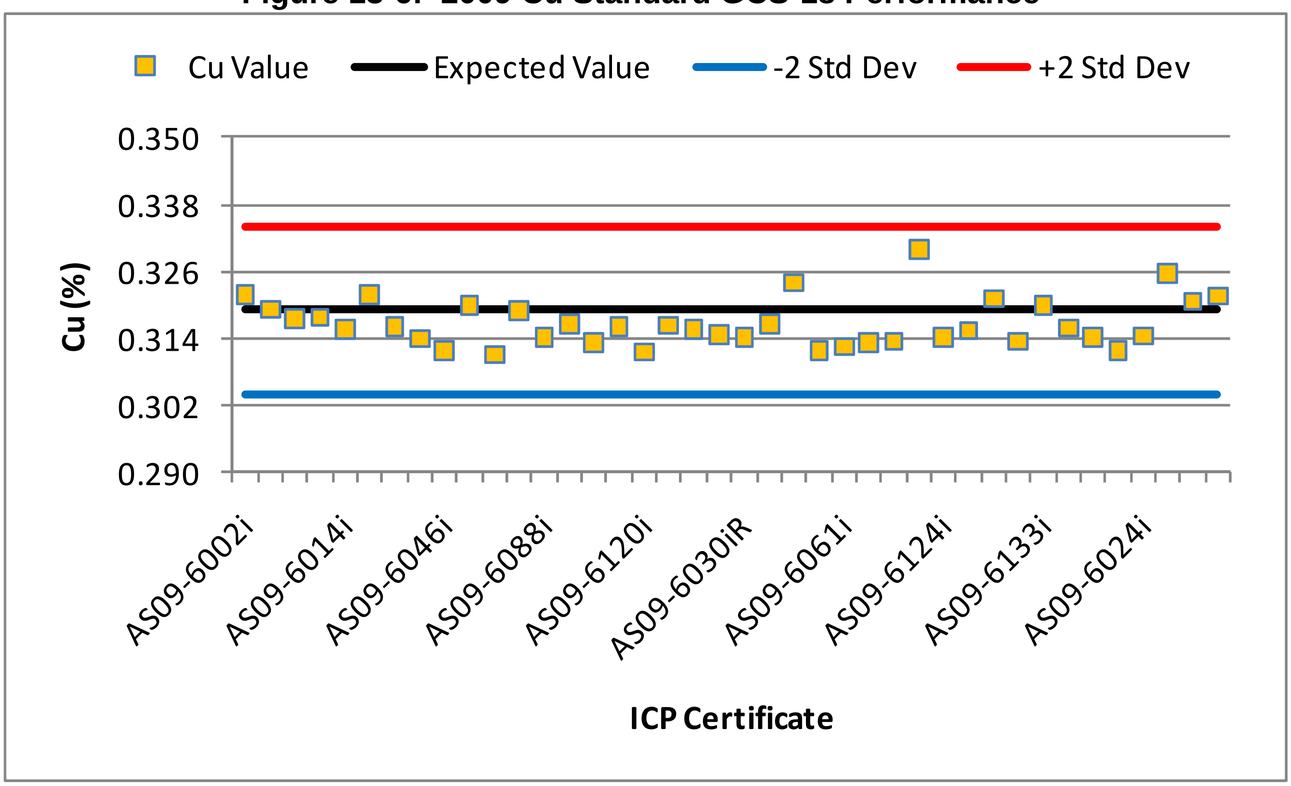


Figure 13-7: 2009 Au Standard GCS-19 Performance

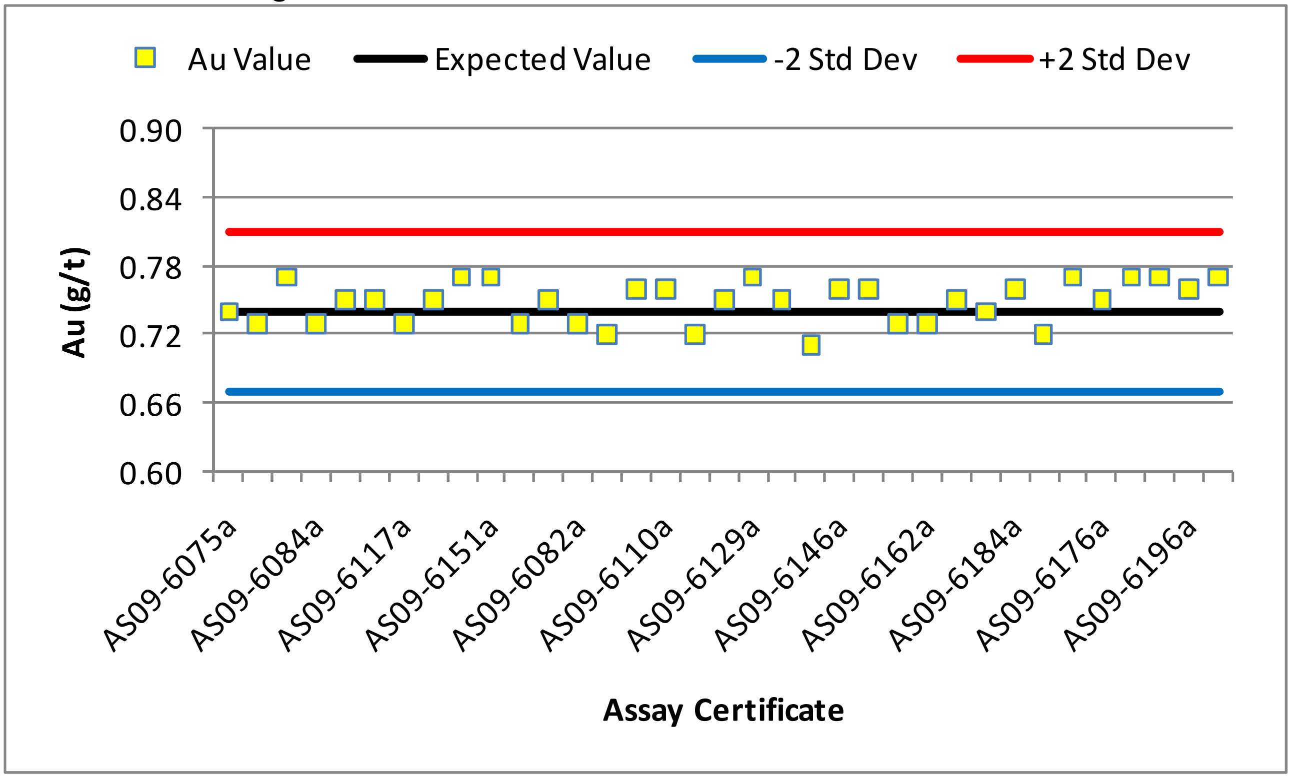


Figure 13-8: 2009 Cu Standard GCS-19 Performance

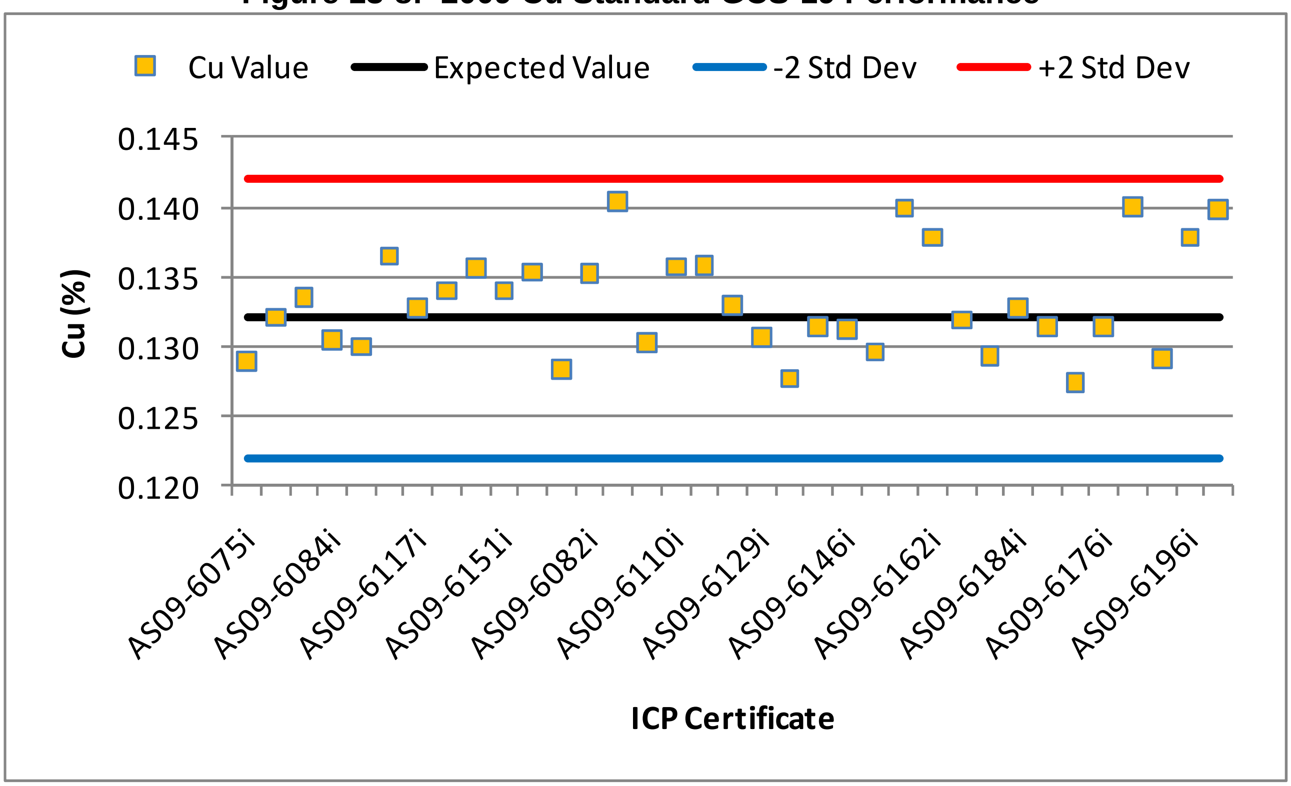


Figure 13-9: 2009 Au Standard CM-1 Performance

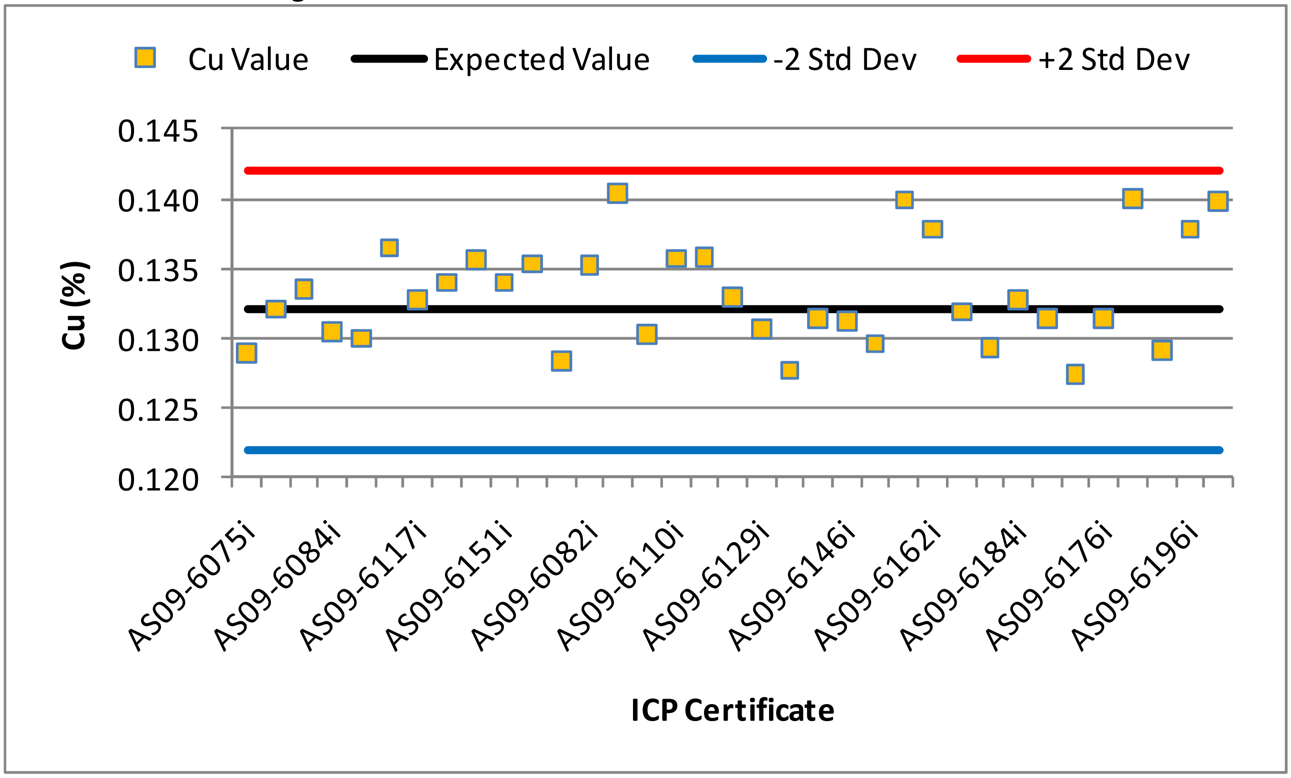


Figure 13-10: 2009 Cu Standard CM-1 Performance

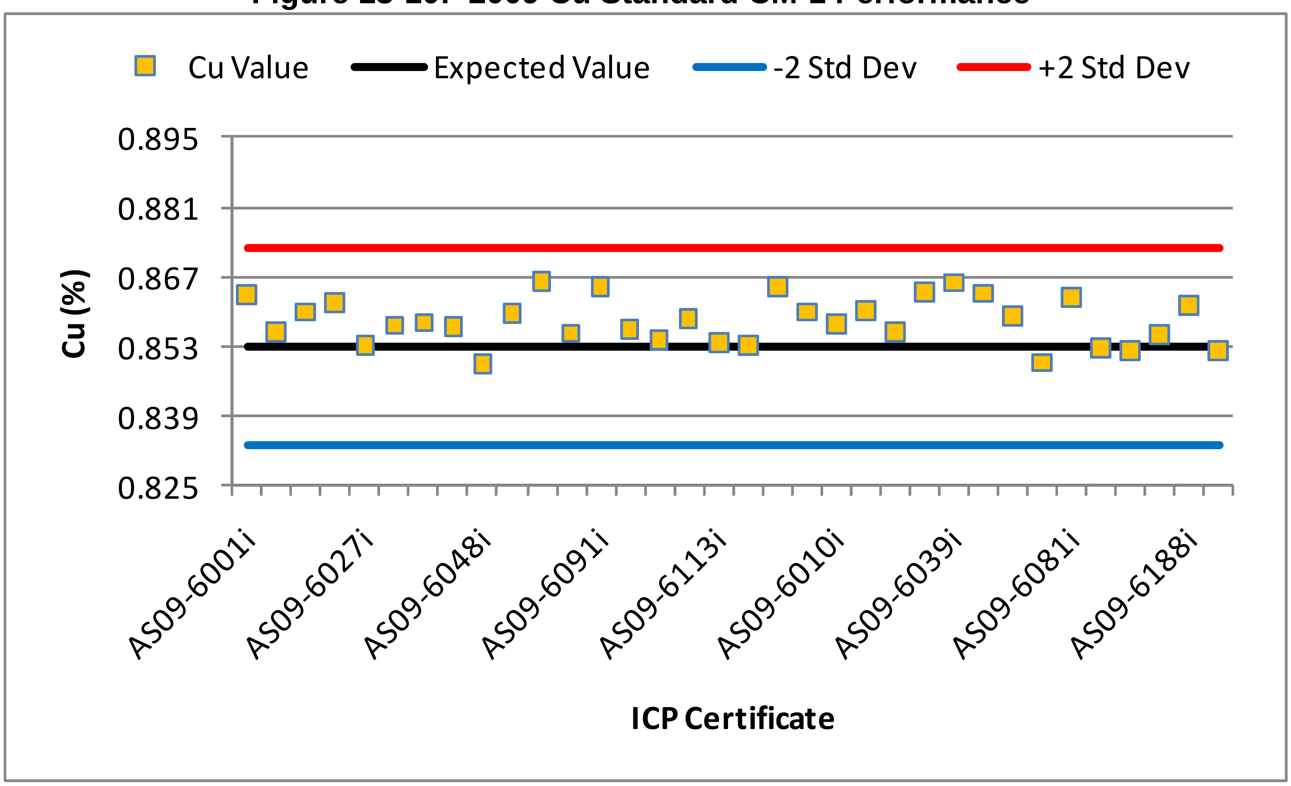


Figure 13-11: 2009 Mo Standard CM-1 Performance

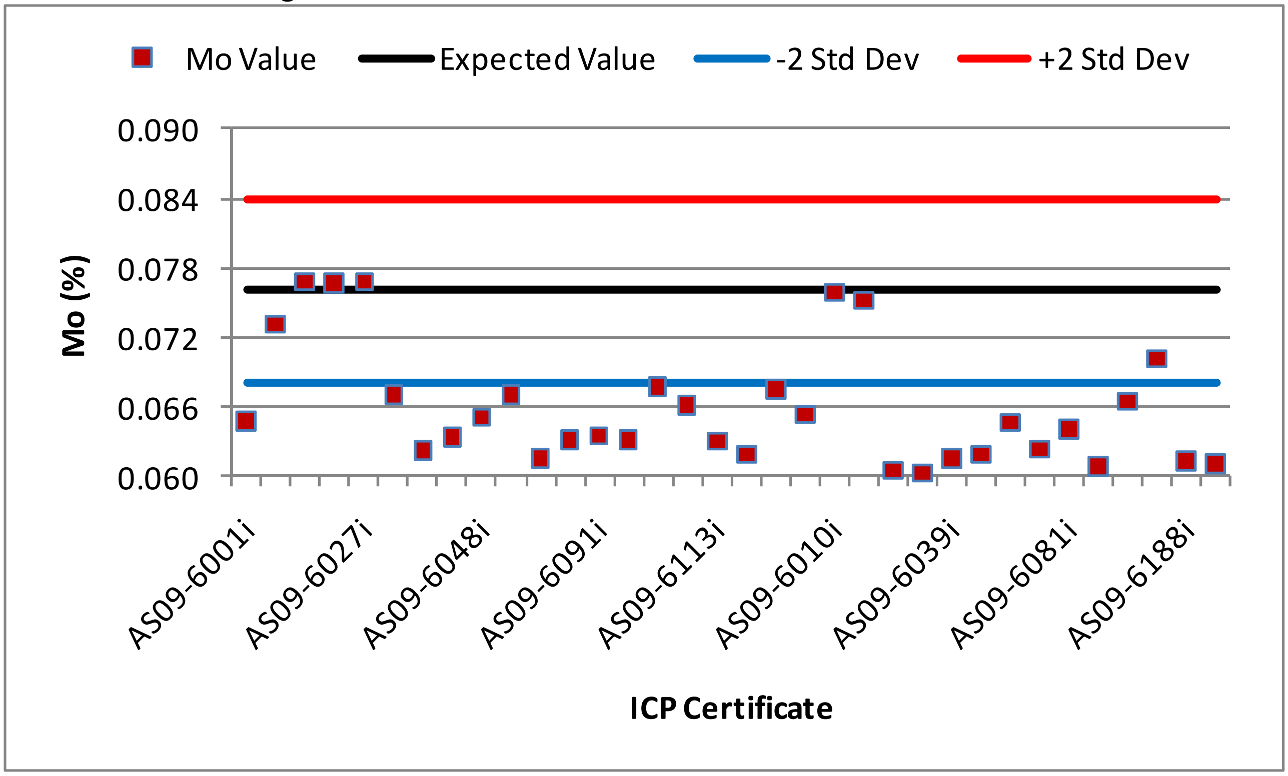


Figure 13-12: 2009 Au Standard CM-4 Performance

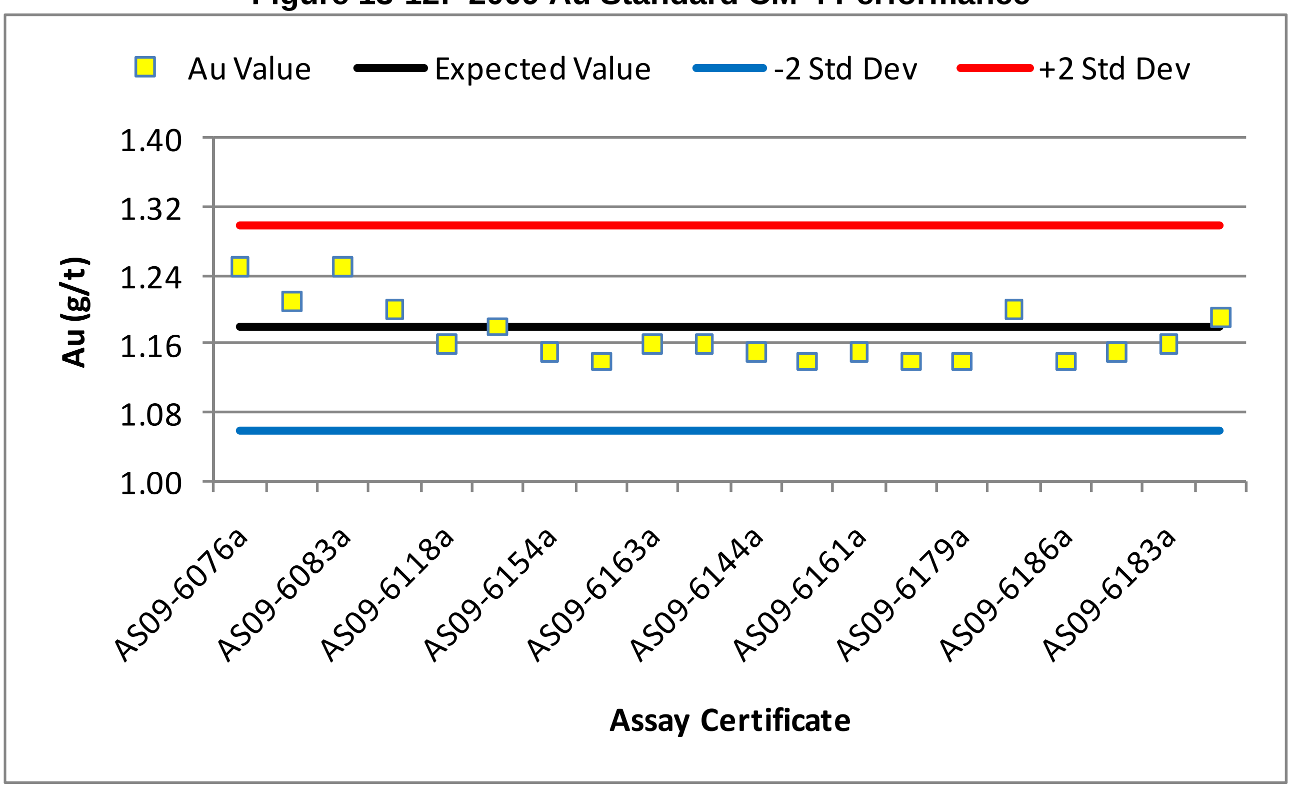


Figure 13-13: 2009 Cu Standard CM-4 Performance

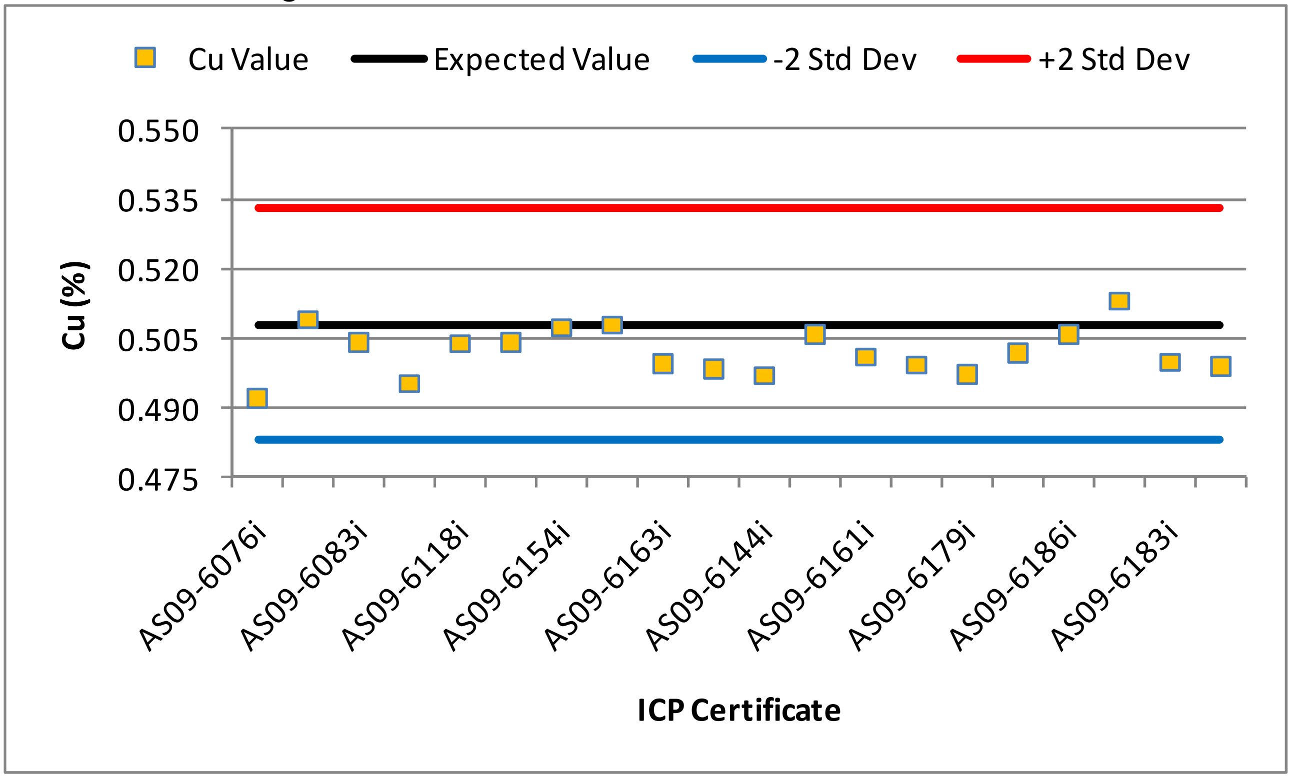


Figure 13-14: 2009 Mo Standard CM-4 Performance

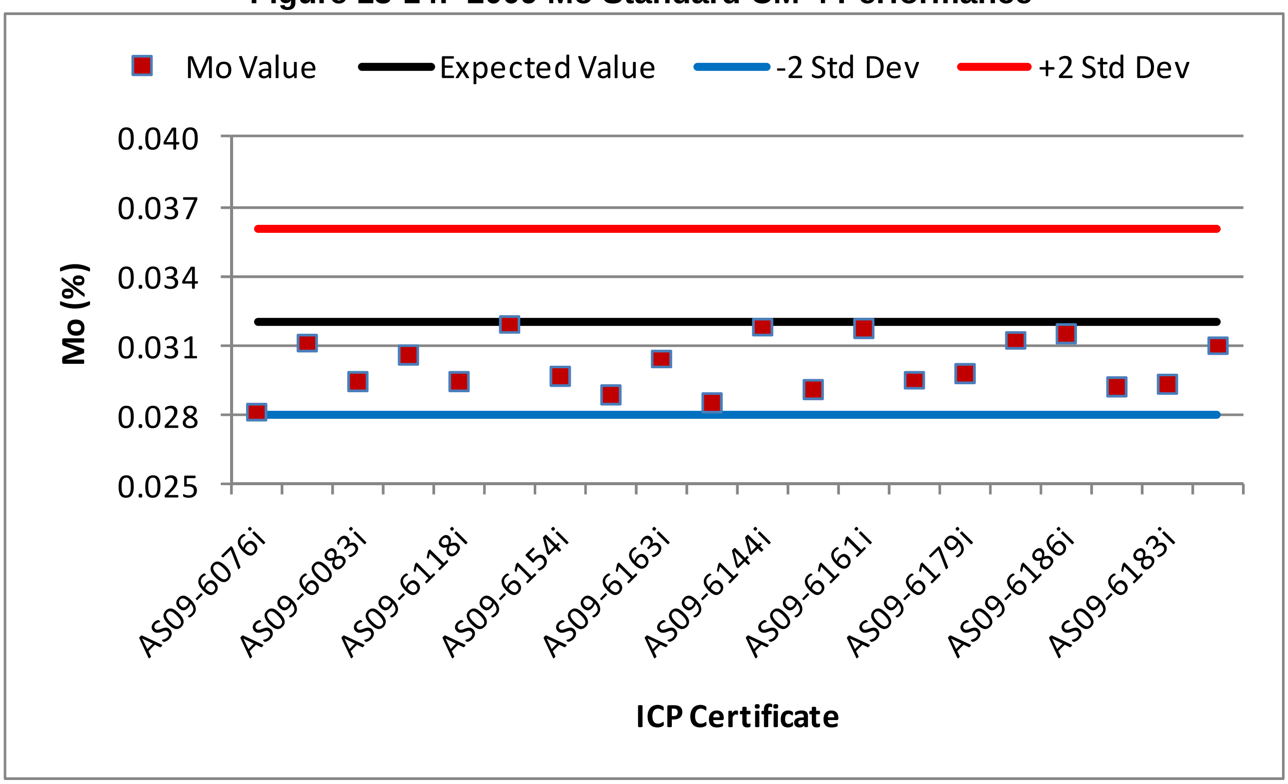


Figure 13-15: 2009 Au Standard CM-5 Performance

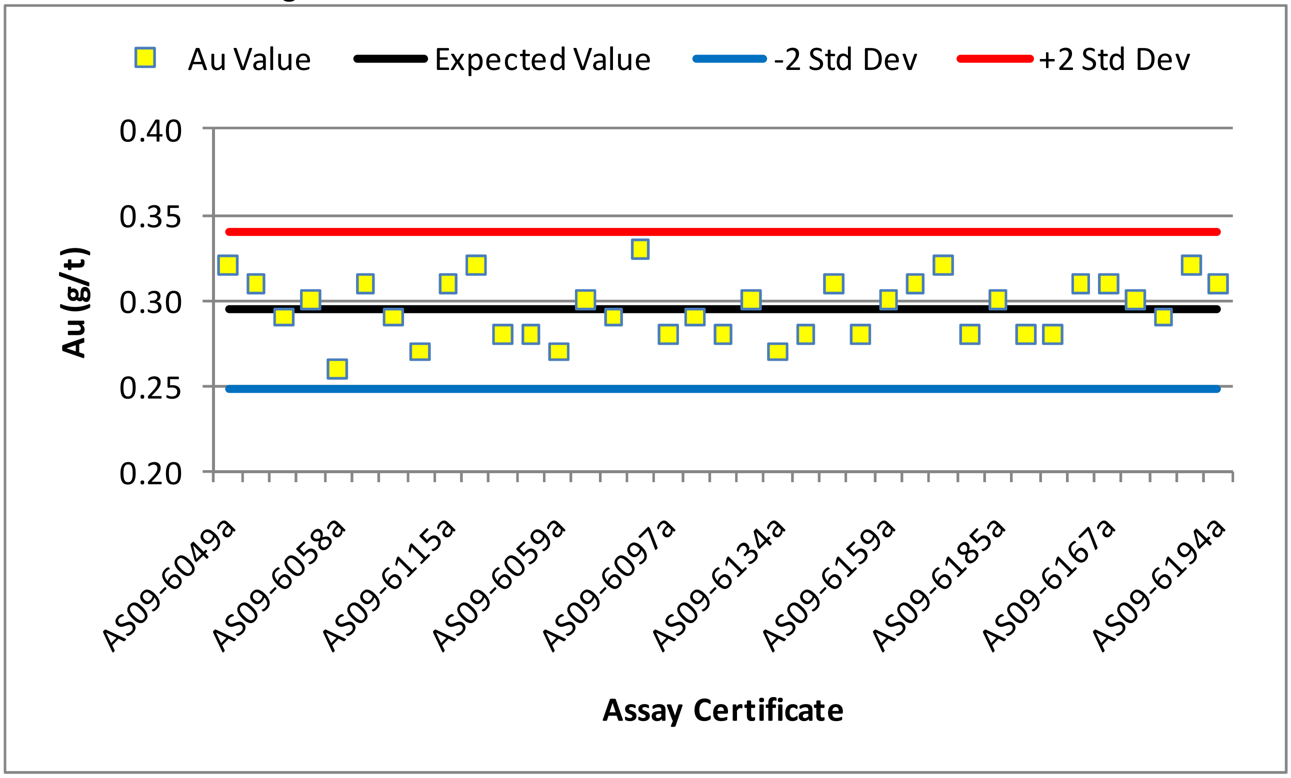


Figure 13-16: 2009 Cu Standard CM-5 Performance

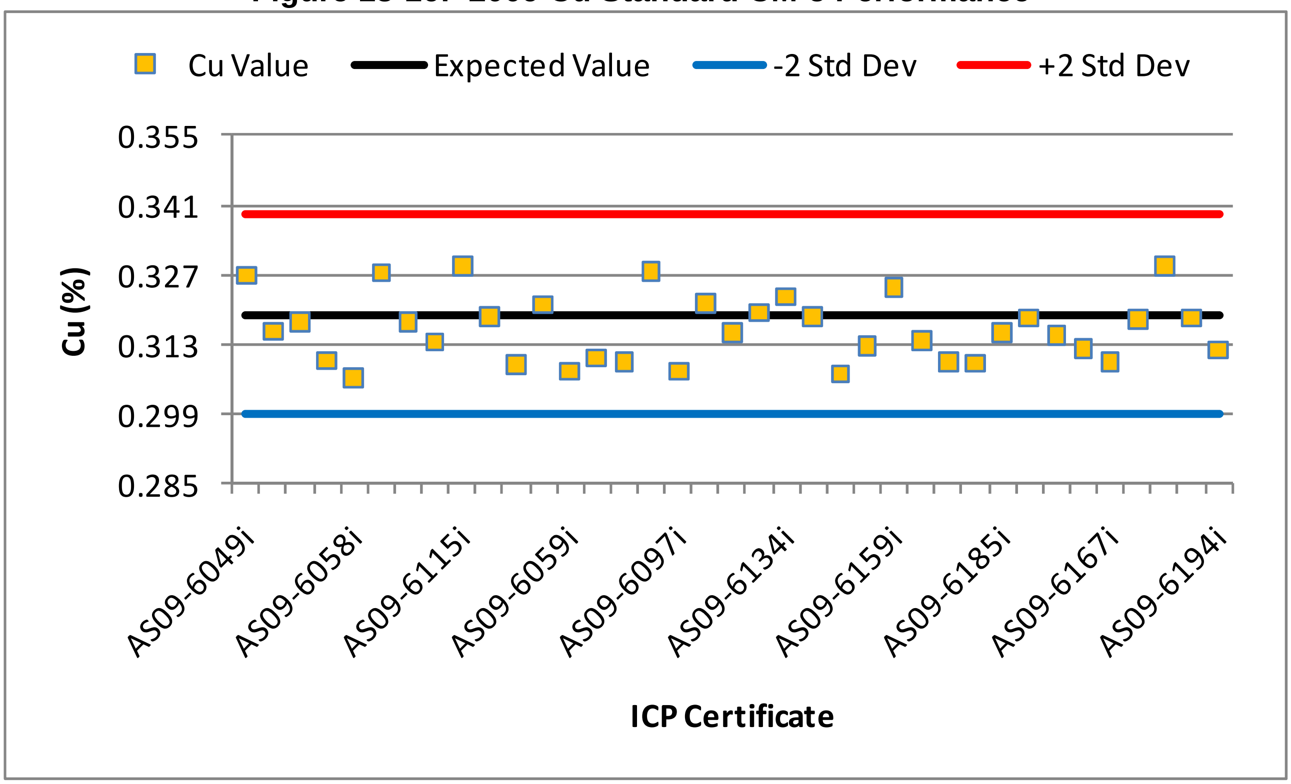
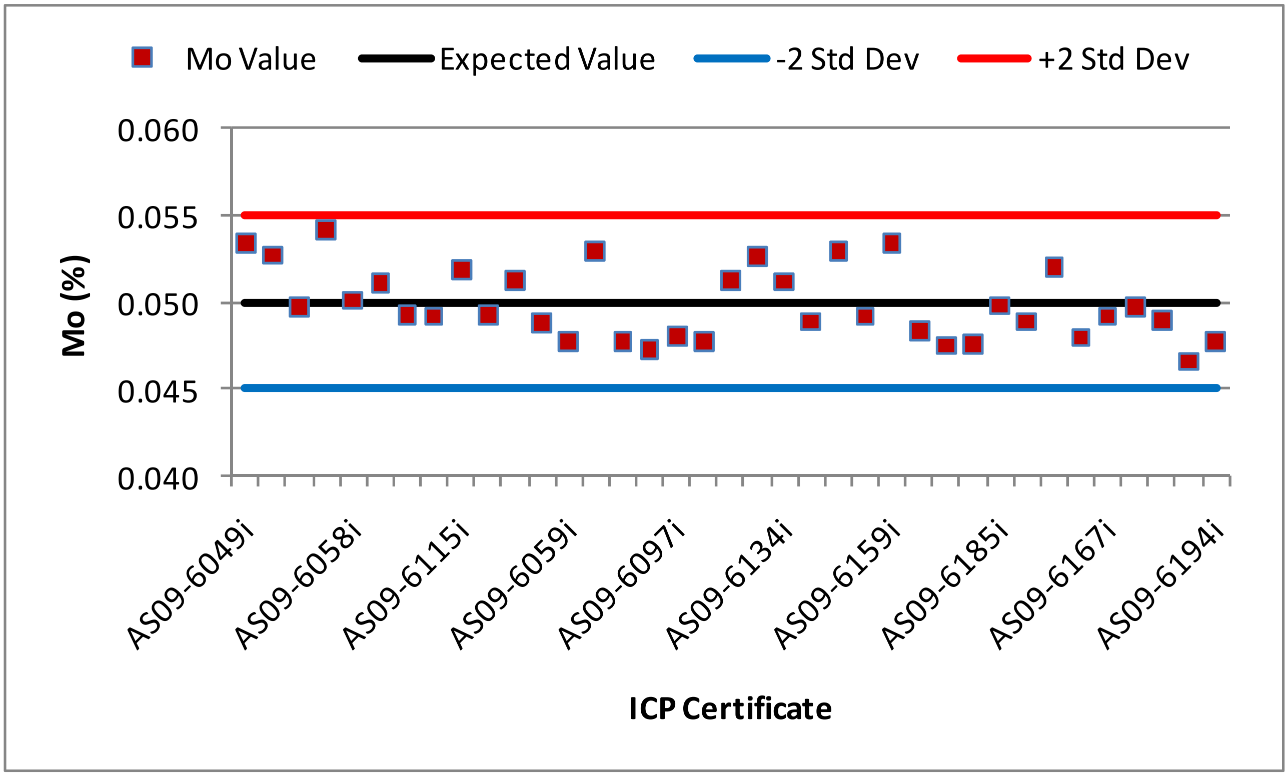


Figure 13-17: 2009 Mo Standard CM-5 Performance



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. Approximately 95 core duplicates or about 2% of the total samples were submitted to Eco Tech in 2009. Table 13-3 summarizes the number of ¼ core samples by zone that were analyzed along with the length weighted average Au, Cu, Ag, and Mo grades for the "initial" and "duplicate" samples.

Table 13-3: Summary of 2009 ¼ Core Assay Results

Zone	Count	Meters	Mean Au (g/t)		Mean Cu (ppm)		Mean Ag (g/t)		Mean Mo (ppm)	
			Initial	Duplicate	Initial	Duplicate	Initial	Duplicate	Initial	Duplicate
Kerr	9	17.00	0.17	0.17	2711	2734	0.6	0.7	13	18
Sulphurets	26	50.32	0.49	0.55	2876	2838	0.7	0.8	91	108
Mitchell	60	118.54	0.36	0.35	992	1004	2.1	2.1	67	67
Grand Total	95	185.86	0.38	0.39	1659	1659	1.6	1.6	69	74

As can be seen in Table 13-3, there is a very close comparison between the initial and duplicate core mean Au, Cu, and Ag assay grade results. There is approximately a 7% difference between the initial and duplicate Mo grades. The ¼ core gold and copper assay results are graphically compared as QQ plots in Figures 13-18 and 13-19, respectively.

Figure 13-18: 2009 ¼ Core Au QQ Plot

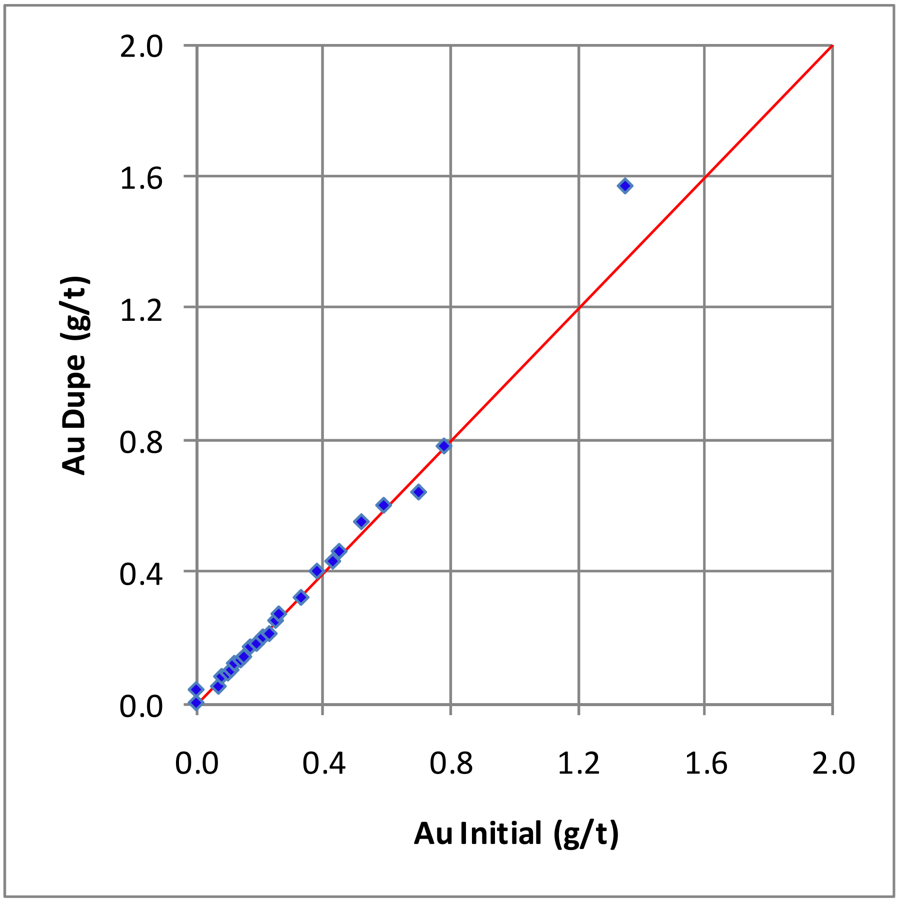
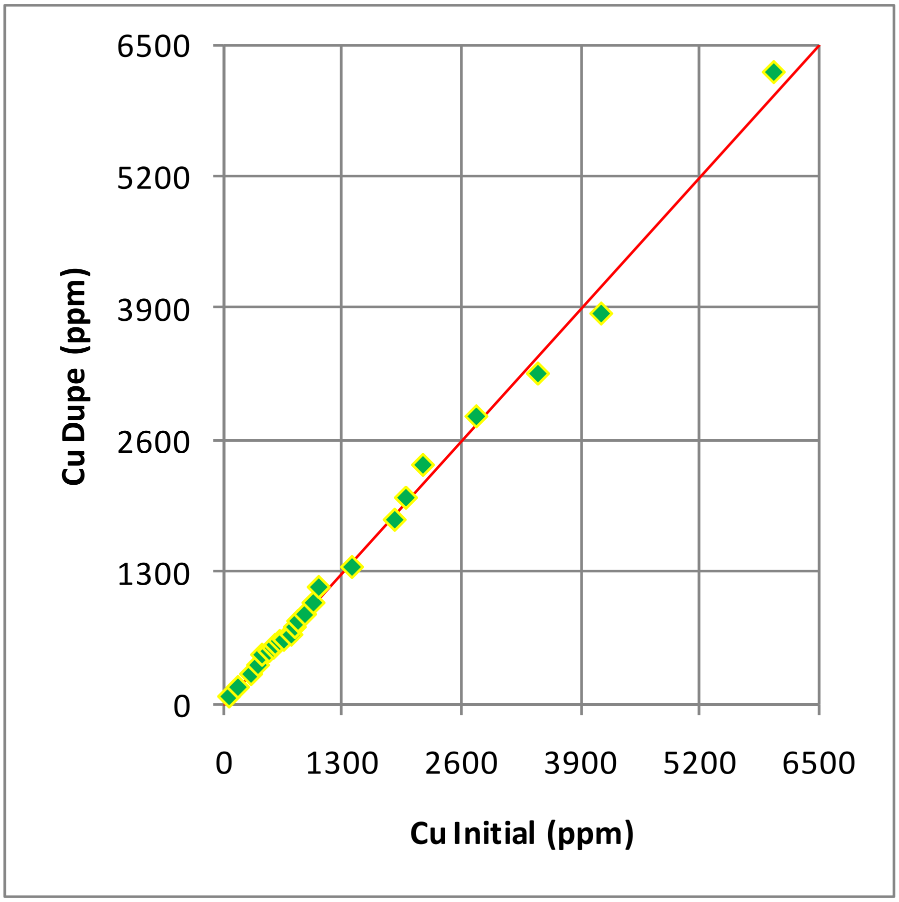


Figure 13-19 2009 ¼ Core Cu QQ Plot



About 9% of the 2009 samples (545 samples) that were assayed by Eco Tech were re-assayed as same pulp "cross-checks" by ALS Chemex of North Vancouver, B.C. QQ plots compare the same pulp gold and copper results in Figures 13-20 and 13-21, respectively.

Figure 13-20: 2009 Eco Tech vs. Chemex Au Check Assays

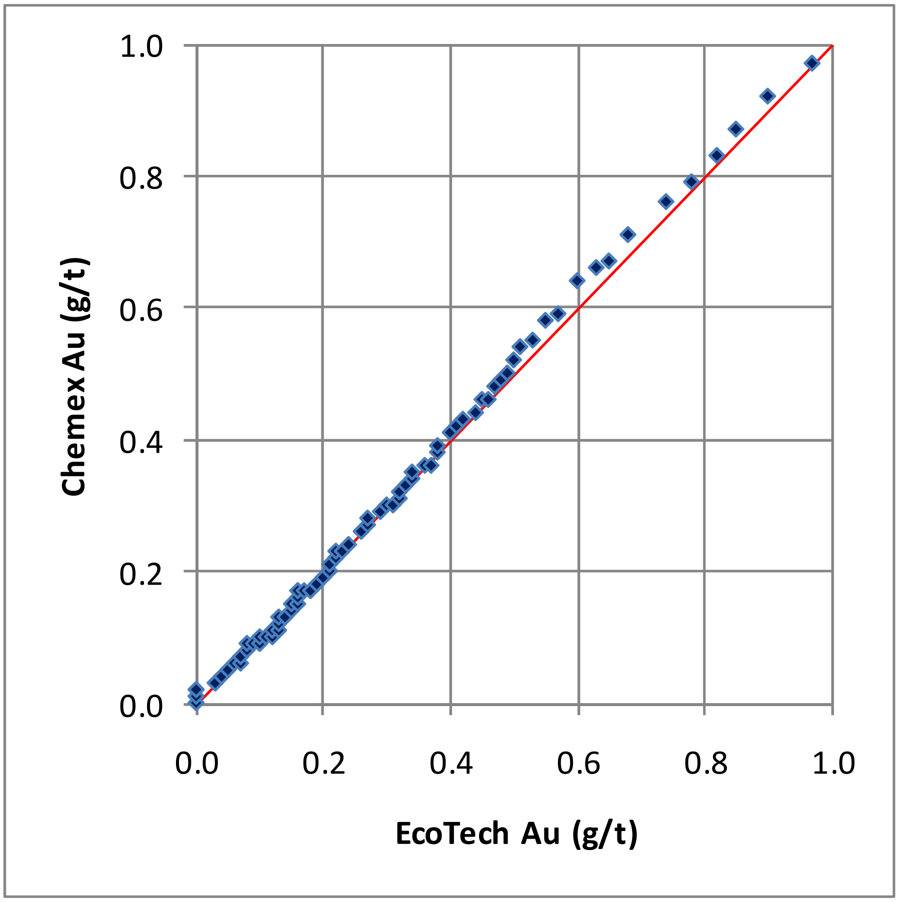
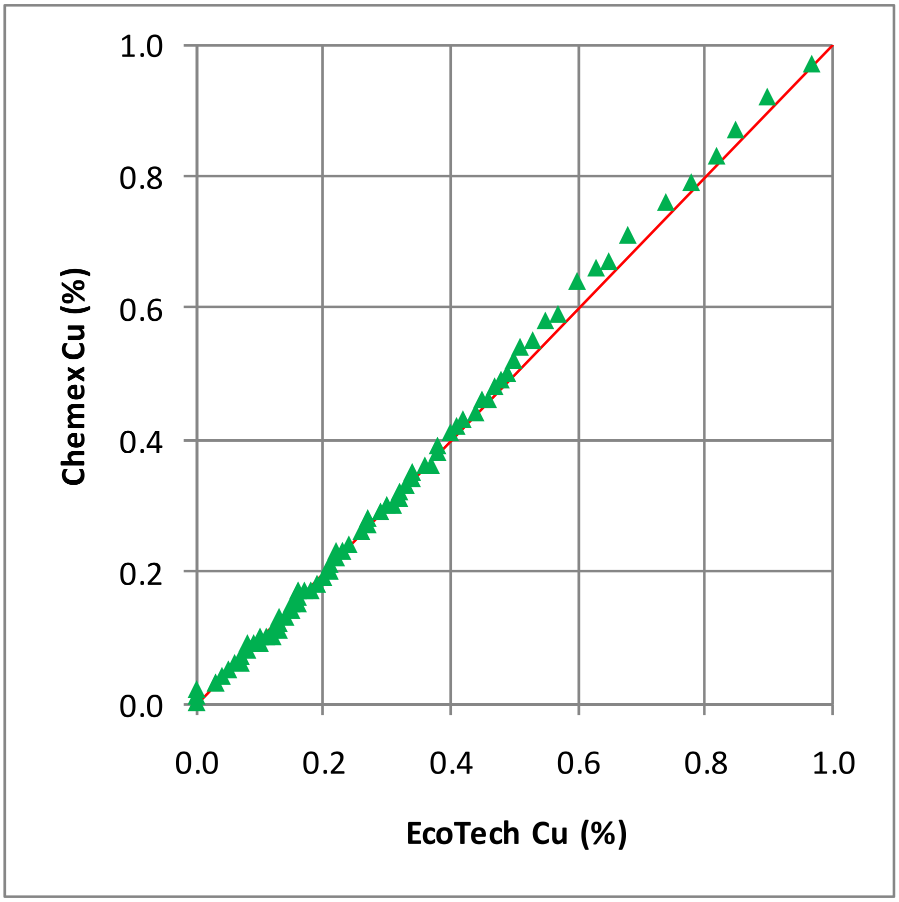


Figure 13-21: 2009 Eco Tech vs. Chemex Cu Check Assays



Both Eco Tech and Chemex employed the same assay preparation and measurement technique 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 **Author's Opinion**

Seabridge noted two QA/QC failures (batches 6004 and 6060) early in the 2009 assaying program. Samples associated with these batches were re-run. They also detected three cases where erroneous standards were recorded by Seabridge personnel and two cases where Eco Tech switched several sample numbers. The Seabridge QA/QC program worked properly in identifying these common errors.

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 2009 data.

14.1 Electronic Database Verification

The author performed an audit of the 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 2009 drill holes for verification. The data that were verified are summarized in Table 14-1. RMI notes that no errors were discovered. The data shown in Table 14-1 represent about 23% of the 2009 Seabridge assay data (34% of the Kerr, 26% of the Sulphurets, and 21% of the Mitchell data).

Table 14-1: **2009 Database Verification**

Drill Hole	Zone	Number Checked	Meters Checked	Au Errors	Cu Errors
K-09-01	Kerr	178	344.25	0	0
MW-09-08A	Kerr	20	45.40	0	0
S-09-09	Sulphurets	237	446.20	0	0
S-09-14	Sulphurets	226	438.00	0	0
MW-09-11A	Sulphurets	16	31.92	0	0
M-09-095	Mitchell	336	647.86	0	0
M-09-099	Mitchell	343	677.44	0	0
M-09-109	Mitchell	103	198.90	0	0
MW-09-06A	Mitchell	45	83.45	0	0
Grand Total	n/a	1,504	2,913.42	0	0

It is the author's opinion that the Mitchell Creek electronic database that was used to estimate Mineral Resources that are subject to this report is accurate. This is based on the author'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. The SRM's were prepared and certified by CDN from gold-copper porphyry material obtained from the Casino property located in northern B.C. Approximately 190 SRM's were submitted to Seabridge's primary laboratory (Eco Tech Laboratories) as a part of their QA/QC program. About 186 blanks were submitted to Eco Tech along with 95 ¼ core duplicate samples. 545 Eco Tech pulps were shipped to ALS

Chemex in Vancouver for check assay purposes. A more thorough discussion of Seabridge's 2009 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 13-20).

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 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".

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 drilled seven holes into the Kerr deposit in 2009 and performed nine bulk density determinations on the primary lithologic and alteration types. The average of those nine samples was 2.84 g/cm^3 corroborating the more exhaustive bulk density testwork. Based on those results, the author determined that the mean bulk density value of 2.84 g/cm^3 was reasonable and so that value was used in calculating resource tonnes.

A total of 373 bulk density determinations have been collected from the Sulphurets zone. The author performed an analysis of the bulk density determinations by lithology and alteration. Like Kerr, there was not a significant difference in bulk density values for the primary host rocks. Lighter and heavier density values were used for unmineralized lithologies (2.71 for monzonite and 2.85 for diorite). An average bulk density value of 2.77 g/cm^3 was chosen by the author for calculating resource tonnes for the Sulphurets deposit.

A total of 814 bulk density determinations have been collected by Seabridge from their 2006 through 2009 Mitchell drilling programs. RMI closely compared these determinations by lithology, alteration, grade, depth, and location relative to the Mitchell thrust fault. Based on those analyses the author and Seabridge elected to assign bulk density values by several factors including lithology, alteration, and fault domain. Table 14-3 summarizes the bulk density values used in the KSM resource area.

Table 14-3: **KSM Bulk Density Values**

Area/Geologic Unit	Bulk Density (g/cm^3)
Kerr Deposit	2.84
Sulphurets Deposit	2.77
Overburden	2.00
Glacial ice	0.90
Hazelton Volcanics	2.77
Diorite	2.85
Monzonite	2.71
Mitchell Chlorite-Propylytic Alteration	2.74
Mitchell QSP-IARG Alteration	2.80
Mitchell Upper Plate Rocks	2.71
Mitchell Lower Plate Rocks	2.77

15.0 ADJACENT PROPERTIES

Silver Standard Resources Inc. has 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 0.35 g/t gold equivalent cutoff grade (P&E Mining Consultants Inc., 2010a).

Table 15-1:
SSRI Snowfield Mineral Resources Using a 0.35 g/t AuEq Cutoff

Resource Category	Tonnes (M)	Au (g/t)	Ag (g/t)	Cu (%)	Mo (ppm)	Au Ozs (M)	Ag Ozs (M)
Measured	136.9	0.94	1.7	0.11	99	4.14	7.7
Indicated	724.8	0.67	1.9	0.12	91	15.63	43.2
Measured + Indicated	861.7	0.71	1.8	0.12	92	19.77	50.9
Inferred	948.9	0.33	1.4	0.07	81	10.05	43.7

In addition to disclosing updated resources for their Snowfield deposit, Silver Standard Resources also disclosed a new resource estimate for their Brucejack property, which is located east of Seabridge's KSM property. The Brucejack deposit consists of seven 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 0.35 g/t gold equivalent cutoff grade (P&E Mining Consultants Inc., 2010b).

Table 15-2:
SSRI Combined Brucejack Mineral Resources Using a 0.35 g/t AuEq Cutoff

Resource Category	Tonnes (M)	Au (g/t)	Ag (g/t)	Au Ozs (M)	Ag Ozs (M)
Measured	9.9	2.06	75.0	0.66	23.8
Indicated	110.7	0.95	11.7	3.38	41.6
Measured + Indicated	120.5	1.04	16.9	4.04	65.4
Inferred	198.0	0.76	11.2	4.87	71.5

The qualified person for this technical report has not verified the resources disclosed by Silver Standard Resources for their Snowfield and Brucejack deposits. While there appears to be similarities between the Mitchell and Snowfield deposits, the Brucejack mineralization reported by Silver Standard is not necessarily indicative of mineralization found at the nearby Kerr, Sulphurets, and Mitchell zones.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

RMI obtained various metallurgical reports from several sources including TJS Mining-Met Services, Ltd., G & T Metallurgical Services, Ltd., SGS Lakefield Mineral Services and Wardrop Engineering Inc. The information has been summarized in chronological order. Most of the text for this section has been taken verbatim from various reports, is attributed and shown in italicized font where applicable.

16.1 Historical Test Work

This section was taken from the September 2009 PEA technical report (Wardrop, 2009).

"Wardrop received several historical test work reports from Seabridge. The historical test work was conducted by the following laboratories:

- *Coastech Research Inc. (Coastech Research), 1989*
- *Metallurgical Laboratory, Brenda Mines, 1989*
- *Research Centre, Placer Dome, 1990*
- *Research Centre, Placer Dome, 1991*

The test work includes the preliminary investigations of mineralogy, mineralized material hardness, and the metallurgical response to flotation process and cyanidation.

Test Samples

Several different samples have been tested in the historical test programs.

Coastech Research - 1989

Two types of samples from the Kerr mineralized zone were tested in the program - one representing the central high grade zone (High Grade) and the other representing the rest of the mineralized zones (Low Grade). The assayed and calculated grade values are shown in Table 16-1.

Table 16-1: Test Samples - Coastech Research, 1989

Sample	Au (g/t)	Ag (g/t)	Cu (%)
Low Grade			
Assay	0.55	-	0.68
High Grade			
Assay	0.44	2.74	1.05

Metallurgical Laboratory, Brenda Mines - 1989

Sample 106 was tested in this program, along with a sample from Brenda Mines. No sample description was included in the report.

Research Center, Placer Dome - 1990

Four composites were made up from a total of 560 individual samples of crushed drill core rejects weighing 2.3 t. They were labelled as Composites K-1 to K-4. Two further composites which were received from Coastech Research were also tested. These two composites were labelled as LG-01 and HG-01 for low grade and high grade samples, respectively.

Table 16-2: Test Samples - Placer Dome, 1990

Composite	Au (g/t)	Ag (g/t)	Cu (%)
K-1	0.26	1	0.52
K-2	0.32	1.1	0.59
K-3	0.29	0.9	0.4
K-4	0.44	3	1.3
LG-01	0.39	2.2	0.71
HG-01	0.36	2.3	1.03

Research Center, Placer Dome - 1991

The bulk samples identified as Rubble Zone Trench and Crackle Breccia Zone Trench were employed for the testing program. Exploration personnel from Placer Dome collected the bulk samples. The average gold, silver, and copper values are shown in Table 16-3.

Table 16-3: Test Samples - Placer Dome, 1991

	Au (g/t)	Ag (g/t)	Cu (%)
Rubble Zone Composite	1.21	2.57	0.78
Crackle Breccia	0.34	1.58	0.4

Mineralogy

In 1990, Placer Dome examined mineralogical characteristics on the K-1 to K-4 composites and the results are summarized in Table 16-4.

Table 16-4: Mineralogical Characteristics - Placer Dome, 1990

Composite	Description
K-1	Sericite/chlorite and silicified tuffaceous rocks
K-2	Rubble zone - quartz/sericite/felsic/volcaniclastic sequence
K-3	Sericite volcaniclastic sequence c/w stockwork and veining
K-4	Quartz-sulphide veins and lenses - high grade

The determination also indicated that the iron and sulphur contents of the 4 samples varied in a narrow range, from 6.7 to 7.2% for iron and from 5.7 to 8% for sulphur.

Grindability

In 1989, using a comparative method, Brenda Mines determined the work index of Sample 106 to be 13.52 kWh/t, which was much lower than the work index of 19.78 kWh/t measured from the Brenda ore.

In 1990, Placer Dome calculated comparative ball mill work indices on Composites K-1 to K-4 and Composites LG-01 and HG-01. The comparative work index increased with the product particle size. No significant difference in grindability among the composites was determined. The obtained work indices were low as well, ranging from 7.4 kWh/t at a coarse product of 80% passing 205 µm (Composite K-4) to 12.8 kWh/t at a fine product particle size of 80% passing 45 µm (Composite K-3).

Similar grindability tests were conducted on the 1991 samples by Placer Dome. The comparative grinding work index of the Rubble Zone composite was similar to the data obtained from the 1990s samples. However, the comparative grinding index from the Crackle Breccia composite was much lower, ranging from 6.4 to 8.0 kWh/t, indicating a softer material.

Specific Gravity

Results of bulk and dry specific gravities conducted by Placer Dome in 1990 and 1991 are summarized in the Table 16-5. The average specific gravity (SG) and bulk SG are 2.89 and 2.82, respectively.

Table 16-5: SG Determination Results

Sample	SG	Bulk SG
K-1	2.94	-
K-2	2.9	-
K-3	2.96	-
K-4	2.9	-
HG-01	2.92	-
LG-01	2.88	-
Rubble Zone	2.83	3
Crackle Breccia	2.82	2.63
Average	**2.89**	**2.82**

Flotation

Metallurgical Laboratory, Brenda Mines - 1989

The test program preliminarily studied the responses of the Sample 106 to conventional copper and gold flotation. Open circuit cleaning tests failed to provide a marketable grade copper concentrate due to the coarse primary grind.

The test work showed that high copper and gold recoveries could be obtained with a grind size of 75% minus 200 mesh. However, to obtain the required concentrate grade it was necessary to depress iron sulphides. Depression of the iron sulphides with cyanide and pH control was shown to be possible; however, iron depression was very sensitive to the dosage of sodium cyanide. Small amounts of cyanide improved rougher concentrate grades and avoided precious metal losses in subsequent cleaning steps. The test results suggested using a selective xanthate collector for copper recovery and a dithiophosphate collector for gold recovery.

Research Centre, Placer Dome – 1990

High copper and precious metal recoveries were achieved in all tests. Saleable copper concentrates were produced in four of the six composites tested. Approximately half of the gold and silver reported to the final copper concentrate. A feed particle size of 80% passing 140 µm and a total of 16 minutes of flotation time were required to provide the metallurgical recoveries.

In the tests, lime and cyanide were added to depress iron sulphides. Sodium ethyl xanthate (R325) and Aerofloat 208 were added as copper and gold collectors. MIBC was added as frother. Rougher flotation was performed at pH 10.5.

The rougher concentrate was reground and the reground concentrate was cleaned in three stages of open circuit. The pH of cleaner flotation was adjusted to 11.

Primary grinds were evaluated from a coarse grinding size of 80% passing 140 to 175 µm to a fine grinding size of 80% passing 35 to 45 µm. It was found that the highest metal recoveries were obtained from the finest feed material.

Total copper recoveries for rougher and scavenger flotation ranged from 89 to 96%. Gold recoveries varied from 67 to 94% and silver from 81 to 95%.

The various samples showed differing metallurgical upgrading responses to the test conditions. Although regrinding and cleaning of the rougher concentrate at pH 11 rejected a significant amount of pyrite, Composite K-1 and K-2 produced poor results. The report indicated that the poor response was possibly due to the presence of sericite and mica slimes. It was recommended that sodium silicate or glue be added to the rougher floatation to suppress these minerals.

Gold recovery in the third cleaner concentrate was approximately 50% on average.

Research Centre, Placer Dome – 1991

The test program confirmed the earlier flotation test results that had been conducted in 1990. High final copper concentrate grades were produced from both composites.

Four grind and flotation tests were performed on each of the two mineral samples. The test results are summarized in Table 16-6.

Table 16-6: Flotation Test Results - Placer Dome, 1991

	Rubble Zone				Crackle Breccia			
Composite Test	**A**	**B**	**C**	**D**	**A**	**B**	**C**	**D**
Primary Grind								
- 80% passing (P_{80}), µm	223	175	149	98	165	110	99	59
Final Concentrate								
Grade								
- Cu (%)	32.0	30.4	32.3	28.2	30.9	29.9	33.2	26.1
- Au (g/t)	30.5	26.8	27.4	25.5	12.8	9.3	15	9.2
Recovery								
- Cu (%)	62.5	76.4	74.2	86.7	50.1	73	51.2	82.5
- Au (%)	41.4	44.2	40.4	48.5	23.1	29.6	26	35.7
Rougher/Scavenger Concentrate								
Recovery (%) weight	6.8	10.5	7.4	12.5	7.1	10.8	10.2	14.7
Recovery (%) Cu	73.3	86.1	89.3	96.6	73.9	83.6	87.1	93.1
Recovery (%) Au	61.1	74.7	68.5	79.8	51.1	56.8	63.9	66.4

The results indicated that copper and gold recoveries increased with an increase in primary grinding fineness. The finest primary grinds produced the best overall recoveries of copper and gold. The copper grades in the final concentrate grades ranged from 28 to 32% for the Rubble Zone sample and from 26 to 33% for the Crackle Breccia sample.

The gold and silver assay of the solutions from the rougher/scavenger tailing showed that the use of minor quantities of sodium cyanide in the flotation circuit for pyrite depression did not dissolve significant amounts of precious metals."

16.2 **2007 G & T Testwork**

This section was taken from the September 2009 PEA technical report (Wardrop, 2009).

"G&T Metallurgical Services Ltd. (G&T) conducted a metallurgical test program in 2007 that included material hardness determinations and flotation and cyanidation testing on three composite samples. These samples were collected from the Mitchell Zone and were of similar chemical and mineralogical composition. Table 16-7 shows the composition of the samples.

Table 16-7: G&T Test Samples Compositions

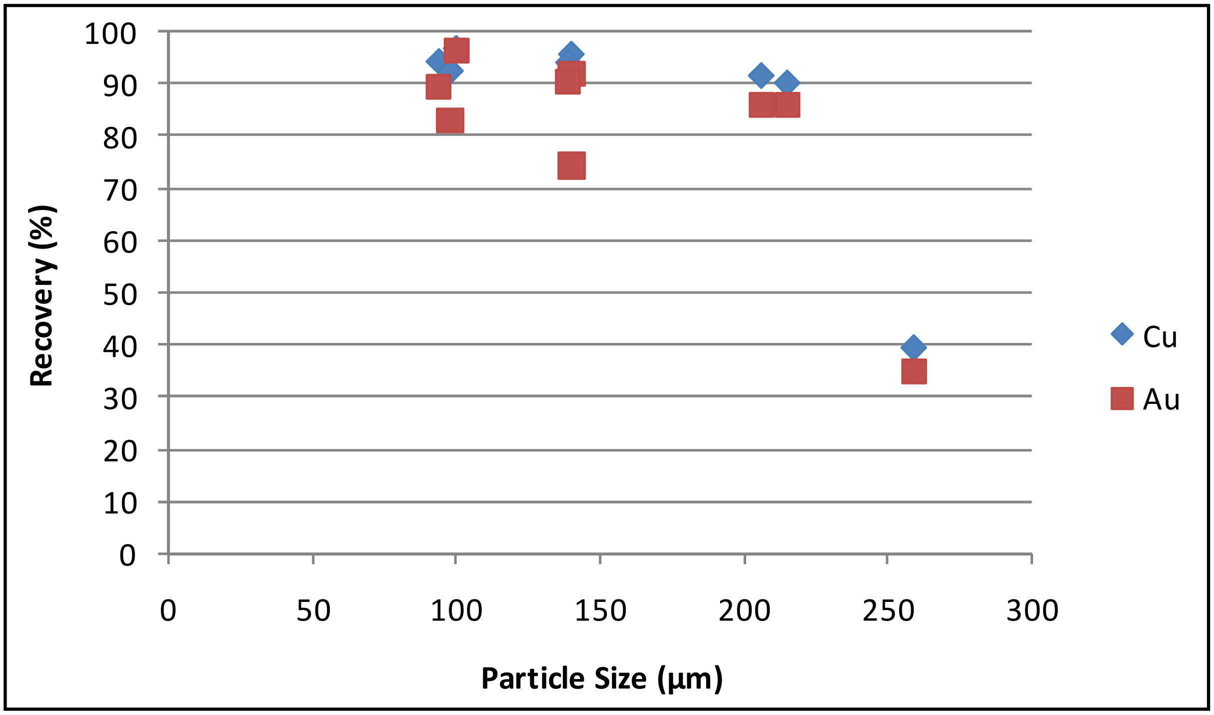
	Units	Composite			
		A	B	C	Average
Element					
Copper	%	0.2	0.2	0.2	0.2
Gold	g/t	0.9	0.9	0.9	0.9
Silver	g/t	3	4	4	4
Sulphur	%	4.6	3.6	1.8	3.3
Mineral					
Chalcopyrite	%	0.6	0.6	0.6	0.6
Pyrite	%	10	9.4	4.2	7.9
Gangue	%	89.5	90	95.2	94.9

Mineralogical determination testing revealed that approximately 53% of chalcopyrite in all three composites liberated at a primary grind size P_{80} of 140 µm. The Bond ball mill work index (Wi) was measured at approximately 14.8 kWh/t as shown in Table 16-8.

Table 16-8: G&T Bond Ball Mill Work Index

Sample	Wi (kWh/t)
A	14.7
B	14.8
C	14.8

Figure 16-1:
Effect of Primary Grind Particle Size on Rougher Flotation Recovery



Results of rougher kinetic flotation tests showed that a primary grind P_{80} of 140 µm generated acceptable metallurgical results. A primary grind size P_{80} of 200 µm resulted in a decrease of about 5% in both copper and gold recovery. However, the report indicated that there was potential to improve metal recovery by slightly increasing mass recovery at the primary grind size. The effect of primary grind size on rougher flotation recovery is shown in Figure 16-1.

Open circuit batch cleaner tests revealed that between 80 to 85% of the copper could be recovered to a final concentrate containing approximately 25% copper. Gold recovery to the final copper concentrate ranged from 46 to 54%. About 25 to 40% of the gold in the flotation feed reported to the combination of the copper first cleaner tailing and the pyrite concentrate.

Cyanidation tests on copper first cleaner tailing and pyrite concentrate were conducted to investigate the response of the gold bearing products to cyanidation. The test results showed that between 65 and 70% of the gold contained was extracted after 24 hours of cyanidation leaching. The combined gold recovery from flotation plus cyanidation was about 72%.

Mineralogical examination showed that some of the gold occurred as small inclusions in pyrite. Regrinding the cyanidation feed to P_{80} of 15 µm from 35 µm improved gold extraction from the leach feed by 15% for Sample A."

16.3 2008 G & T Testwork

This section was taken from the September 2009 PEA technical report (Wardrop, 2009).

"In 2008 comprehensive test work was carried out by G&T and Hazen Research Inc., under supervision performed by T.J. Smolik of TJS Mining-Met Services, Inc. The test work included mineralogical characteristic determination, grinding resistance determination and mill sizing simulation, flotation flowsheet development, gold extraction of gold-bearing pyrite concentrate by cyanidation, free gold recovery by gravity separation, and ancillary tests.

Test Samples

A total of approximately 5,720 kg of individual samples were shipped to the G&T laboratory in two shipments. Most of the samples were collected from the Mitchell Zone. Two samples were generated from the Sulphurets Zone. The samples were constructed into 34 variability test samples (MET samples). The key element assay on the heads is shown in Table 16-9. The drill hole distribution for the Mitchell Zone and section views are shown in Figure 16-2 and Figure 16-3.

Figure 16-2: Mitchell Zone Metallurgical Samples - Plan View

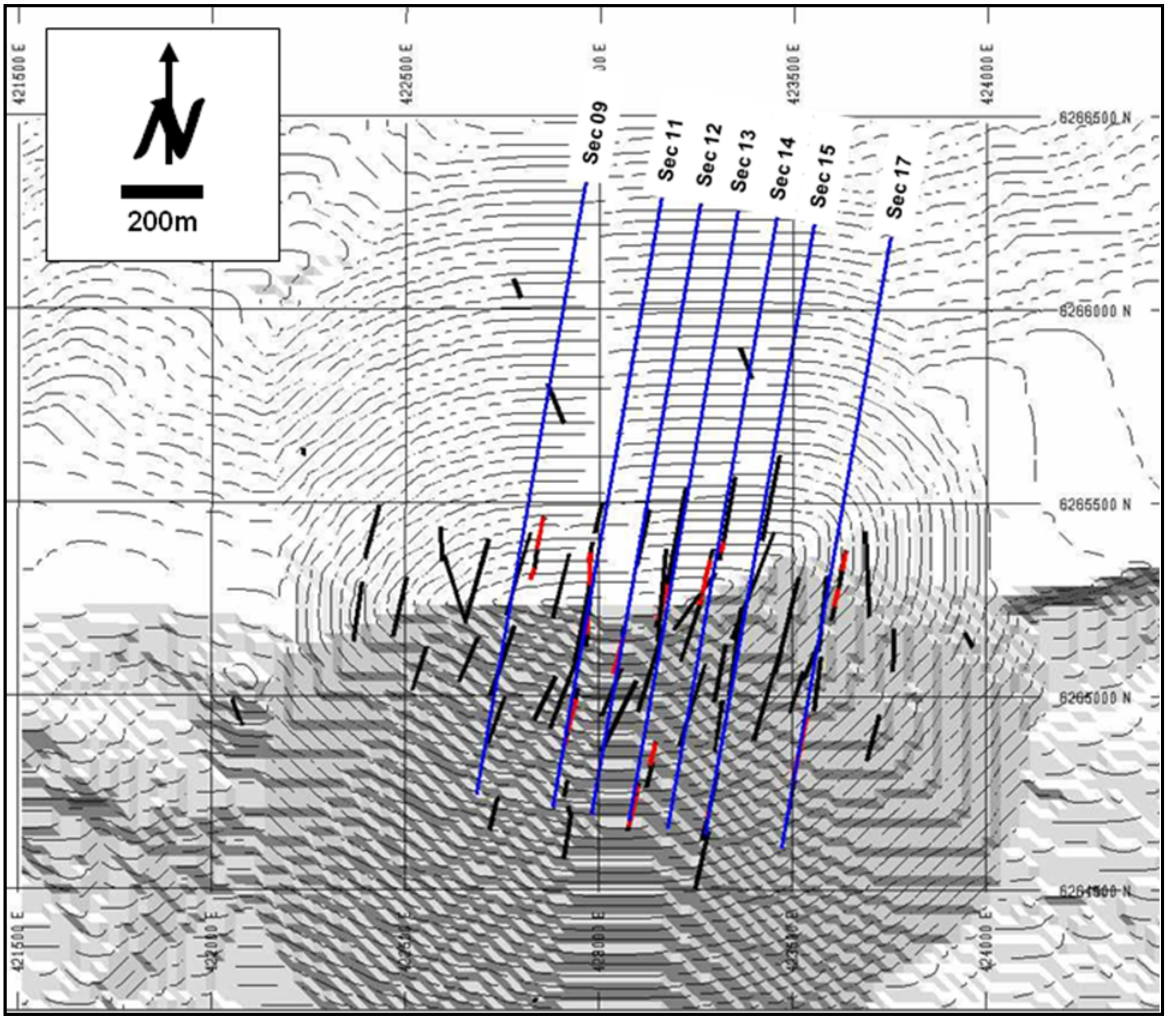
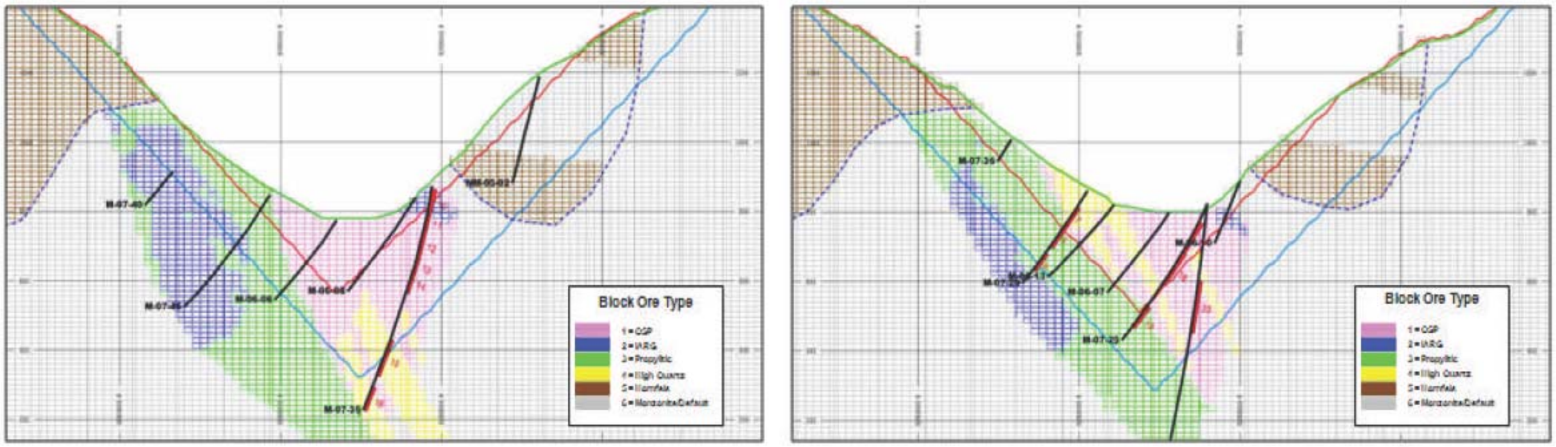


Figure 16-3: Mitchell Zone Metallurgical Samples - Section Views

Mitchell Zone Met Samples – Section 09
View Looking N80W

Mitchell Zone Met Samples – Section 11
View Looking N80W

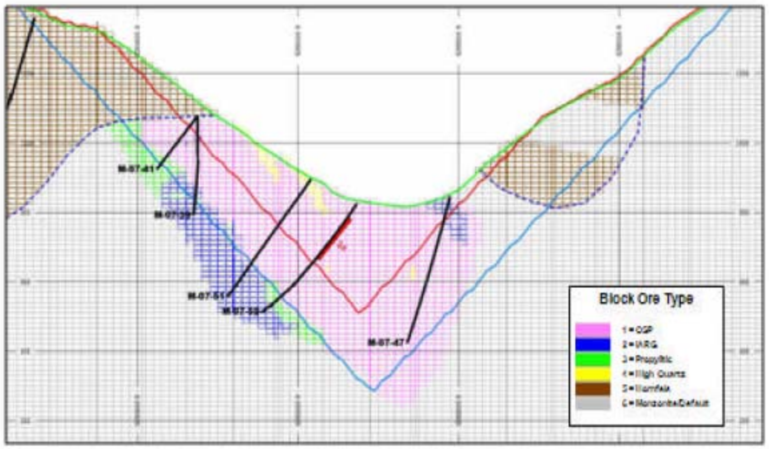


Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

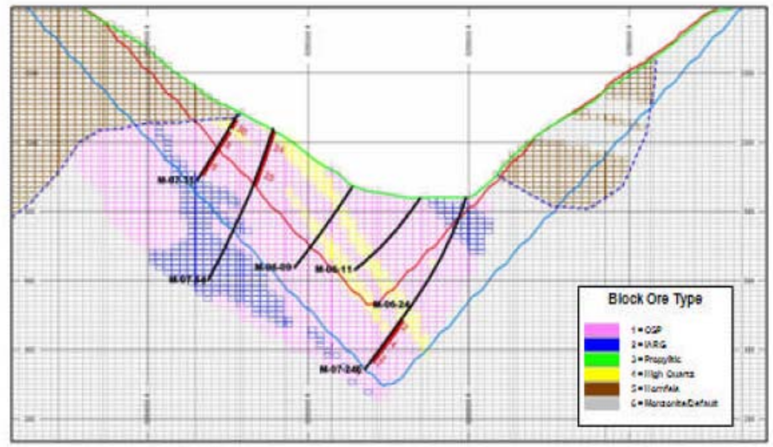
Figure 16-3 (con't): Mitchell Zone Metallurgical Samples - Section Views

Mitchell Zone Met Samples – Section 12
View Looking N80W



Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

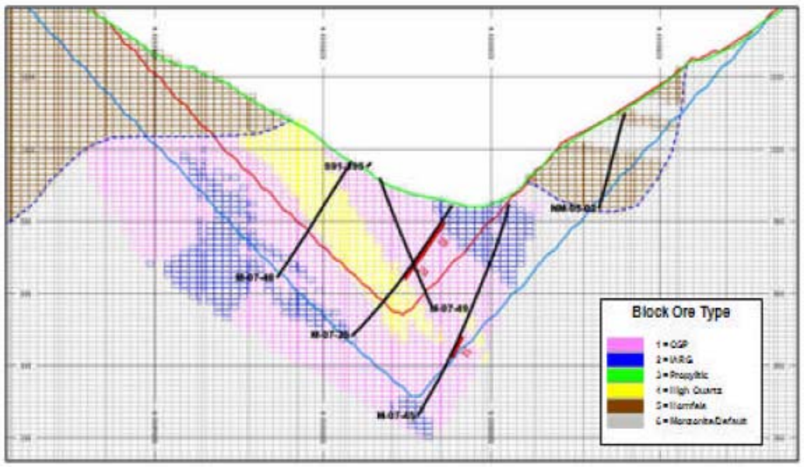
Mitchell Zone Met Samples – Section 13
View Looking N80W



Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

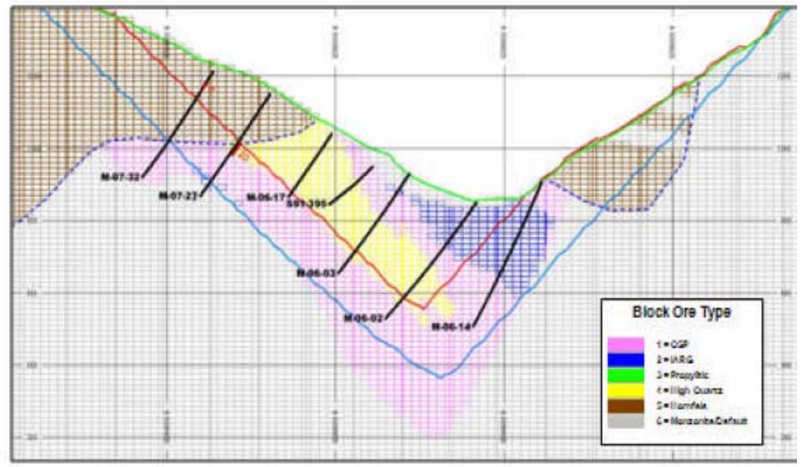
Figure 16-3 (con't): Mitchell Zone Metallurgical Samples - Section Views

**Mitchell Zone Met Samples – Section 14
View Looking N80W**



Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

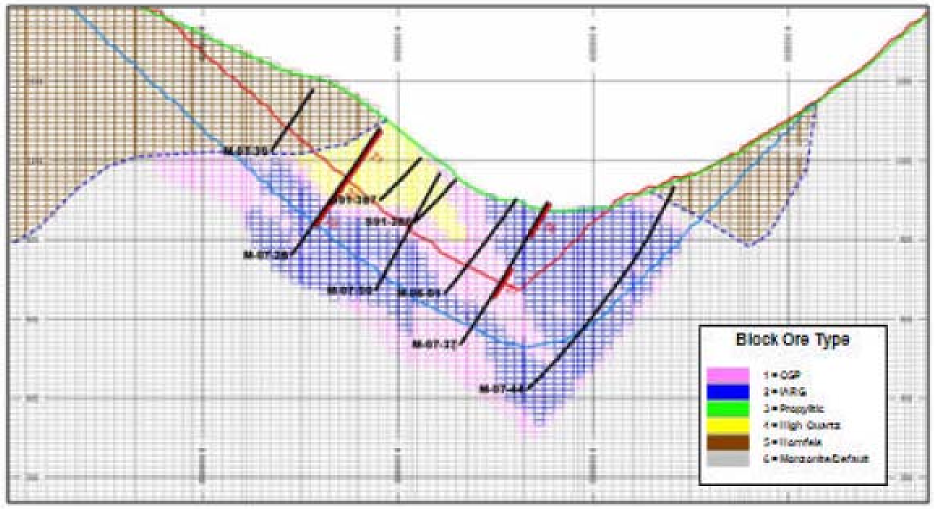
**Mitchell Zone Met Samples – Section 15
View Looking N80W**



Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

Figure 16-3 (con't): Mitchell Zone Metallurgical Samples - Section Views

Mitchell Zone Met Samples – Section 17
View Looking N80W



Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

Table 16-9: Head Assay on Variability Test Samples

Sample	Assay (% or g/t)*					Sample	Assay (% or g/t)*				
ID	Cu	Au	Ag	Mo	As	ID	Cu	Au	Ag	Mo	As
MET 2	0.25	0.82	4	0.003	0.003	MET 19	0.3	0.67	4	0.002	0.001
MET 3	0.24	0.65	8	0.004	0.02	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	0.2	0.66	2	0.004	0.001	MET 22	0.2	0.85	3	0.011	0.002
MET 6	0.21	0.74	2	0.01	0.001	MET 23	0.11	0.32	3	0.025	0.01
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	0.01	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	0.2	0.79	3	0.002	0.001	MET 29	0.19	0.79	5	0.018	0.006
MET 13	0.3	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	0.3	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 10 composites were generated from the MET samples. Five composites representing the major Mitchell Zone mineralization types projected to be mined in the initial 0-10 years were composed from the various drilling interval samples. The major mineralization types include:

- *Composite QSP: quartz, sericite, pyrite*
- *Composite Hi Qtz: significant quartz veining and pyrite*
- *Composite IARG: intermediate argillic with sericite, chlorite, and pyrite*
- *Composite Prop: propylitic rock with sericite, epidote, and pyrite.*

Table 16-10: Head Assay on Composites from Main Mineralization Type

Sample ID	Assay (% or g/t)*				
	Cu	Au	Ag	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

* g/t for Au and Ag

Two samples representing 10-30 years mine production and two samples representing 0-30 years mine production from two of the main rock components (QSP and Hi-Qtz) in the Mitchell Zone were also generated from the drilling interval samples (MET samples). A master composite was also generated from QSP and Hi-Qtz composites for locked cycle tests. The feed grades for the composites are shown in Table 16-10.

The origin of the MET samples and the composite samples are detailed in G&T's test work report submitted to Seabridge.

Mineralogical Determination

The mineralogical composition study shows that the sulphide mineral content in all three studied samples of QSP 0-30, Hi Qtz 0-30, and Master Composite 1, is dominated by pyrite. About 6 to 8 % of the sample weight is present as pyrite and chalcopyrite. It was indicated that copper was present in the form of chalcopyrite. Detailed analysis data are presented in Table 16-11.

Table 16-11: Mineral Composition Data

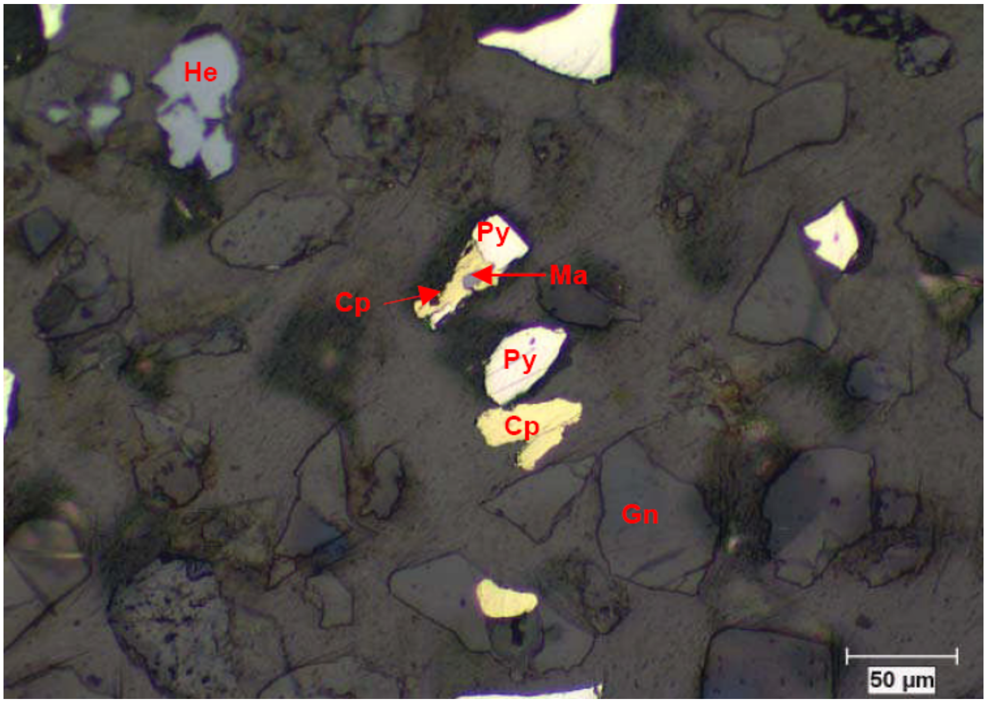
Sample	Mineral Composition (%)		
	Chalcopyrite	Pyrite	Gangues
QSP 0-30	0.66	6.6	92.7
Hi Qtz 0-30	0.67	8.2	91.2
Master Comp	0.54	8.1	91.4

The pyrite to chalcopyrite ratios are relatively high across the three tested samples. The average ratio is 12:1 while the highest ratio reaches 15:1. The two minerals do not show a high degree of interlocking in these samples. Figure 16-4 shows the relationship among main minerals in the samples.

The degree of chalcopyrite liberation ranged from 46 to 56% across the samples tested at a primary grind 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.

Figure 16-4: Mineral Relationship - Master Composite

Particle Fraction < 75 µm:



Particle Fraction < 150 µ > 75 µm:

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

Grindability And Mill Sizing Simulation

Grindability Determination

The grindability tests included semi-autogenous mill comminution (SMC) testing and standard Bond ball mill Wi. The samples used for the SMC grindability tests were identified as QSP, IARG, CL-RICH, QSP STW/QTVN, and H FELDS. The SMC test results are shown in Table 16-12.

Table 16-12: SMC Test Results

Parameter Sample	Value				
	QSP	IARG	CL-RICH	QSP N	H FELDS
Specific Gravity	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
DWi	5.5	7.9	7.1	5.4	7.5
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 in comparison to the JKTech database.

A separate standard Bond ball mill Wi determination testing was carried out by G&T on five composites identified as High Quartz 0-10, High Quartz 10-30, IARG 0-10, QSP 0-10, and QSP 10-30. The test results in Table 16-13 show that the samples have an averaged Bond Wi of 14.8 kWh/t indicating medium hardness to moderate hardness.

Table 16-13: Bond Ball Mill Wi Test Results

Samples	High Quartz 0-10	High Quartz 10-30	IARG 0-10	QSP 0-10	QSP 10 - 30	Average
Wi (kWh/t)	15.2	15.3	13.9	14.5	15.2	14.8

G&T also compared hardness variation on various variability test sample and main mineralization type composites by the comparative work index (CWi) method. The CWi was calculated from grind calibration data and the standard Bond ball Wi. The data is compared in Figure 16-5 for the various variability test samples and in Figure 16-6 for the composite samples. The average CWi values are 16.7 kWh/t for the individual samples and 15.5 kWh/t for the composite samples. Two of the mineral samples, Met 35 and Met 36, which were from the Sulphurets zone, produced much higher CWi values.

Figure 16-5: Comparative Ball Mill Wi Values - Variability Samples

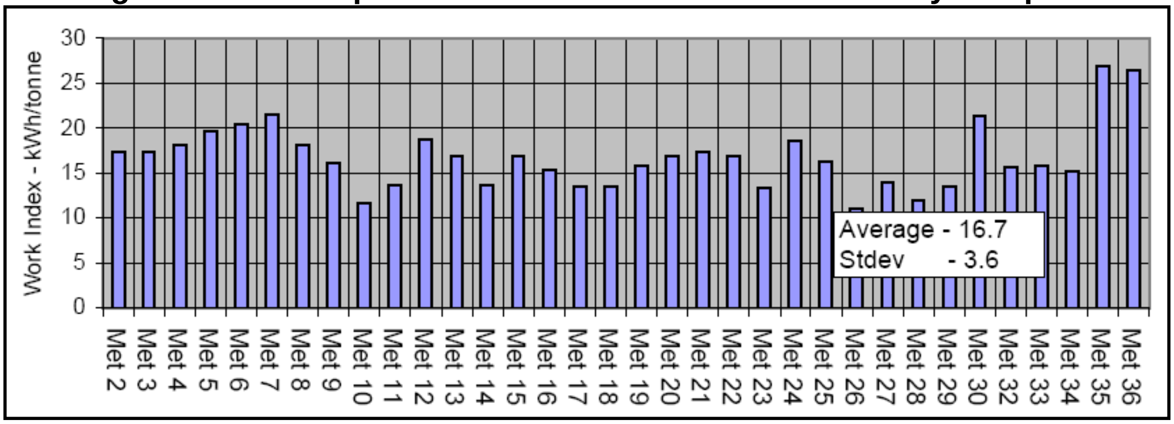
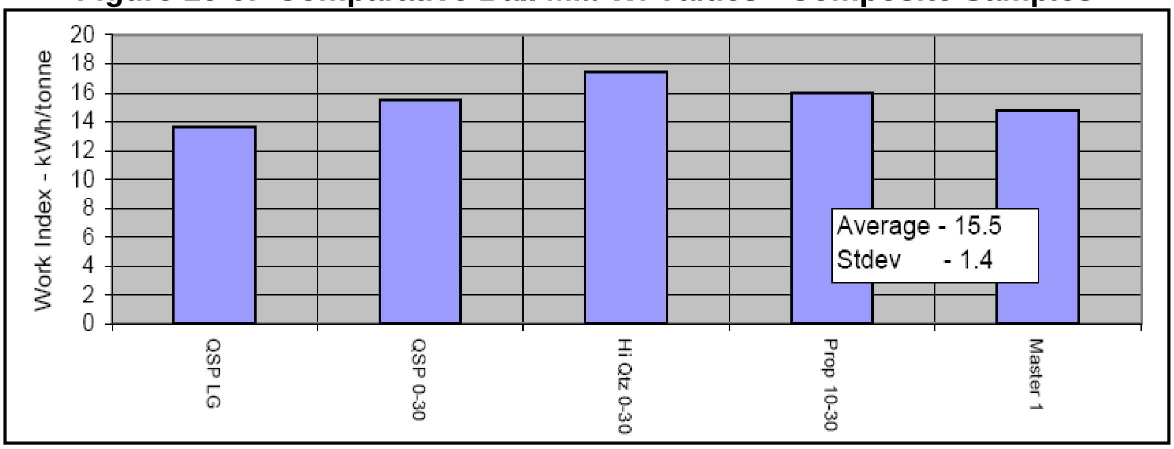


Figure 16-6: Comparative Ball Mill Wi Values - Composite Samples



Mill Sizing Simulation

Three mill sizing simulations were conducted by Contract Support Services Inc. using JK SimMet software. All the simulations were based on the data generated from SMC testing. The simulation input conditions are based on 120,000 t/d (two streams, 60,000 t/d each stream), 92% availability, a feed particle size of 80% passing 150 mm and one of the following conditions:

- *Simulation 1: Bond ball mill Wi 14.8 kWh/t, a product particle size of 80% passing 150 µm.*
- *Simulation 2: Bond ball mill Wi 16 kWh/t, a product particle size of 80% passing 150 µm.*
- *Simulation 3: Bond ball mill Wi 15 kWh/t, a product particle size of 80% passing 120 µm.*

Table 16-14: JK SimMet Simulation Results

Simulation		1a	1b	2a	2b	3a	3b
SAG Mill	Size, D x L (EGL) (ft x ft)	40 x 24	37.7 x 21	40 x 24	37.7 x 21	40 x 24	37.7 x 21
	Circulation Load (% of Feed)	19.5	18.4	19.5	18.4	19.5	18.4
	Gross Power Draw (kW)	18,843	15,570	18,843	15,570	18,843	15,570
Transfer Particle Size, mm		2,500	3,035	2,500	3,035	2,500	3,035
Ball Mills	Size, D x L (EGL) (ft x ft)	22 x 36	22 x 36	22 x 36	22 x 36	22 x 36	24 x 38
	Mill Number	2	2	2	2	2	2
	Gross Power Draw* (kW)	15,644	17,293	16,912	18,695	19,283	21,017
Total Power Draw (kW)		34,487	32,863	35,755	34,265	38,126	36,587
Cyclone Diameter (in)		26	26	26	26	26	26

* with Phantom Cyclones

Simulation results for each primary grinding stream are summarized in Table 16-14. The simulations are based on Phantom cyclone assumption and with primary cyclones for SAG mill discharges. The simulation results appear to show that, when the primary grind size is increased to 80% passing 120 µm, either of the following options will meet the primary grinding requirements:

- *one-40 ft dia. x 24 ft L SAG mill and two-22 ft dia. x 36 ft L ball mills, or*
- *one-38 ft dia. x 21 ft L SAG mill and two-24 ft dia. x 38 ft L ball mills.*

The simulation also indicated that less energy consumption would be expected if SAG mill discharges are classified by primary cyclones prior to ball mill grinding. The descriptions are detailed in the simulation reports.

Process Flowsheet Development

The flowsheet used for copper and gold recovery from the 2008 metallurgical samples was developed from the 2007 test program. The process used a combination of flotation and cyanidation. Copper and gold were first recovered into a copper-gold rougher concentrate. The copper flotation tailing was refloated to produce a gold-bearing pyrite concentrate. The copper-gold rougher concentrate was reground and cleaned to produce a copper-gold concentrate. The cleaner tailing from the copper cleaning circuit was cyanide leached together with the gold bearing pyrite concentrate after regrinding and aeration.

Flotation Flowsheet Development

Collectors dithiophosphinates (3418A) and dithiophospate A208 were used in the copper circuits and collectors potassium amyl xanthate (PAX) and A208 in the gold-pyrite circuit. The slurry pH at the copper and pyrite rougher flotation was at 10.

Variability Tests

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 at 149 µm. The rougher concentrate from the copper circuit was reground to approximately 18 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 16-7, G&T established the relationship between copper recovery and copper feed grade at a fixed concentrate grade of 25% Cu. In general, copper recovery increased with an increase in copper feed grade. The variation in the metallurgical performance of various mineral samples is shown in Figure 16-8.

Figure 16-7: Copper Recovery vs. Copper Feed Grade - Individual Samples

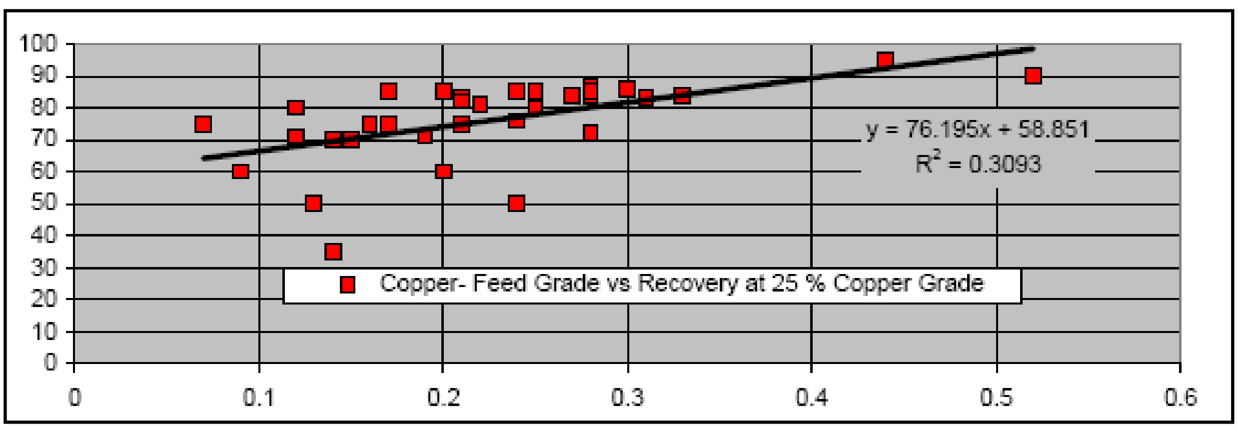
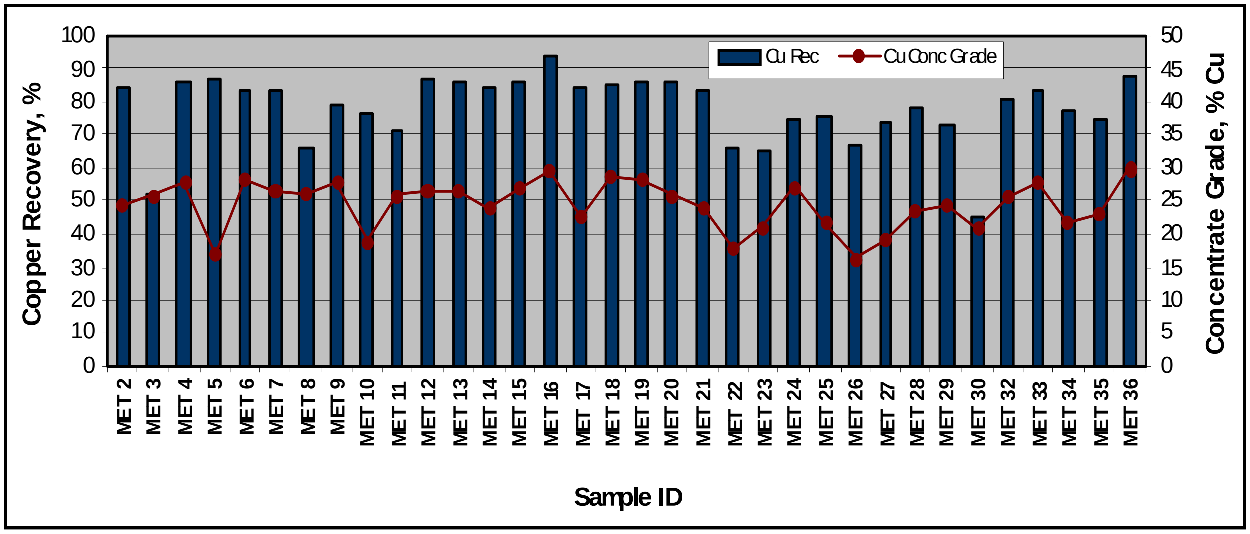
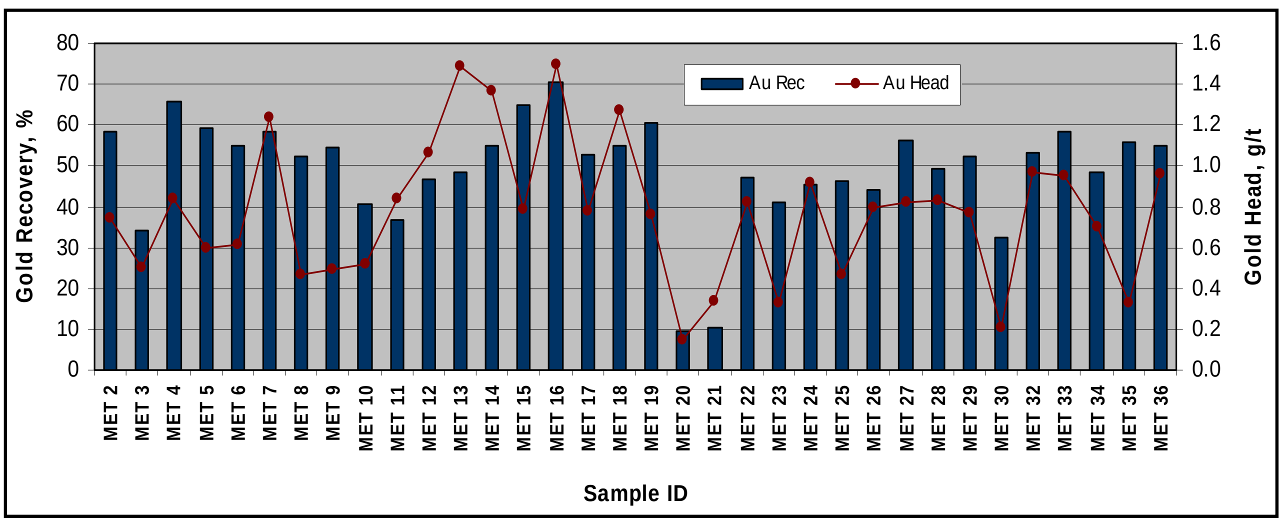


Figure 16-8: Copper Recovery and Concentrate Grade - Individual Samples



The gold recovery to the copper concentrate fluctuated from 30 to 70%, except for the MET 20 and 21 samples, which produced much lower gold recoveries. The tests seemed to shown 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 16-9.

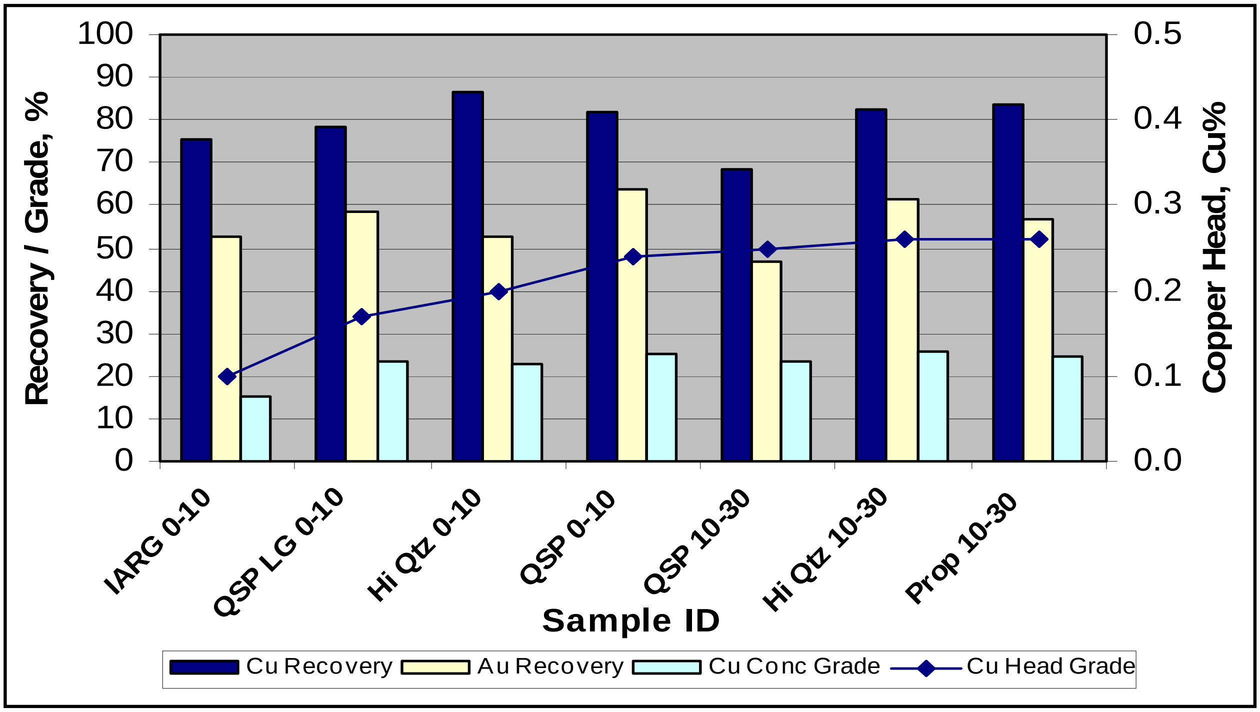
Figure 16-9: Gold Recovery and Feed Grade - Individual Samples



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

Further testing were conducted on seven composites representing the major Mitchell zone mineralization types projected to be mined in the initial 0-10 years and the later years. The test results are shown in Figure 16-10. At primary grind sizes ranging from 130 to 168 µm, the open cycle tests produced third cleaner concentrates with between 69 to 86% copper recovery and between 47 to 64% gold recovery.

Figure 16-10: Metallurgical Performance - Composite Samples



Similar to the variability tests, on average, the combined gold recovery from both the copper circuit and gold-pyrite circuit from the composite samples was approximately 86%.

- Primary Grind Size Optimization

The effect of primary grind size and regrind size on the metallurgical performance of QSP 0-30 and Hi Qtz 0-30 composites was conducted. The test results, as summarized in Figure 16-11 and Figure 16-12, show that copper and gold metallurgical performance at rougher flotation stage improved with a decrease in primary grind size, although much less significantly when the grind size was finer than 120 µm.

Figure 16-11: Metallurgical Performance vs. Primary Grind Size - QSP 0-30

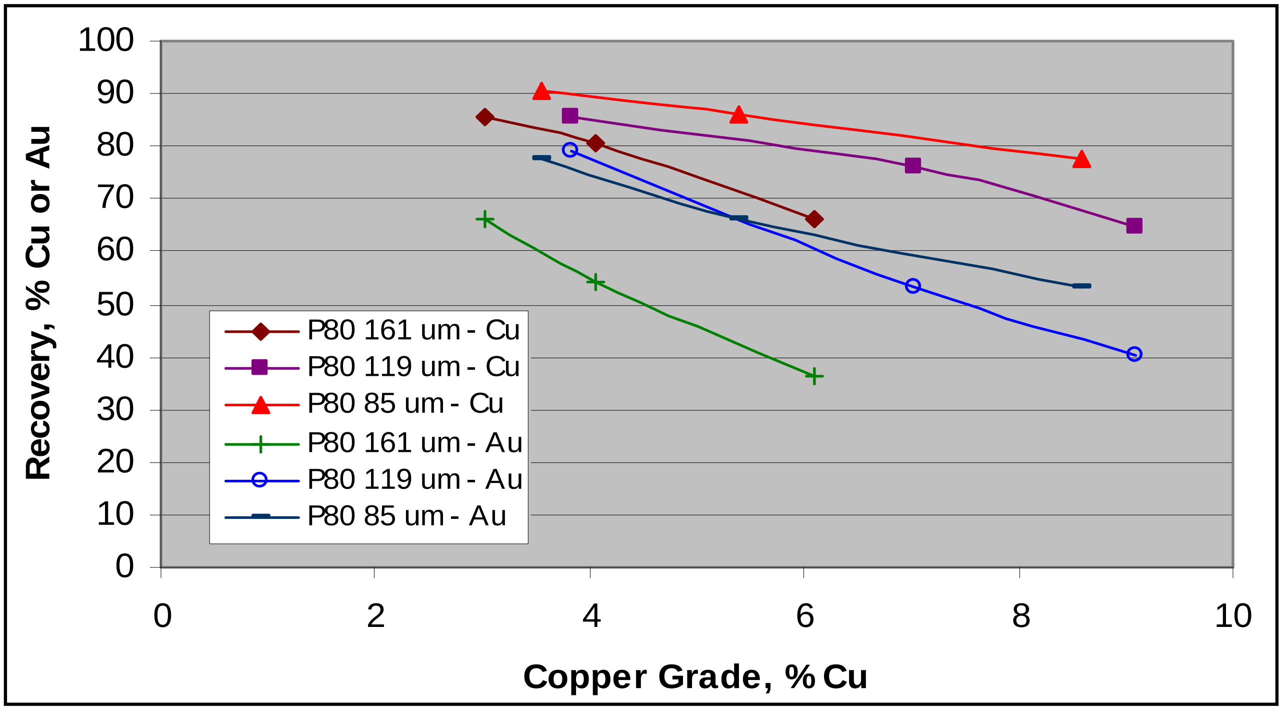
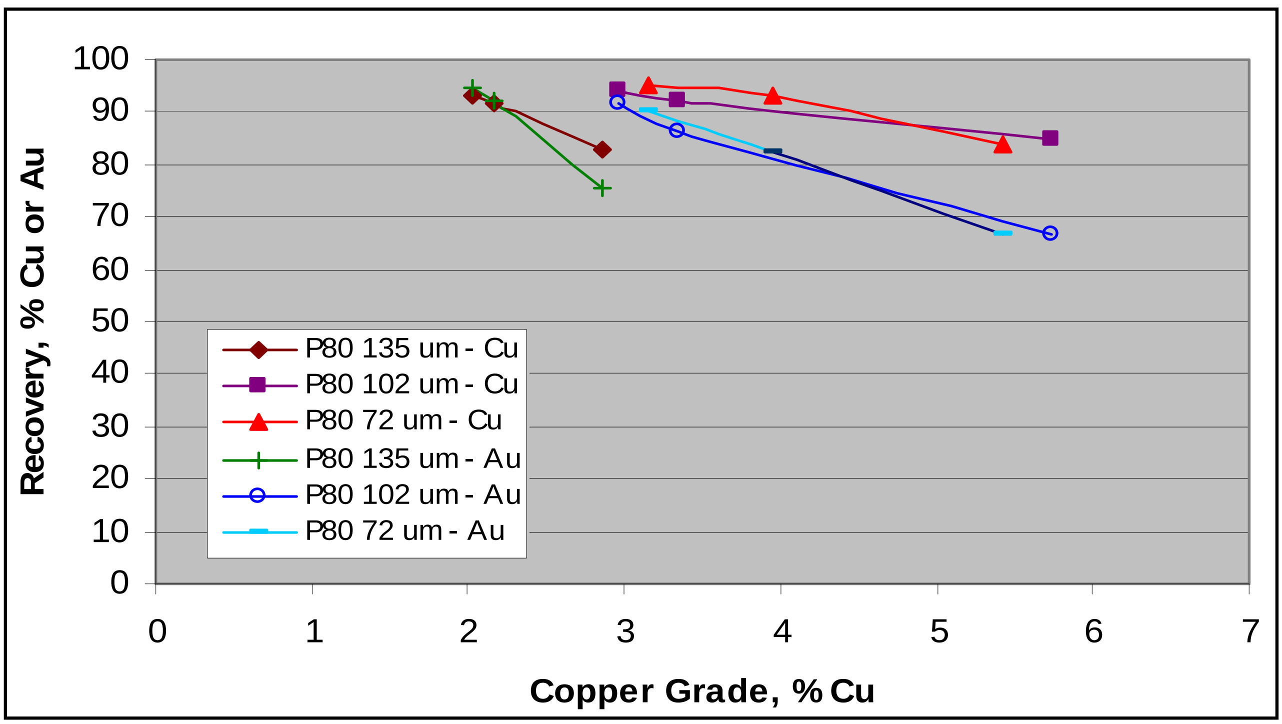


Figure 16-12: Metallurgical Performance vs. Primary Grind Size - Hi Qtz 0-30

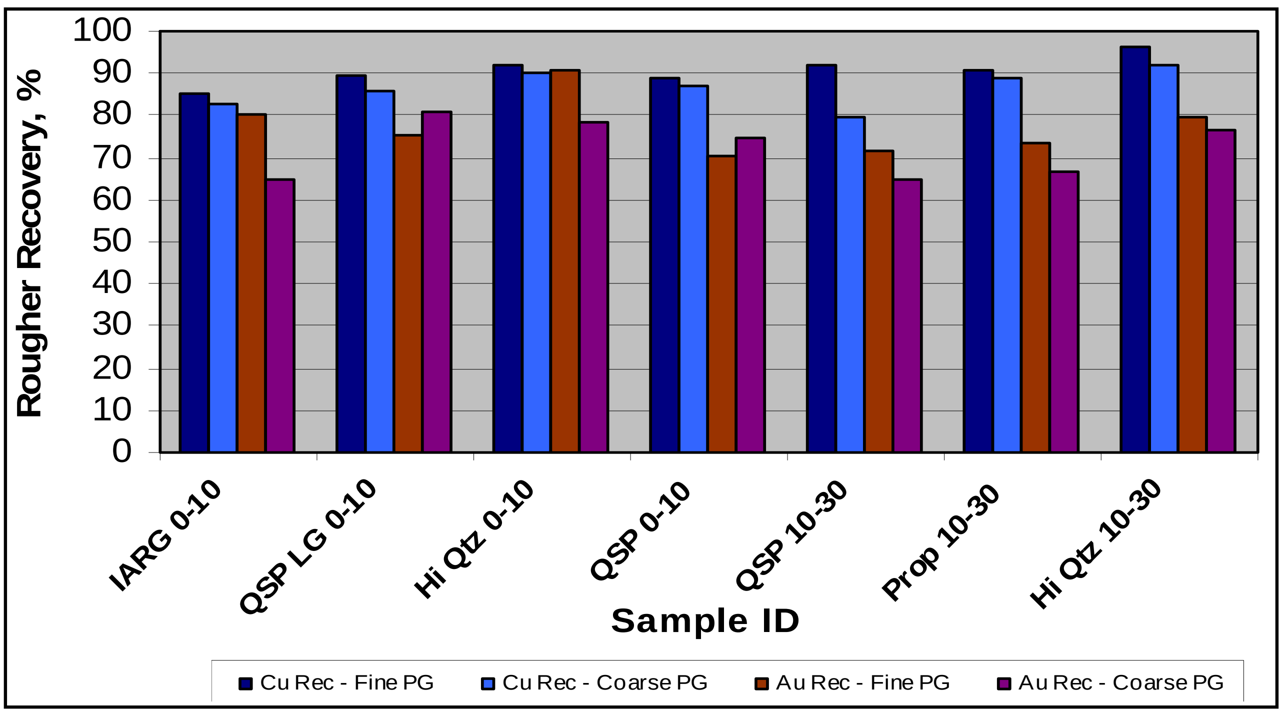


For QSP 0-30 composite, the copper recovery to a rougher concentrate, grading 4% Cu, improved from 81 to 89% when the primary grind size was decreased from 80% passing 161 µm to 80% passing 85 µm. Gold recovery increased significantly with the increase in the grind fineness; however, there was no significant increase when the grind size was finer than 80% passing 120 µm.

Hi Qtz 0-30 composite produced better metal recoveries compared with QSP 0-30 composite. The effect of primary grind size on the metallurgical performance was similar to that observed on QSP 0-30 composite.

Apart from QSP 0-30 and Hi Qtz 0-30 composites, two sets of comparison tests were performed on all the other composite samples to investigate the effect of primary grind size on copper and gold recovery. The average primary grind sizes tested were 80% passing 143 µm and 119 µm. The effect of the grind size on the metal recovery to rougher and third cleaner concentrates are shown in Figure 16-13.

Figure 16-13: Effect of Primary Grind Size on Metallurgical Performance



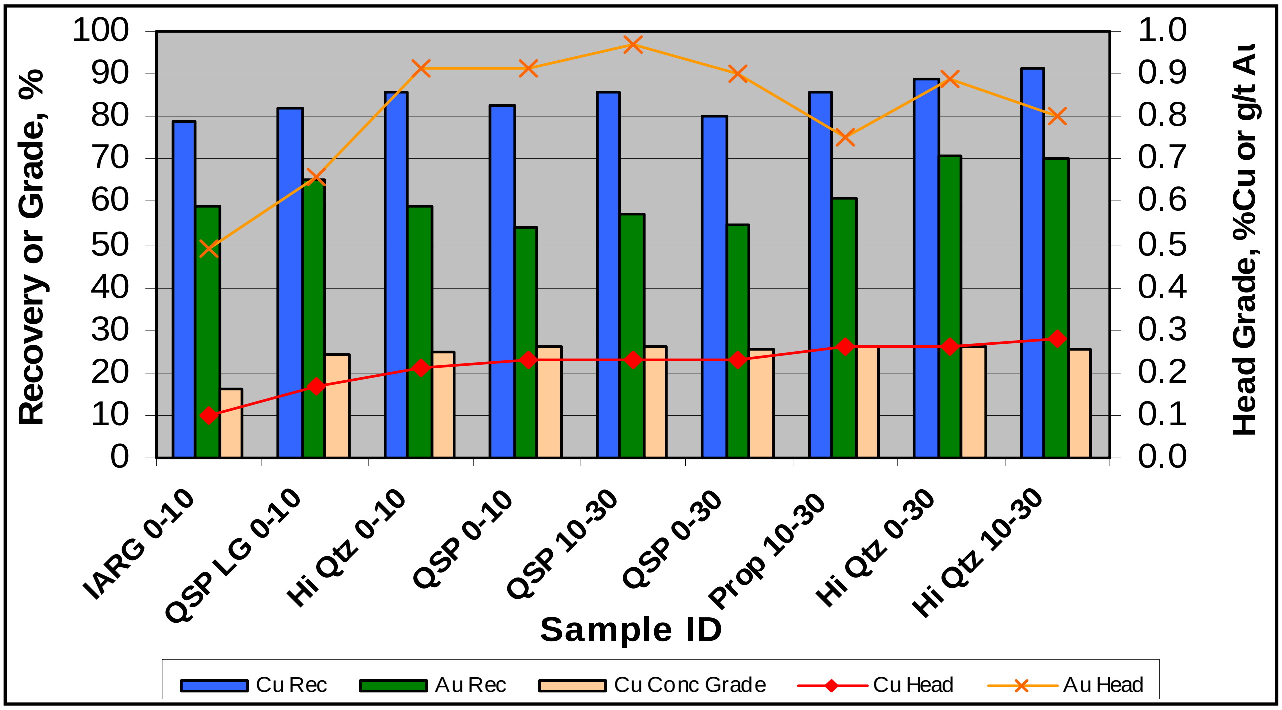
On average, the copper recovery reporting to rougher concentrate was 90.6% at the fine grind size, comparing to 86.6% at the coarse grind size. The average gold recovery to the concentrate increased from 72.3 to 77.3%. However, QSP 0-10 and QSP LG 0-10 composites appeared to show different gold metallurgical responses with a change in primary grind sizes.

At the fine grind size, the total average gold recovery from the copper circuit and pyrite circuit improved by approximately 4 to 89%.

- Open Circuit Tests

Open circuit tests with two stages of cleaner flotation at a pH value of 11.5 were performed on the 9 composite samples. Primary grind sizes ranged from 80% passing 87 µm to 137 µm, averaging at 119 µm. Regrind sizes varied from 80% passing 12 µm to 22 µm, averaging at 18 µm. The results are shown in Figure 16-14.

Figure 16-14: Metallurgical Performance - Open Circuit Tests



The second cleaner concentrate recovered between 79 to 91% of the copper and 54 to 71% of the gold from the 9 composites. On average, the metal recovery was 84.6% for copper and 61.2% for gold.

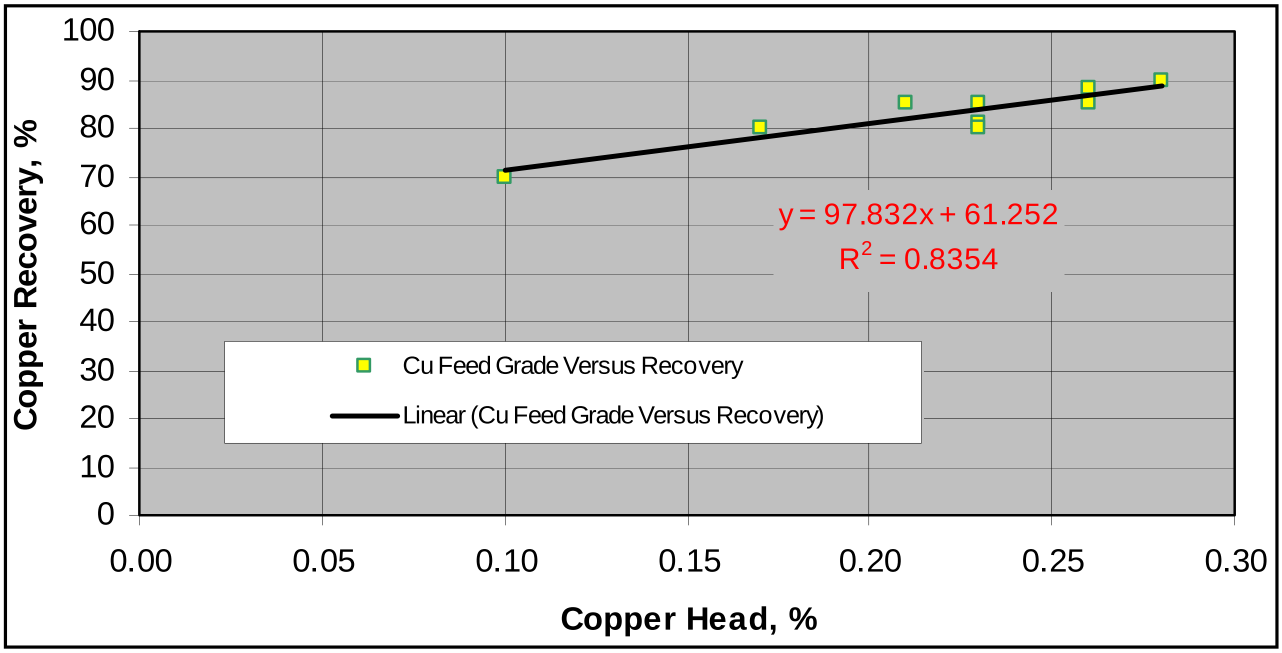
The results also appeared to show that copper recovery increased with an increase in copper head grade. The test results also showed that gold recovery did not seem to correlate with gold head grade or copper head grade.

Seven composites produced a concentrate of higher than 25% Cu, excluding 16.2% Cu from the IAGR 0-10 composite and 24.0% Cu from the QSP LG 0-10 composite.

After adjusting the copper recovery to reflect a concentrate grade of 25% Cu, a relationship between the adjusted copper recovery and copper feed grade is plotted in

Figure 16-15. It appears that copper recovery is relatively closely related to copper head grade.

Figure 16-15: Copper Recovery vs. Copper Feed - Open Circuit Tests



- Locked Cycle Tests

A master composite generated from Hi Qtz 0-30, QSP 0-30, and QSP 0-10 composites was used for locked cycle flotation testing. A total of three locked cycle tests (LCT) were carried out. The average results from two of the LCTs are shown in Table 16-15.

Table 16-15: Average LCT Results (Tests 141 and 142)

	Grade			Recovery			
Product	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Mass (%)**	**Cu (%)**	**Au (%)**	**Ag (%)**
Feed	0.21	0.9	4	100	100	100	100
Copper Concentrate	21.1	63.8	2.6	0.9	87	61	56
Copper 1st Cleaner Tailing	0.12	1.87	11	6.9	4	14	19
Gold-Pyrite Concentrate	0.1	2.14	8	5.8	3	14	11
Final Tailing	0.01	0.12	1	86.4	6	11	14

The average results indicate that 87% of the copper was recovered to the copper concentrate containing 21% copper. The concentrate also recovered 61% of the gold and 56% of the silver.

The gold and silver reporting to the gold bearing products, copper cleaner tailing, and gold-pyrite concentrates were approximately 28% and 30%, respectively.

Cyanidation of Gold-Bearing Pyrite Products

Because a portion of the gold is associated with pyrite, the first cleaner tailing and the gold-pyrite concentrate from the flotation circuit were subjected to cyanide leaching to recover the gold. The following discussion presents the results from the bottle roll cyanidation tests on the gold bearing products obtained from the flotation variability tests, open circuit tests, and locked cycle tests.

- Cyanidation Tests – Products from Flotation Variability Tests

A total of 30 cyanide leach tests were carried out on the gold bearing products from the flotation variability tests. Prior to the leach, the combined first cleaner tailing 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 16-16. The average gold extraction was approximately 79%. Increasing leach retention time did not improve gold extraction.

Table 16-16: Cyanidation Test Results - Individual Samples

Leach Time (h)	Sample ID	CN Test No.	Flotation Test No.	Regrind Size (P80 µm)	CN Feed (g/t Au)	CN Ext (% Au)
48	MET 2	50	4	11	1.7	60
	MET 5	51	7	9	1.6	79
	MET 8	52	10	9	2.2	74
	MET 11	53	13	10	6.3	94
	MET 14	54	16	15	2.7	81
	MET 17	55	19	13	1.9	87
	MET 20	56	22	11	1.1	58
	MET 23	57	25	15	1.3	82
	MET 26	58	28	13	2.7	85
	MET 29	59	31	10	4.1	83
	MET 33	60	34	16	1.9	88
Average				**12**	**2.5**	**79.2**
24	MET 3	64	5	12	1.4	65
	MET 4	65	6	13	1.6	78
	MET 6	66	8	9	2.4	84
	MET 7	67	9	11	3.4	78
	MET 9	68	11	9	1.3	74
	MET 10	69	12	11	2.7	91
	MET 12	70	14	10	3.3	87
	MET 13	71	15	10	8.9	90
	MET 15	72	17	14	2	85
	MET 16	73	18	13	3.2	82
	MET 18	74	20	11	1.37	63
	MET 19	75	21	12	1.99	82
	MET 21	76	23	9	2.15	69
	MET 22	77	24	12	2.74	63
	MET 24	78	26	10	4.1	87
	MET 25	79	27	9	1.7	78
	MET 27	80	29	13	2.21	81
	MET 30	82	32	11	1.63	76
	MET 32	83	33	7	3.35	91
Average				**11**	**2.7**	**79.2**
Average All				**11**	**2.6**	**79.2**

- Cyanidation Tests – Products from Flotation Open Circuit Tests

Similar to the leach tests of the individual samples, the combined products of the first cleaner tailing and the gold-pyrite concentrate from the open cleaner circuit tests were cyanide leached to confirm the responses of the gold bearing materials to cyanidation. The leach retention time was 24 hours. As shown in Table 16-17, the gold extractions *from the leach feed ranged from 65 to 89%. The average extraction was approximately 78% Au.*

Table 16-17: Cyanidation Test Results - Composite Samples

Sample ID	CN Test No.	Flotation Test No.	Regrind Size (P_{80} µm)	CN Feed (g/t Au)	CN Extraction (% Au)
QSP 0-10	126	116	10	2.2	82
IARG 0-10	127	117	12	1.3	80
Hi Qtz 0-10	128	118	11	2.3	74
QSP LG 0-10	129	119	12	1.7	74
QSP 10-30	130	120	11	2.3	89
Prop 10-30	131	121	11	1.6	82
Hi Qtz 10-30	133	123	21	2	66
QSP 0-30	134	124	12	2.2	78
Hi Qtz 0-30	135	125	12	1.6	65
Average			**12**	**1.9**	**78**

- Cyanidation Tests – Products from Flotation Locked Circuit Tests

The mixture of the first cleaner tailing and the gold-pyrite from the LCTs contained approximately 2.0 g/t Au and 9.6 g/t Ag. The leach tests showed that 70% of the gold and 63% of the silver were able to be extracted from the gold bearing products.

Average cyanide and lime consumptions in the two locked cycle flotation/cyanidation tests were 3.2 and 2.3 kg/t. The sodium cyanide concentration was varied between 1,000 and 2,000 ppm in Tests 144 and 145. The cyanide consumption was approximately 10% lower in the test using the lower NaCN concentration. The gold and silver recoveries were equivalent between the two tests. The test results are summarized in Table 16-18.

Table 16-18: Cyanidation Test Results - Master Composite

Sample ID	CN Test No.	Flotation Test No.	Regrind Size (P_{80} µm)	CN Feed		Extraction	
				g/t Au	g/t Ag	% Au	% Ag
Master	143	142	15	2.2	10.1	73.2	64.4
Master	144	141	15	1.8	9.1	67.6	61.5
Average			**15**	**2**	**9.6**	**70.4**	**63**

By adding the gold recoveries from the copper flotation circuit and the leach circuit, total gold and silver recoveries from the combined flowsheet were 80.7 and 74.9%, respectively.

Free-Gold Recovery

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

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.

This data indicates that the MET 4 sample responded well to the gravity separation, although most of them produced poor metallurgical performances.

Ancillary Tests

Settling Tests

Preliminary settling tests were conducted on pyrite flotation tailing. As reported by G&T, the tests on the tailing in slurry form failed to generate normal settling curves. As a result, the tests were carried out on the re-pulped sample from dried tailing from the same test.

The test data reveal that the settling area required was 0.73 m^2/t/d without adding flocculant and 0.30 m^2/t/d with the addition of 10 g/t of flocculant.

Magnetic Separation Tests

In the test program, Davis Tube magnetic separation was used in an effort to recover the metal values lost in the coarser than 200 mesh fraction of the pyrite flotation tailing from Tests 10, 11, and 25. Test results indicated that less than 3% of the coarse tailing weight was recovered into a magnetic fraction assaying approximately 23% iron. No copper or gold assay data was reported.

Concentrate Assay

The copper concentrate from the LCT (Test 142) was subjected to multi-element analysis. The assay results are shown in Table 16-20.

Table 16-20: Multi-element Analysis on Concentrate - Master Composite

Element	Unit	Data
Sb	ppm	696
AS	ppm	1184
Co	ppm	48
Cd	ppm	72
Bi	ppm	36
Hg	ppm	0.6
Ni	ppm	120
F	ppm	346
Se	ppm	72

Element	Unit	Data
P	ppm	230
SiO_2	%	9.84
CaO	%	0.54
Al_2O_3	%	3.31
MgO	%	0.48
MnO	%	0.02
Pb	%	0.92
Zn	%	0.42

The concentrates produced from various drill interval samples and composites were assayed for molybdenum, arsenic, and silver contents. The results are presented in Table 16-21 and Table 16-22.

Table 16-21: Multi-element Analysis on Concentrate - Drill Interval Samples

Sample ID	Content Mo (%)	As (%)	Ag (g/t)
MET 2	0.25	0.012	232
MET 3	0.40	0.376	306
MET 4	0.45	0.008	264
MET 5	0.05	0.002	156
MET 6	0.93	0.001	242
MET 7	0.06	0.013	204
MET 8	0.09	0.010	236
MET 9	0.12	0.008	420
MET 10	1.02	0.253	498
MET 11	0.17	0.077	260
MET 12	0.09	0.013	274
MET 13	0.03	0.056	230
MET 14	0.03	0.193	1118
MET 15	0.09	0.015	228
MET 16	0.05	<0.001	246
MET 17	0.26	0.082	180
MET 18	0.06	0.065	324

Sample ID	Content Mo (%)	As (%)	Ag (g/t)
MET 19	0.17	0.011	274
MET 20	0.43	0.017	282
MET 21	0.30	0.082	96
MET 22	0.76	0.093	184
MET 23	1.16	0.172	316
MET 24	0.12	0.016	192
MET 25	0.38	0.003	164
MET 26	0.86	0.017	140
MET 27	0.69	0.201	126
MET 28	0.31	0.032	190
MET 29	0.19	0.069	268
MET 30	0.02	0.374	178
MET 32	0.51	0.004	158
MET 33	0.06	0.050	402
MET 34	0.10	0.010	162
MET 35	0.29	0.014	100
MET 36	0.38	0.010	26

Table 16-22: Multi-element Analysis on Concentrate - Composite Samples

	Content		
Sample ID	Mo (%)	As (%)	Ag (g/t)
QSP 0-10	0.10	0.12	158
QSP 10-30	0.30	0.05	478
QSP 0-30	0.27	0.06	382
QSP LG 0-10	0.58	0.16	208
Hi Qtz 0-10	0.18	0.05	154
Hi Qtz 10-30	0.15	0.02	224
Hi Qtz 0-30	0.26	0.04	220
Prop 10-30	0.27	0.06	382
IARG 0-10	0.87	0.12	192
Master	0.58	0.16	316
Average	**0.36**	**0.08**	**271**

As indicated, the element contents in the drill interval samples varied significantly from sample to sample. The composite samples showed much less variation in the element contents. On average, the arsenic content should not attract smelting penalties by most smelters.

Molybdenum contents in some of the samples were high enough to consider producing a molybdenum concentrate, in particular from QSP LG 0-10 and IARG 0-10 composites.

16.4 **2009 G & T Testwork**

The following information was excerpted from a memo from G & T Metallurgical Services Ltd. Dated December 17, 2009 entitled "Metallurgical and Pilot Plant Testing on Samples from the Kerr-Sulphurets-Mitchell (KSM) Project-KM2344". The memo was prepared by Mr. John Folinsbee, P. Eng. and Mr. Tom Shouldice, P. Eng. (G & T, 2009).

"We received a shipment of 3218 coarse crushed samples for this project on March 10, 2009. The sample shipment had an estimated weight of about 12 tonnes. From these samples we constructed 12 composites, nine representing the Mitchell and three the Sulphurets Zones.

A standard Bond ball mill work index test was carried out on each composite. For the Mitchell Zone samples, the average Bond ball mill work index was 14.0 kWh/tonne. By comparison, the Sulphurets samples had an average Bond ball mill work index of 18.8 kWh/tonne. The Mitchell Zone samples can be considered to be of moderate hardness, whereas the Sulphurets samples can be classified as hard.

Open circuit flotation testing on all 12 samples produced average copper recoveries and grades to the copper concentrate that were similar to those produced in previous test programs. On average, the best open circuit tests on the nine Mitchell Zone composites produced an average copper recovery of 83 percent with 28 percent copper grade in the final concentrate. Previous testing on main ore types from the Mitchell Zone produced average copper concentrate recoveries of 85 percent at 25 percent copper grade in the final concentrate in open circuit tests.

Two pilot plant composites were constructed, one with low and one with high molybdenum content in the feed. A single locked cycle test was carried out on the low molybdenum content sample (Composite 1). The test (test 73) produced a copper recovery, to the copper concentrate, of 89 percent with 22.3 percent copper grade in the copper concentrate.

Gold recovery, to the copper concentrate, was 66 percent at a gold grade in the concentrate of 55 g/tonne. The copper cleaner tailing, and pyrite concentrate, were further processed in the standard CIL cyanidation flowsheet. Combined flotation plus cyanidation gold recoveries averaged about 80 percent.

Average copper recoveries and grades, to the copper concentrate, were lower in the pilot plant runs when compared to locked cycle test results. The main reasons for poorer performance in the pilot plant are related to control and circuit stability issues.

Insufficient material was available to overcome these problems. The locked cycle test data provides the best estimate of projected metallurgical performance.

Copper losses to the pyrite circuit can likely be reduced through better measurement and control of copper contents in the copper rougher tailing. This will require the use of a split rougher bank so that the copper circuit tailing can be more easily sampled.

Copper losses in the cleaner tailing may potentially be reduced through better control of pyrite flotation in the rougher circuit. The competing objectives of producing high gold recoveries in the copper concentrate while maintaining high copper grades can create stability issues in the cleaning circuit.

It is clear that additional pilot plant testing will be required to fully demonstrate the viability of the flowsheet on the basis of a continuous operation. There was not enough operating time in the pilot plant runs to fully explore the potential.

We were unable to produce a marketable grade final molybdenum concentrate from the bulk concentrate products in pilot plan runs P6 to P8. Multiple cleaning steps and passage of time appear to have caused the bulk concentrate to age. A single large scale cleaner test was carried out to produce fresh bulk concentrate. From this bulk concentrate, a final molybdenum concentrate assaying about 48 percent molybdenum was produced. The associated molybdenum recovery was about 30 percent. Further work will be required to establish the viability of the copper-molybdenum separation on the higher molybdenum feed material."

16.5 **2009 Lakefield SGS Testwork**

The following information was summarized from SGS Lakefield Mineral Services flotation test results that they have recently obtained (SGS Lakefield, 2010).

"During 2009, two other composites were composed from 2008 core samples and shipped to the SGS Lakefield laboratories, Ontario. These samples were used for HPGR testing and metallurgical verification of flotation and gold leaching results achieved by G & T Laboratory in British Columbia. One composite sample was prepared from Sulphurets core and the other sample was prepared from Mitchell core.

A grind series of flotation test work produced copper recoveries consistent with the G & T results reported previously. Carbon-in-Leach gold extractions of intermediate sulphide flotation products in the range of 70% were also achieved by SGS Mineral Services.

SGS also conducted two locked cycle tests on the composites from Mitchell and Sulphurets. The test results as shown in Table 16-23 are comparable with the data produced by G & T laboratory."

Table 16-23:
Locked Cycle Test Results - Mitchell and Sulphurets - SGS Laboratories

Product	Weight	Assays, %, g/t			Distribution, %		
	%	Cu	Au	Mo	Cu	Au	Mo
Mitchell							
3rd Cleaner Conc	0.80	23.1	53.7	0.41	89.0	59.6	59.6
Gold-Bearing Pyrite Conc	15.1	0.07	1.25	0.011	5.2	26.3	29.1
Pyrite Rougher Tails	84.1	0.01	0.12	0.001	5.8	14.1	11.3
Head	100.0	0.21	0.72	0.005	100.0	100.0	100.0
Sulphurets							
3rd Cleaner Conc	0.75	22.7	49.1	0.63	85.7	56.1	66.6
Gold-Bearing Pyrite Conc	17.3	0.08	1.31	0.008	6.7	34.3	20.3
Pyrite Rougher Tails	82.0	0.02	0.08	0.001	7.6	9.6	13.2
Head	100.0	0.200	0.66	0.007	100.0	100.0	100.0

It is the author’s opinion that the samples that have been chosen for completed and ongoing metallurgical testwork are representative.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Mineral Resources were estimated for the Mitchell deposit 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 and is a Certified Professional Geologist with the AIPG. 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 and Mitchell 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

The distribution of uncapped and capped raw gold assay grades is summarized at four different cutoff grades by selected lithologic and alteration types in Tables 17-1 through 17-6 for the Kerr, Sulphurets, and Mitchell deposits. Grade capping is discussed in Section 17.3.

As can be seen Tables 17-1 through 17-6 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. 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 (7%) to Sulphurets (26%) to Mitchell (43%). Another important statistical parameter is that the coefficient of variation (CV) decreases from 2.29 for uncapped Kerr assays to 0.98 for uncapped Mitchell gold assays. 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.

Gold is seen to be distributed in a number of logged lithologic and alteration types at Kerr and Sulphurets. In the Mitchell deposit, approximately 59% of the contained gold metal is contained in four lithologic units: VATF, VU, VULT, and VUTF, respectively. Contained gold metal is spread throughout a number of logged alteration types at Mitchell.

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 deposits. 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 (see Section 17.5).

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

Lithology	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	25,424	72%	0.22	5,672	36.8%	0.51	2.29	0.22	5,607	37.2%	0.37	1.66
	0.25	7,069	21%	0.51	3,587	31.2%	0.90	1.78	0.50	3,522	31.6%	0.60	1.21
	0.50	1,821	6%	1.00	1,816	16.7%	1.68	1.69	0.96	1,751	16.9%	1.06	1.10
	1.00	370	1%	2.35	870	15.3%	3.40	1.45	2.17	805	14.4%	1.89	0.87
DDAP	0.00	630	97%	0.07	41	58.6%	0.35	5.30	0.07	41	58.6%	0.35	5.30
	0.25	19	2%	0.91	17	10.2%	1.80	1.98	0.91	17	10.2%	1.80	1.98
	0.50	5	1%	2.59	13	7.0%	2.89	1.12	2.59	13	7.0%	2.89	1.12
	1.00	1	0%	7.25	10	24.2%	0.00	0.00	7.25	10	24.2%	0.00	0.00
DDPL	0.00	1,147	87%	0.15	173	50.6%	0.43	2.85	0.15	170	51.7%	0.31	2.09
	0.25	155	10%	0.55	86	20.3%	1.08	1.95	0.53	82	20.8%	0.71	1.35
	0.50	45	2%	1.13	50	9.4%	1.89	1.66	1.05	47	9.6%	1.18	1.12
	1.00	19	2%	1.83	34	19.7%	2.76	1.51	1.63	30	17.9%	1.64	1.01
DDRK	0.00	650	98%	0.04	29	53.4%	0.23	5.18	0.04	29	53.4%	0.23	5.18
	0.25	16	2%	0.82	13	13.5%	1.20	1.45	0.82	13	13.5%	1.20	1.45
	0.50	5	0%	2.10	9	0.0%	1.69	0.80	2.10	9	0.0%	1.69	0.80
	1.00	5	1%	2.10	9	33.0%	1.69	0.80	2.10	9	33.0%	1.69	0.80
FELS	0.00	1,010	66%	0.22	222	40.0%	0.16	0.72	0.22	222	40.0%	0.16	0.72
	0.25	347	30%	0.38	133	46.2%	0.16	0.41	0.38	133	46.2%	0.16	0.41
	0.50	46	4%	0.67	31	11.6%	0.25	0.38	0.67	31	11.6%	0.25	0.38
	1.00	3	0%	1.43	5	2.1%	0.44	0.31	1.43	5	2.1%	0.44	0.31
INPP	0.00	1,898	65%	0.27	510	28.9%	0.53	1.99	0.27	505	29.2%	0.47	1.78
	0.25	665	23%	0.55	363	30.0%	0.83	1.52	0.54	357	30.3%	0.72	1.34
	0.50	230	10%	0.91	210	23.9%	1.33	1.46	0.89	205	24.1%	1.14	1.28
	1.00	32	2%	2.72	88	17.3%	2.96	1.09	2.56	83	16.4%	2.44	0.95
SCNG	0.00	1,475	72%	0.21	304	44.1%	0.21	1.00	0.21	304	44.1%	0.21	1.00
	0.25	406	23%	0.42	170	36.9%	0.28	0.66	0.42	170	36.9%	0.28	0.66
	0.50	68	4%	0.85	58	12.3%	0.47	0.55	0.85	58	12.3%	0.47	0.55
	1.00	11	1%	1.78	20	6.7%	0.42	0.24	1.78	20	6.7%	0.42	0.24
SSED	0.00	4,167	80%	0.19	787	56.8%	0.24	1.27	0.19	787	56.8%	0.24	1.27
	0.25	838	17%	0.41	340	29.8%	0.46	1.13	0.41	340	29.8%	0.46	1.13
	0.50	113	2%	0.93	105	7.8%	1.10	1.18	0.93	105	7.8%	1.10	1.18
	1.00	19	0%	2.32	44	5.5%	2.21	0.95	2.32	44	5.5%	2.21	0.95
SSST	0.00	942	84%	0.17	156	62.9%	0.16	0.96	0.17	156	62.9%	0.16	0.96
	0.25	147	13%	0.39	58	24.8%	0.28	0.72	0.39	58	24.8%	0.28	0.72
	0.50	24	2%	0.82	19	7.7%	0.52	0.63	0.82	19	7.7%	0.52	0.63
	1.00	4	0%	1.97	7	4.7%	0.29	0.15	1.97	7	4.7%	0.29	0.15
VHLP	0.00	3,209	78%	0.21	674	39.7%	0.59	2.80	0.21	660	40.5%	0.50	2.43
	0.25	695	16%	0.58	406	25.7%	1.18	2.02	0.56	392	26.3%	0.99	1.75
	0.50	184	5%	1.27	233	13.9%	2.16	1.70	1.19	219	14.2%	1.77	1.48
	1.00	38	1%	3.70	139	20.6%	3.90	1.05	3.34	125	19.0%	3.08	0.92
VLTH	0.00	417	63%	0.42	177	14.9%	0.96	2.26	0.42	175	15.1%	0.90	2.14
	0.25	153	20%	0.99	150	16.6%	1.41	1.43	0.97	148	16.8%	1.31	1.35
	0.50	70	9%	1.72	121	16.0%	1.82	1.06	1.69	119	16.2%	1.66	0.98
	1.00	31	7%	2.99	93	52.5%	2.14	0.72	2.92	91	51.9%	1.87	0.64
VTLP	0.00	2,160	61%	0.25	538	28.3%	0.26	1.03	0.25	538	28.3%	0.26	1.03
	0.25	839	29%	0.46	386	40.1%	0.29	0.64	0.46	386	40.1%	0.29	0.64
	0.50	216	9%	0.79	170	22.6%	0.42	0.54	0.79	170	22.6%	0.42	0.54
	1.00	30	1%	1.62	48	9.0%	0.64	0.39	1.62	48	9.0%	0.64	0.39
VTUF	0.00	3,707	55%	0.27	989	29.3%	0.20	0.74	0.27	989	29.3%	0.20	0.74
	0.25	1,677	36%	0.42	699	45.5%	0.20	0.47	0.42	699	45.5%	0.20	0.47
	0.50	351	8%	0.71	250	20.3%	0.24	0.34	0.71	250	20.3%	0.24	0.34
	1.00	39	1%	1.25	49	5.0%	0.26	0.20	1.25	49	5.0%	0.26	0.20
VTXL	0.00	1,565	78%	0.23	356	35.9%	0.65	2.87	0.22	345	37.0%	0.46	2.10
	0.25	345	15%	0.66	228	21.4%	1.29	1.96	0.63	218	22.1%	0.86	1.37
	0.50	116	5%	1.31	152	14.7%	2.08	1.59	1.22	141	15.1%	1.30	1.07
	1.00	35	2%	2.81	100	28.0%	3.30	1.17	2.51	89	25.8%	1.76	0.70
VU	0.00	829	80%	0.19	155	61.6%	0.13	0.68	0.19	155	61.6%	0.13	0.68
	0.25	165	18%	0.36	59	30.6%	0.18	0.49	0.36	59	30.6%	0.18	0.49
	0.50	17	2%	0.73	12	4.4%	0.36	0.49	0.73	12	4.4%	0.36	0.49
	1.00	4	0%	1.32	5	3.4%	0.25	0.19	1.32	5	3.4%	0.25	0.19

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

Lithology	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	17,365	49%	0.44	7,680	12.3%	0.73	1.66	0.44	7,626	12.4%	0.68	1.54
	0.25	8,820	24%	0.76	6,738	19.4%	0.92	1.20	0.76	6,684	19.6%	0.83	1.10
	0.50	4,613	17%	1.14	5,245	27.5%	1.15	1.01	1.13	5,191	27.7%	1.02	0.91
	1.00	1,618	9%	1.94	3,135	40.8%	1.66	0.86	1.90	3,081	40.4%	1.41	0.74
ANDS	0.00	1,964	40%	0.47	916	11.3%	0.59	1.27	0.47	916	11.3%	0.59	1.27
	0.25	1,184	33%	0.69	813	24.8%	0.68	0.98	0.69	813	24.8%	0.68	0.98
	0.50	530	16%	1.11	586	24.9%	0.83	0.75	1.11	586	24.9%	0.83	0.75
	1.00	206	10%	1.74	358	39.1%	1.05	0.60	1.74	358	39.1%	1.05	0.60
DDRT	0.00	686	84%	0.12	85	34.1%	0.20	1.62	0.12	85	34.1%	0.20	1.62
	0.25	113	10%	0.50	56	26.9%	0.25	0.51	0.50	56	26.9%	0.25	0.51
	0.50	45	6%	0.74	33	31.8%	0.23	0.31	0.74	33	31.8%	0.23	0.31
	1.00	5	1%	1.25	6	7.1%	0.33	0.26	1.25	6	7.1%	0.33	0.26
PHBX	0.00	842	30%	0.86	726	4.1%	1.04	1.20	0.86	726	4.1%	1.04	1.20
	0.25	587	16%	1.19	696	6.8%	1.09	0.92	1.19	696	6.8%	1.09	0.92
	0.50	452	24%	1.43	647	20.4%	1.13	0.79	1.43	647	20.4%	1.13	0.79
	1.00	254	30%	1.97	499	68.8%	1.28	0.65	1.97	499	68.8%	1.28	0.65
PPFP	0.00	524	91%	0.10	54	59.0%	0.15	1.45	0.10	54	59.0%	0.15	1.45
	0.25	47	7%	0.47	22	22.6%	0.25	0.54	0.47	22	22.6%	0.25	0.54
	0.50	12	2%	0.83	10	13.3%	0.26	0.32	0.83	10	13.3%	0.26	0.32
	1.00	2	0%	1.36	3	5.0%	0.00	0.00	1.36	3	5.0%	0.00	0.00
PQMZ	0.00	1,039	38%	0.57	596	6.6%	0.84	1.47	0.57	596	6.6%	0.84	1.47
	0.25	640	23%	0.87	557	14.6%	0.96	1.10	0.87	557	14.6%	0.96	1.10
	0.50	404	25%	1.16	470	30.8%	1.10	0.95	1.16	470	30.8%	1.10	0.95
	1.00	149	14%	1.92	286	48.0%	1.54	0.80	1.92	286	48.0%	1.54	0.80
SEDS	0.00	576	63%	0.38	218	10.4%	0.68	1.80	0.38	218	10.4%	0.68	1.80
	0.25	212	13%	0.92	195	12.0%	0.89	0.97	0.92	195	12.0%	0.89	0.97
	0.50	138	14%	1.23	169	26.6%	0.97	0.79	1.23	169	26.6%	0.97	0.79
	1.00	59	10%	1.88	111	51.0%	1.20	0.64	1.88	111	51.0%	1.20	0.64
VAAT	0.00	1,728	57%	0.32	551	23.7%	0.34	1.06	0.32	551	23.7%	0.34	1.06
	0.25	740	26%	0.57	420	29.4%	0.39	0.69	0.57	420	29.4%	0.39	0.69
	0.50	290	13%	0.89	258	27.8%	0.47	0.52	0.89	258	27.8%	0.47	0.52
	1.00	67	4%	1.57	105	19.1%	0.52	0.33	1.57	105	19.1%	0.52	0.33
VATF	0.00	622	42%	0.59	364	10.2%	1.24	2.11	0.56	347	10.7%	0.92	1.65
	0.25	358	26%	0.91	327	15.4%	1.55	1.70	0.86	310	16.2%	1.11	1.29
	0.50	197	20%	1.38	271	25.3%	1.97	1.43	1.29	254	26.5%	1.36	1.06
	1.00	70	11%	2.55	179	49.1%	2.95	1.16	2.31	162	46.5%	1.89	0.82
VAXT	0.00	786	49%	0.37	294	18.4%	0.52	1.39	0.37	294	18.4%	0.52	1.39
	0.25	405	32%	0.59	240	28.9%	0.65	1.09	0.59	240	28.9%	0.65	1.09
	0.50	157	15%	0.99	155	28.6%	0.91	0.92	0.99	155	28.6%	0.91	0.92
	1.00	36	5%	1.96	71	24.1%	1.51	0.77	1.96	71	24.1%	1.51	0.77
VU	0.00	2,152	42%	0.43	926	12.2%	0.43	0.99	0.43	926	12.2%	0.43	0.99
	0.25	1,247	27%	0.65	813	22.2%	0.44	0.68	0.65	813	22.2%	0.44	0.68
	0.50	674	23%	0.90	607	37.5%	0.47	0.52	0.90	607	37.5%	0.47	0.52
	1.00	171	8%	1.52	260	28.1%	0.55	0.36	1.52	260	28.1%	0.55	0.36
VUAT	0.00	858	37%	0.45	385	12.2%	0.38	0.85	0.45	385	12.2%	0.38	0.85
	0.25	544	32%	0.62	338	25.4%	0.38	0.61	0.62	338	25.4%	0.38	0.61
	0.50	272	23%	0.88	240	34.9%	0.38	0.43	0.88	240	34.9%	0.38	0.43
	1.00	77	9%	1.38	106	27.5%	0.34	0.25	1.38	106	27.5%	0.34	0.25
VUTF	0.00	1,261	40%	0.49	620	9.3%	0.61	1.24	0.49	620	9.3%	0.61	1.24
	0.25	761	29%	0.74	562	21.2%	0.68	0.92	0.74	562	21.2%	0.68	0.92
	0.50	396	21%	1.09	431	30.7%	0.79	0.72	1.09	431	30.7%	0.79	0.72
	1.00	131	10%	1.84	240	38.7%	1.00	0.54	1.84	240	38.7%	1.00	0.54
VUXT	0.00	513	50%	0.41	210	16.1%	0.72	1.75	0.41	210	16.1%	0.72	1.75
	0.25	258	27%	0.68	176	22.3%	0.93	1.37	0.68	176	22.3%	0.93	1.37
	0.50	119	18%	1.08	130	29.5%	1.26	1.16	1.08	130	29.5%	1.26	1.16
	1.00	28	5%	2.40	68	32.2%	2.09	0.87	2.40	68	32.2%	2.09	0.87

Table 17-3: Distribution of Gold by Lithology - Mitchell

Lithology	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	46,872	31%	0.50	23,355	7.2%	0.49	0.98	0.49	23,194	7.2%	0.42	0.85
	0.25	32,259	26%	0.67	21,675	19.4%	0.50	0.74	0.67	21,514	19.5%	0.40	0.60
	0.50	19,956	33%	0.86	17,141	46.4%	0.56	0.65	0.85	16,980	46.7%	0.41	0.48
	1.00	4,462	10%	1.41	6,301	27.0%	0.96	0.68	1.38	6,140	26.5%	0.57	0.42
ANDS	0.00	2,175	50%	0.37	802	13.3%	0.58	1.57	0.36	779	13.7%	0.39	1.07
	0.25	1,090	21%	0.64	695	20.9%	0.72	1.12	0.62	673	21.5%	0.40	0.64
	0.50	632	25%	0.84	528	45.9%	0.89	1.07	0.80	505	47.2%	0.43	0.54
	1.00	94	4%	1.70	160	19.9%	2.09	1.23	1.46	137	17.6%	0.81	0.56
IVOL	0.00	2,489	26%	0.60	1,494	4.9%	0.67	1.11	0.59	1,459	5.0%	0.49	0.83
	0.25	1,840	19%	0.77	1,421	11.4%	0.70	0.90	0.75	1,386	11.6%	0.46	0.61
	0.50	1,376	42%	0.91	1,251	50.6%	0.76	0.83	0.88	1,217	51.8%	0.46	0.52
	1.00	332	13%	1.49	495	33.2%	1.37	0.92	1.39	461	31.6%	0.70	0.50
PMON	0.00	2,100	75%	0.18	387	22.1%	0.26	1.43	0.18	387	22.1%	0.26	1.43
	0.25	525	11%	0.57	302	22.4%	0.25	0.44	0.57	302	22.4%	0.25	0.44
	0.50	287	12%	0.75	215	43.7%	0.21	0.28	0.75	215	43.7%	0.21	0.28
	1.00	39	2%	1.15	45	11.7%	0.16	0.14	1.15	45	11.7%	0.16	0.14
SARG	0.00	1,143	77%	0.18	205	44.3%	0.18	0.98	0.18	205	44.3%	0.18	0.98
	0.25	268	19%	0.43	114	36.8%	0.19	0.45	0.43	114	36.8%	0.19	0.45
	0.50	52	4%	0.74	39	15.9%	0.23	0.31	0.74	39	15.9%	0.23	0.31
	1.00	5	0%	1.28	6	3.0%	0.18	0.14	1.28	6	3.0%	0.18	0.14
SCHT	0.00	610	16%	0.60	365	3.5%	0.33	0.55	0.60	365	3.5%	0.33	0.55
	0.25	515	23%	0.68	352	15.2%	0.28	0.41	0.68	352	15.2%	0.28	0.41
	0.50	373	54%	0.80	297	65.4%	0.25	0.31	0.80	297	65.4%	0.25	0.31
	1.00	43	7%	1.34	58	15.8%	0.26	0.19	1.34	58	15.8%	0.26	0.19
SEDS	0.00	1,015	65%	0.26	259	28.9%	0.37	1.46	0.25	258	29.1%	0.35	1.39
	0.25	355	25%	0.52	184	34.3%	0.53	1.03	0.51	183	34.5%	0.49	0.96
	0.50	97	7%	0.98	95	18.8%	0.85	0.87	0.97	94	18.9%	0.77	0.79
	1.00	25	2%	1.87	47	18.0%	1.32	0.70	1.81	45	17.5%	1.14	0.63
VALT	0.00	4,217	20%	0.58	2,453	4.6%	0.53	0.91	0.58	2,433	4.6%	0.44	0.77
	0.25	3,386	30%	0.69	2,341	18.9%	0.53	0.77	0.69	2,321	19.1%	0.43	0.62
	0.50	2,127	38%	0.88	1,876	45.7%	0.59	0.67	0.87	1,856	46.0%	0.44	0.50
	1.00	527	12%	1.43	756	30.8%	0.98	0.68	1.40	736	30.3%	0.60	0.43
VATF	0.00	6,767	21%	0.55	3,710	5.4%	0.39	0.71	0.55	3,710	5.4%	0.39	0.71
	0.25	5,368	30%	0.65	3,511	20.4%	0.37	0.56	0.65	3,511	20.4%	0.37	0.56
	0.50	3,314	39%	0.83	2,755	49.0%	0.36	0.44	0.83	2,755	49.0%	0.36	0.44
	1.00	701	10%	1.34	936	25.2%	0.48	0.36	1.34	936	25.2%	0.48	0.36
VAXT	0.00	1,841	25%	0.61	1,130	5.6%	0.44	0.71	0.61	1,130	5.6%	0.44	0.71
	0.25	1,372	19%	0.78	1,067	11.6%	0.39	0.50	0.78	1,067	11.6%	0.39	0.50
	0.50	1,024	37%	0.91	937	43.7%	0.36	0.39	0.91	937	43.7%	0.36	0.39
	1.00	341	19%	1.30	442	39.1%	0.35	0.27	1.30	442	39.1%	0.35	0.27
VU	0.00	11,447	24%	0.55	6,343	6.1%	0.46	0.82	0.55	6,319	6.1%	0.43	0.78
	0.25	8,689	28%	0.69	5,958	18.5%	0.45	0.65	0.68	5,933	18.5%	0.41	0.60
	0.50	5,516	37%	0.87	4,786	47.1%	0.47	0.54	0.86	4,761	47.3%	0.42	0.49
	1.00	1,297	11%	1.39	1,799	28.4%	0.73	0.52	1.37	1,775	28.1%	0.59	0.43
VULT	0.00	2,482	9%	0.61	1,517	2.0%	0.37	0.60	0.61	1,517	2.0%	0.37	0.60
	0.25	2,265	34%	0.66	1,487	21.3%	0.35	0.54	0.66	1,487	21.3%	0.35	0.54
	0.50	1,424	46%	0.82	1,164	51.3%	0.35	0.43	0.82	1,164	51.3%	0.35	0.43
	1.00	284	11%	1.35	385	25.4%	0.44	0.33	1.35	385	25.4%	0.44	0.33
VUTF	0.00	7,011	25%	0.51	3,559	7.3%	0.55	1.08	0.50	3,520	7.3%	0.39	0.78
	0.25	5,261	33%	0.63	3,301	23.8%	0.59	0.93	0.62	3,262	24.1%	0.39	0.62
	0.50	2,952	34%	0.83	2,454	45.7%	0.72	0.86	0.82	2,415	46.2%	0.41	0.51
	1.00	553	8%	1.50	827	23.2%	1.46	0.97	1.43	788	22.4%	0.62	0.44

Table 17-4: Distribution of Gold by Alteration - Kerr

Alteration	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	25,424	72%	0.22	5,672	36.8%	0.51	2.29	0.22	5,607	37.2%	0.37	1.66
	0.25	7,069	21%	0.51	3,587	31.2%	0.90	1.78	0.50	3,522	31.6%	0.60	1.21
	0.50	1,821	6%	1.00	1,816	16.7%	1.68	1.69	0.96	1,751	16.9%	1.06	1.10
	1.00	370	1%	2.35	870	15.3%	3.40	1.45	2.17	805	14.4%	1.89	0.87
CL	0.00	7,061	68%	0.22	1,573	33.0%	0.35	1.55	0.22	1,566	33.2%	0.31	1.40
	0.25	2,270	25%	0.46	1,054	37.5%	0.52	1.13	0.46	1,047	37.7%	0.45	0.98
	0.50	535	6%	0.87	464	18.2%	0.97	1.11	0.85	457	18.3%	0.81	0.95
	1.00	102	1%	1.74	177	11.3%	1.97	1.13	1.67	170	10.9%	1.59	0.95
IARG	0.00	471	79%	0.19	88	56.5%	0.15	0.81	0.19	88	56.5%	0.15	0.81
	0.25	100	18%	0.38	38	29.8%	0.22	0.57	0.38	38	29.8%	0.22	0.57
	0.50	16	3%	0.73	12	7.7%	0.36	0.49	0.73	12	7.7%	0.36	0.49
	1.00	4	1%	1.32	5	6.0%	0.25	0.19	1.32	5	6.0%	0.25	0.19
MY	0.00	1,964	84%	0.18	353	64.9%	0.28	1.55	0.18	353	64.9%	0.28	1.55
	0.25	314	14%	0.39	124	24.7%	0.64	1.62	0.39	124	24.7%	0.64	1.62
	0.50	35	2%	1.04	36	5.3%	1.78	1.71	1.04	36	5.3%	1.78	1.71
	1.00	5	0%	3.90	18	5.0%	3.91	1.00	3.90	18	5.0%	3.91	1.00
QP	0.00	493	14%	0.54	266	4.0%	0.65	1.21	0.53	262	4.1%	0.47	0.88
	0.25	424	40%	0.60	255	27.1%	0.68	1.13	0.59	251	27.5%	0.48	0.81
	0.50	228	41%	0.80	183	51.2%	0.88	1.09	0.79	179	51.9%	0.58	0.74
	1.00	26	5%	1.80	47	17.7%	2.34	1.30	1.65	43	16.5%	1.40	0.85
QS	0.00	3,666	75%	0.22	823	42.7%	0.44	1.98	0.22	813	43.2%	0.37	1.67
	0.25	903	18%	0.52	472	26.9%	0.82	1.57	0.51	462	27.2%	0.66	1.29
	0.50	237	5%	1.06	250	14.6%	1.47	1.39	1.02	241	14.7%	1.15	1.13
	1.00	55	1%	2.40	131	15.9%	2.65	1.11	2.22	121	14.9%	1.94	0.87
QSP	0.00	341	78%	0.18	61	61.3%	0.09	0.50	0.18	61	61.3%	0.09	0.50
	0.25	76	21%	0.31	24	36.1%	0.06	0.21	0.31	24	36.1%	0.06	0.21
	0.50	3	1%	0.50	2	2.6%	0.00	0.01	0.50	2	2.6%	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
SE	0.00	8,947	73%	0.21	1,853	40.4%	0.33	1.61	0.21	1,849	40.5%	0.32	1.54
	0.25	2,421	21%	0.46	1,104	34.7%	0.56	1.23	0.45	1,100	34.7%	0.53	1.16
	0.50	517	5%	0.89	462	15.0%	1.10	1.23	0.88	458	15.0%	1.03	1.16
	1.00	73	1%	2.51	184	9.9%	2.33	0.93	2.46	180	9.7%	2.12	0.86
SI	0.00	782	74%	0.38	295	15.8%	1.90	5.03	0.34	265	17.6%	0.80	2.35
	0.25	204	10%	1.22	248	8.9%	3.58	2.94	1.07	219	9.9%	1.30	1.21
	0.50	127	9%	1.74	222	16.5%	4.45	2.55	1.51	192	18.4%	1.49	0.98
	1.00	61	8%	2.86	173	58.7%	6.26	2.19	2.37	144	54.1%	1.79	0.75
UDEF	0.00	1,190	78%	0.23	275	22.1%	0.81	3.50	0.22	264	23.0%	0.61	2.77
	0.25	258	14%	0.83	214	22.0%	1.59	1.92	0.79	203	22.9%	1.15	1.45
	0.50	90	4%	1.71	154	12.5%	2.47	1.44	1.59	143	13.0%	1.67	1.05
	1.00	37	3%	3.19	119	43.4%	3.30	1.04	2.90	108	41.1%	1.93	0.67

Table 17-5: Distribution of Gold by Alteration - Sulphurets

Alteration	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	17,365	49%	0.44	7,680	12.3%	0.73	1.66	0.44	7,626	12.4%	0.68	1.54
	0.25	8,820	24%	0.76	6,738	19.4%	0.92	1.20	0.76	6,684	19.6%	0.83	1.10
	0.50	4,613	17%	1.14	5,245	27.5%	1.15	1.01	1.13	5,191	27.7%	1.02	0.91
	1.00	1,618	9%	1.94	3,135	40.8%	1.66	0.86	1.90	3,081	40.4%	1.41	0.74
CARB	0.00	238	91%	0.08	18	45.6%	0.14	1.83	0.08	18	45.6%	0.14	1.83
	0.25	22	8%	0.44	10	37.2%	0.18	0.41	0.44	10	37.2%	0.18	0.41
	0.50	4	2%	0.74	3	13.7%	0.21	0.28	0.74	3	13.7%	0.21	0.28
	1.00	1	0%	1.29	1	3.6%	0.00	0.00	1.29	1	3.6%	0.00	0.00
CL	0.00	1,041	59%	0.31	324	19.4%	0.47	1.50	0.31	324	19.4%	0.47	1.50
	0.25	426	25%	0.61	261	28.1%	0.61	1.00	0.61	261	28.1%	0.61	1.00
	0.50	164	11%	1.04	170	23.3%	0.82	0.79	1.04	170	23.3%	0.82	0.79
	1.00	53	5%	1.78	95	29.2%	1.10	0.62	1.78	95	29.2%	1.10	0.62
IARG	0.00	243	64%	0.22	52	27.6%	0.19	0.90	0.22	52	27.6%	0.19	0.90
	0.25	87	24%	0.43	38	38.5%	0.15	0.34	0.43	38	38.5%	0.15	0.34
	0.50	29	12%	0.62	18	33.9%	0.09	0.14	0.62	18	33.9%	0.09	0.14
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
KP	0.00	283	85%	0.15	44	47.5%	0.23	1.51	0.15	44	47.5%	0.23	1.51
	0.25	44	10%	0.53	23	22.2%	0.42	0.79	0.53	23	22.2%	0.42	0.79
	0.50	14	4%	0.94	13	15.0%	0.52	0.55	0.94	13	15.0%	0.52	0.55
	1.00	4	1%	1.64	7	15.3%	0.48	0.30	1.64	7	15.3%	0.48	0.30
PR	0.00	5,008	66%	0.25	1,276	27.9%	0.38	1.48	0.25	1,272	28.0%	0.35	1.38
	0.25	1,686	22%	0.55	921	29.8%	0.53	0.98	0.54	917	29.8%	0.48	0.89
	0.50	596	9%	0.91	541	24.9%	0.77	0.85	0.90	537	25.0%	0.67	0.74
	1.00	130	3%	1.72	223	17.4%	1.34	0.78	1.68	219	17.2%	1.10	0.65
PSBX	0.00	335	20%	0.64	214	5.0%	0.57	0.89	0.64	214	5.0%	0.57	0.89
	0.25	268	32%	0.76	204	18.6%	0.58	0.76	0.76	204	18.6%	0.58	0.76
	0.50	161	31%	1.02	164	34.3%	0.62	0.61	1.02	164	34.3%	0.62	0.61
	1.00	57	17%	1.58	90	42.1%	0.74	0.47	1.58	90	42.1%	0.74	0.47
QA	0.00	364	31%	0.53	193	8.6%	0.68	1.27	0.53	193	8.6%	0.68	1.27
	0.25	250	28%	0.71	177	19.2%	0.75	1.07	0.71	177	19.2%	0.75	1.07
	0.50	148	34%	0.94	140	46.1%	0.90	0.96	0.94	140	46.1%	0.90	0.96
	1.00	23	6%	2.23	51	26.2%	1.81	0.81	2.23	51	26.2%	1.81	0.81
QB	0.00	283	10%	0.78	221	2.1%	0.55	0.71	0.78	221	2.1%	0.55	0.71
	0.25	255	22%	0.85	216	11.3%	0.54	0.64	0.85	216	11.3%	0.54	0.64
	0.50	192	43%	1.00	191	38.0%	0.54	0.55	1.00	191	38.0%	0.54	0.55
	1.00	71	25%	1.51	107	48.6%	0.58	0.39	1.51	107	48.6%	0.58	0.39
QP	0.00	2,288	32%	0.60	1,367	7.9%	0.89	1.48	0.59	1,357	8.0%	0.83	1.39
	0.25	1,558	30%	0.81	1,259	17.9%	1.01	1.24	0.80	1,249	18.1%	0.93	1.16
	0.50	870	25%	1.16	1,014	29.4%	1.23	1.06	1.15	1,004	29.6%	1.13	0.98
	1.00	302	13%	2.02	611	44.7%	1.79	0.88	1.99	601	44.3%	1.59	0.80
SI	0.00	1,610	25%	0.85	1,371	3.8%	1.21	1.42	0.84	1,354	3.8%	1.06	1.26
	0.25	1,211	23%	1.09	1,320	10.1%	1.30	1.20	1.07	1,302	10.2%	1.13	1.05
	0.50	838	26%	1.41	1,181	22.5%	1.46	1.04	1.39	1,163	22.8%	1.24	0.89
	1.00	412	26%	2.12	873	63.6%	1.82	0.86	2.07	855	63.2%	1.47	0.71
SIH	0.00	1,415	40%	0.48	682	10.3%	0.61	1.28	0.48	680	10.3%	0.60	1.25
	0.25	849	26%	0.72	611	18.9%	0.70	0.97	0.72	610	18.9%	0.67	0.94
	0.50	488	24%	0.99	483	35.9%	0.82	0.83	0.99	481	36.0%	0.78	0.80
	1.00	142	10%	1.68	238	34.9%	1.26	0.75	1.67	236	34.8%	1.19	0.71
SIL	0.00	1,573	63%	0.30	470	22.7%	0.51	1.71	0.30	470	22.7%	0.51	1.71
	0.25	583	22%	0.62	363	25.4%	0.73	1.17	0.62	363	25.4%	0.73	1.17
	0.50	234	11%	1.04	244	25.4%	1.01	0.97	1.04	244	25.4%	1.01	0.97
	1.00	60	4%	2.10	125	26.6%	1.57	0.75	2.10	125	26.6%	1.57	0.75
UDEF	0.00	2,236	36%	0.58	1,300	6.8%	0.97	1.67	0.57	1,279	6.9%	0.85	1.49
	0.25	1,420	28%	0.85	1,211	17.4%	1.13	1.33	0.84	1,190	17.7%	0.97	1.16
	0.50	786	21%	1.25	985	25.5%	1.40	1.12	1.23	965	26.0%	1.17	0.95
	1.00	324	15%	2.01	653	50.2%	1.94	0.96	1.95	632	49.4%	1.55	0.79

Table 17-6: Distribution of Gold by Alteration - Mitchell

Alteration	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	46,872	31%	0.50	23,355	7.2%	0.49	0.98	0.49	23,194	7.2%	0.42	0.85
	0.25	32,259	26%	0.67	21,675	19.4%	0.50	0.74	0.67	21,514	19.5%	0.40	0.60
	0.50	19,956	33%	0.86	17,141	46.4%	0.56	0.65	0.85	16,980	46.7%	0.41	0.48
	1.00	4,462	10%	1.41	6,301	27.0%	0.96	0.68	1.38	6,140	26.5%	0.57	0.42
CL	0.00	8,192	18%	0.55	4,531	4.6%	0.42	0.77	0.55	4,511	4.6%	0.37	0.67
	0.25	6,694	31%	0.65	4,324	21.3%	0.41	0.64	0.64	4,305	21.4%	0.34	0.54
	0.50	4,130	41%	0.81	3,361	50.4%	0.45	0.55	0.81	3,341	50.6%	0.34	0.42
	1.00	802	10%	1.35	1,079	23.8%	0.79	0.58	1.32	1,060	23.5%	0.45	0.34
CL2	0.00	6,278	10%	0.59	3,690	3.1%	0.51	0.86	0.58	3,648	3.1%	0.37	0.64
	0.25	5,633	37%	0.63	3,576	23.5%	0.51	0.81	0.63	3,534	23.7%	0.36	0.58
	0.50	3,293	43%	0.82	2,711	50.0%	0.60	0.73	0.81	2,669	50.5%	0.38	0.46
	1.00	619	10%	1.40	867	23.5%	1.20	0.86	1.33	825	22.6%	0.58	0.43
IARG	0.00	7,160	28%	0.50	3,568	8.5%	0.38	0.77	0.50	3,564	8.6%	0.37	0.75
	0.25	5,132	27%	0.64	3,263	20.1%	0.37	0.58	0.64	3,259	20.1%	0.36	0.56
	0.50	3,170	37%	0.80	2,545	51.8%	0.38	0.47	0.80	2,541	51.9%	0.36	0.45
	1.00	503	7%	1.38	697	19.5%	0.64	0.46	1.38	692	19.4%	0.57	0.42
KP	0.00	871	48%	0.36	312	10.1%	0.40	1.12	0.36	312	10.1%	0.40	1.12
	0.25	451	26%	0.62	280	26.4%	0.40	0.65	0.62	280	26.4%	0.40	0.65
	0.50	227	19%	0.87	198	36.2%	0.44	0.50	0.87	198	36.2%	0.44	0.50
	1.00	59	7%	1.44	85	27.3%	0.49	0.34	1.44	85	27.3%	0.49	0.34
MTH	0.00	630	87%	0.11	70	54.1%	0.14	1.22	0.11	70	54.1%	0.14	1.22
	0.25	83	11%	0.39	32	34.0%	0.16	0.42	0.39	32	34.0%	0.16	0.42
	0.50	12	2%	0.72	8	8.9%	0.18	0.25	0.72	8	8.9%	0.18	0.25
	1.00	2	0%	1.04	2	3.0%	0.00	0.00	1.04	2	3.0%	0.00	0.00
PR	0.00	5,974	41%	0.41	2,469	10.1%	0.42	1.03	0.41	2,454	10.2%	0.37	0.91
	0.25	3,529	25%	0.63	2,219	22.5%	0.43	0.69	0.62	2,204	22.7%	0.34	0.55
	0.50	2,011	27%	0.83	1,663	44.8%	0.48	0.59	0.82	1,648	45.1%	0.34	0.42
	1.00	419	7%	1.33	557	22.6%	0.86	0.65	1.29	542	22.1%	0.45	0.35
QSP	0.00	6,802	23%	0.64	4,359	4.9%	0.67	1.04	0.63	4,301	4.9%	0.52	0.82
	0.25	5,260	23%	0.79	4,146	13.5%	0.69	0.88	0.78	4,088	13.7%	0.50	0.64
	0.50	3,682	37%	0.97	3,557	41.6%	0.76	0.79	0.95	3,499	42.2%	0.50	0.53
	1.00	1,169	17%	1.49	1,744	40.0%	1.18	0.79	1.44	1,685	39.2%	0.64	0.44
QSTW	0.00	3,509	13%	0.69	2,418	2.7%	0.53	0.77	0.68	2,403	2.7%	0.47	0.69
	0.25	3,036	23%	0.77	2,353	12.6%	0.52	0.67	0.77	2,338	12.7%	0.45	0.58
	0.50	2,220	46%	0.92	2,049	48.0%	0.53	0.58	0.92	2,033	48.3%	0.44	0.48
	1.00	623	18%	1.43	889	36.7%	0.79	0.55	1.40	873	36.3%	0.55	0.40
SIH	0.00	3,685	77%	0.19	697	38.3%	0.31	1.66	0.19	694	38.5%	0.30	1.59
	0.25	830	16%	0.52	430	28.8%	0.53	1.03	0.51	427	29.0%	0.50	0.97
	0.50	239	5%	0.96	229	17.4%	0.84	0.88	0.95	226	17.5%	0.77	0.81
	1.00	58	2%	1.84	107	15.4%	1.35	0.73	1.78	104	15.0%	1.19	0.67
SIL	0.00	1,427	71%	0.26	370	28.2%	0.37	1.44	0.26	367	28.4%	0.34	1.33
	0.25	421	12%	0.63	266	16.8%	0.52	0.82	0.62	263	16.9%	0.44	0.70
	0.50	244	14%	0.84	204	36.8%	0.60	0.72	0.82	201	37.1%	0.49	0.59
	1.00	45	3%	1.49	68	18.3%	1.16	0.78	1.42	64	17.6%	0.86	0.61
UDEF	0.00	638	63%	0.28	177	21.8%	0.32	1.14	0.28	177	21.8%	0.32	1.14
	0.25	234	19%	0.59	139	23.2%	0.33	0.56	0.59	139	23.2%	0.33	0.56
	0.50	115	13%	0.85	98	34.5%	0.29	0.35	0.85	98	34.5%	0.29	0.35
	1.00	29	5%	1.25	36	20.5%	0.28	0.22	1.25	36	20.5%	0.28	0.22

17.2 **Copper Grade Distribution**

The distribution of uncapped and capped raw copper assay grades is summarized at four different cutoff grades by selected lithologic and alteration types in Tables 17-7 through 17-12 for the Kerr, Sulphurets, and Mitchell deposits. Grade capping is discussed in Section 17.3.

As can be seen Tables 17-7 through 17-12 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 44% of the copper assays are above a 0.25% copper cutoff. These distributions decrease dramatically going northward, with Sulphurets at 21% and Mitchell at only 13% of the assays above a 0.25% cutoff grade. The CV decreases in going from Kerr (1.19) to Mitchell (0.88).

Like gold, copper is seen to be distributed in a number of logged lithologic and alteration types in the three deposits. 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. 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 to constrain the estimate of block copper grades (see Section 17.5).

Table 17-7: Distribution of Copper by Lithology - Kerr

Lithology	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	25,281	36%	0.32	8,103	4.4%	0.38	1.19	0.32	8,088	4.4%	0.37	1.17
	0.10	16,130	20%	0.48	7,745	10.8%	0.40	0.83	0.48	7,730	10.8%	0.39	0.80
	0.25	10,955	15%	0.63	6,868	14.6%	0.40	0.65	0.63	6,853	14.7%	0.39	0.62
	0.40	7,217	29%	0.79	5,682	70.1%	0.41	0.53	0.79	5,666	70.1%	0.39	0.50
DDAP	0.00	630	81%	0.06	39	30.3%	0.11	1.80	0.06	39	30.3%	0.11	1.80
	0.10	117	14%	0.23	27	35.5%	0.17	0.73	0.23	27	35.5%	0.17	0.73
	0.25	28	2%	0.47	13	11.1%	0.20	0.42	0.47	13	11.1%	0.20	0.42
	0.40	14	2%	0.64	9	23.2%	0.15	0.23	0.64	9	23.2%	0.15	0.23
DDPL	0.00	1,147	63%	0.15	173	12.9%	0.28	1.83	0.15	172	13.0%	0.26	1.72
	0.10	422	18%	0.36	151	19.7%	0.37	1.04	0.35	150	19.9%	0.34	0.95
	0.25	217	10%	0.54	117	19.8%	0.45	0.83	0.53	115	20.0%	0.39	0.74
	0.40	105	9%	0.78	82	47.5%	0.54	0.69	0.77	81	47.1%	0.46	0.59
DDRK	0.00	650	92%	0.03	18	38.1%	0.09	3.07	0.03	18	38.1%	0.09	3.07
	0.10	50	6%	0.23	11	34.4%	0.23	0.99	0.23	11	34.4%	0.23	0.99
	0.25	10	1%	0.53	5	10.8%	0.38	0.72	0.53	5	10.8%	0.38	0.72
	0.40	3	1%	0.93	3	16.7%	0.42	0.45	0.93	3	16.7%	0.42	0.45
FELS	0.00	1,010	37%	0.31	314	4.6%	0.34	1.08	0.31	314	4.6%	0.34	1.08
	0.10	640	18%	0.47	299	9.4%	0.33	0.71	0.47	299	9.4%	0.33	0.71
	0.25	459	15%	0.59	270	15.0%	0.32	0.54	0.59	270	15.0%	0.32	0.54
	0.40	310	31%	0.72	223	70.9%	0.31	0.43	0.72	223	70.9%	0.31	0.43
INPP	0.00	1,898	23%	0.54	1,030	1.9%	0.52	0.96	0.54	1,029	1.9%	0.52	0.95
	0.10	1,468	16%	0.69	1,011	5.3%	0.50	0.73	0.69	1,010	5.3%	0.50	0.73
	0.25	1,155	13%	0.83	956	7.5%	0.48	0.58	0.83	955	7.5%	0.48	0.58
	0.40	914	48%	0.96	879	85.3%	0.46	0.47	0.96	878	85.3%	0.45	0.47
SCNG	0.00	1,475	45%	0.19	282	10.7%	0.19	0.99	0.19	282	10.7%	0.19	0.99
	0.10	818	27%	0.31	252	23.3%	0.19	0.60	0.31	252	23.3%	0.19	0.60
	0.25	426	14%	0.44	186	23.3%	0.17	0.39	0.44	186	23.3%	0.17	0.39
	0.40	215	15%	0.56	120	42.7%	0.16	0.29	0.56	120	42.7%	0.16	0.29
SMUD	0.00	387	94%	0.06	25	43.9%	0.50	7.84	0.05	19	56.0%	0.16	3.28
	0.10	23	4%	0.62	14	8.1%	1.99	3.24	0.38	8	10.4%	0.58	1.55
	0.25	7	1%	1.69	12	3.3%	3.33	1.97	0.93	6	4.2%	0.81	0.88
	0.40	4	1%	2.76	11	44.8%	4.09	1.48	1.42	6	29.4%	0.76	0.54
SSED	0.00	4,167	38%	0.22	917	8.5%	0.23	1.03	0.22	916	8.5%	0.22	1.02
	0.10	2,591	30%	0.32	839	22.2%	0.23	0.71	0.32	838	22.2%	0.23	0.70
	0.25	1,339	16%	0.47	635	22.7%	0.23	0.49	0.47	635	22.7%	0.23	0.48
	0.40	682	16%	0.63	427	46.6%	0.24	0.38	0.63	427	46.6%	0.23	0.37
SSST	0.00	942	52%	0.15	143	14.2%	0.16	1.04	0.15	143	14.2%	0.16	1.04
	0.10	456	25%	0.27	122	28.0%	0.15	0.57	0.27	122	28.0%	0.15	0.57
	0.25	218	17%	0.38	82	35.8%	0.16	0.41	0.38	82	35.8%	0.16	0.41
	0.40	57	6%	0.55	31	22.0%	0.22	0.39	0.55	31	22.0%	0.22	0.39
VHLP	0.00	3,209	34%	0.29	939	6.3%	0.28	0.97	0.29	939	6.3%	0.28	0.97
	0.10	2,121	22%	0.41	880	13.4%	0.28	0.67	0.41	880	13.4%	0.28	0.67
	0.25	1,402	16%	0.54	754	16.5%	0.27	0.49	0.54	754	16.5%	0.27	0.49
	0.40	901	28%	0.67	600	63.9%	0.25	0.38	0.67	600	63.9%	0.25	0.38
VTLP	0.00	2,160	16%	0.48	1,038	1.5%	0.45	0.94	0.48	1,031	1.5%	0.43	0.91
	0.10	1,820	22%	0.56	1,022	8.4%	0.45	0.80	0.56	1,016	8.4%	0.43	0.76
	0.25	1,335	18%	0.70	935	12.0%	0.45	0.64	0.70	929	12.1%	0.42	0.60
	0.40	941	44%	0.86	810	78.1%	0.44	0.51	0.85	804	77.9%	0.40	0.47
VTUF	0.00	3,707	7%	0.53	1,968	0.7%	0.38	0.72	0.53	1,968	0.7%	0.38	0.72
	0.10	3,442	13%	0.57	1,954	4.5%	0.37	0.65	0.57	1,954	4.5%	0.37	0.65
	0.25	2,952	24%	0.63	1,866	14.6%	0.36	0.57	0.63	1,866	14.7%	0.36	0.57
	0.40	2,071	56%	0.76	1,578	80.2%	0.36	0.47	0.76	1,578	80.2%	0.36	0.47
VTXL	0.00	1,565	44%	0.23	368	5.4%	0.29	1.21	0.23	368	5.4%	0.29	1.21
	0.10	884	22%	0.39	348	15.6%	0.29	0.75	0.39	348	15.6%	0.29	0.75
	0.25	546	15%	0.53	290	19.6%	0.30	0.56	0.53	290	19.6%	0.30	0.56
	0.40	316	20%	0.69	218	59.3%	0.30	0.44	0.69	218	59.3%	0.30	0.44
VU	0.00	829	26%	0.32	265	4.4%	0.29	0.92	0.32	265	4.4%	0.29	0.92
	0.10	613	27%	0.41	253	14.2%	0.29	0.70	0.41	253	14.2%	0.29	0.70
	0.25	389	14%	0.55	216	14.2%	0.27	0.49	0.55	216	14.2%	0.27	0.49
	0.40	271	33%	0.66	178	67.3%	0.27	0.40	0.66	178	67.3%	0.27	0.40

Table 17-8: Distribution of Copper by Lithology - Sulphurets

Lithology	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	16,897	36%	0.16	2,766	4.6%	0.22	1.37	0.16	2,757	4.6%	0.22	1.33
	0.05	10,857	20%	0.24	2,640	8.7%	0.25	1.01	0.24	2,631	8.8%	0.24	0.97
	0.10	7,425	23%	0.32	2,398	22.4%	0.26	0.81	0.32	2,389	22.5%	0.25	0.77
	0.25	3,542	21%	0.50	1,778	64.3%	0.28	0.56	0.50	1,769	64.2%	0.26	0.52
ANDS	0.00	2,170	17%	0.18	402	2.2%	0.21	1.14	0.18	402	2.2%	0.21	1.14
	0.05	1,805	32%	0.22	393	10.8%	0.22	1.00	0.22	393	10.8%	0.22	1.00
	0.10	1,102	27%	0.32	349	24.0%	0.23	0.72	0.32	349	24.0%	0.23	0.72
	0.25	523	24%	0.48	253	63.0%	0.23	0.48	0.48	253	63.0%	0.23	0.48
DDRT	0.00	684	80%	0.04	27	24.8%	0.08	1.97	0.04	27	24.8%	0.08	1.97
	0.05	140	11%	0.14	20	19.7%	0.12	0.84	0.14	20	19.7%	0.12	0.84
	0.10	64	7%	0.23	15	28.0%	0.13	0.58	0.23	15	28.0%	0.13	0.58
	0.25	18	3%	0.41	7	27.5%	0.11	0.28	0.41	7	27.5%	0.11	0.28
PHBX	0.00	783	47%	0.10	78	10.7%	0.15	1.55	0.10	78	10.7%	0.15	1.55
	0.05	419	25%	0.17	69	16.9%	0.19	1.12	0.17	69	16.9%	0.19	1.12
	0.10	223	20%	0.25	56	30.2%	0.22	0.88	0.25	56	30.2%	0.22	0.88
	0.25	69	9%	0.47	33	42.1%	0.28	0.60	0.47	33	42.1%	0.28	0.60
PPFP	0.00	538	68%	0.06	31	24.1%	0.07	1.29	0.06	31	24.1%	0.07	1.29
	0.05	171	12%	0.14	24	14.3%	0.09	0.63	0.14	24	14.3%	0.09	0.63
	0.10	106	16%	0.18	19	40.1%	0.09	0.49	0.18	19	40.1%	0.09	0.49
	0.25	20	4%	0.33	7	21.4%	0.08	0.25	0.33	7	21.4%	0.08	0.25
PQMZ	0.00	863	30%	0.27	235	2.5%	0.27	0.98	0.27	235	2.5%	0.27	0.98
	0.05	608	8%	0.38	229	1.9%	0.25	0.67	0.38	229	1.9%	0.25	0.67
	0.10	542	19%	0.41	225	12.0%	0.24	0.58	0.41	225	12.0%	0.24	0.58
	0.25	380	44%	0.52	196	83.7%	0.22	0.42	0.52	196	83.7%	0.22	0.42
SEDS	0.00	576	59%	0.11	62	9.7%	0.18	1.70	0.11	62	9.7%	0.18	1.70
	0.05	236	14%	0.24	56	9.0%	0.23	0.97	0.24	56	9.0%	0.23	0.97
	0.10	158	16%	0.32	50	23.1%	0.24	0.76	0.32	50	23.1%	0.24	0.76
	0.25	67	12%	0.54	36	58.2%	0.23	0.43	0.54	36	58.2%	0.23	0.43
VAAT	0.00	1,657	28%	0.14	236	4.2%	0.17	1.17	0.14	236	4.2%	0.17	1.17
	0.05	1,188	27%	0.19	226	13.8%	0.18	0.92	0.19	226	13.8%	0.18	0.92
	0.10	737	28%	0.26	194	30.4%	0.19	0.72	0.26	194	30.4%	0.19	0.72
	0.25	281	17%	0.43	122	51.5%	0.21	0.48	0.43	122	51.5%	0.21	0.48
VAXT	0.00	719	29%	0.16	117	4.5%	0.18	1.14	0.16	117	4.5%	0.18	1.14
	0.05	512	23%	0.22	112	9.7%	0.19	0.88	0.22	112	9.7%	0.19	0.88
	0.10	350	28%	0.29	100	27.1%	0.20	0.69	0.29	100	27.1%	0.20	0.69
	0.25	148	21%	0.46	69	58.7%	0.19	0.41	0.46	69	58.7%	0.19	0.41
VU	0.00	2,152	19%	0.27	575	1.5%	0.29	1.07	0.27	574	1.5%	0.28	1.07
	0.05	1,737	15%	0.33	566	4.3%	0.29	0.89	0.33	565	4.3%	0.29	0.88
	0.10	1,420	28%	0.38	541	17.7%	0.29	0.77	0.38	540	17.7%	0.29	0.76
	0.25	812	38%	0.54	439	76.5%	0.30	0.55	0.54	439	76.4%	0.29	0.54
VUAT	0.00	858	37%	0.17	149	5.6%	0.21	1.23	0.17	149	5.6%	0.21	1.23
	0.05	543	17%	0.26	141	6.9%	0.23	0.89	0.26	141	6.9%	0.23	0.89
	0.10	397	23%	0.33	131	20.6%	0.23	0.71	0.33	131	20.6%	0.23	0.71
	0.25	202	23%	0.50	100	67.0%	0.22	0.44	0.50	100	67.0%	0.22	0.44
VUTF	0.00	1,220	37%	0.17	202	5.0%	0.24	1.43	0.16	200	5.0%	0.22	1.36
	0.05	768	20%	0.25	192	9.4%	0.26	1.06	0.25	190	9.5%	0.24	0.99
	0.10	520	21%	0.33	173	20.4%	0.29	0.86	0.33	171	20.5%	0.26	0.79
	0.25	261	21%	0.51	132	65.3%	0.32	0.63	0.50	130	65.0%	0.27	0.55
VUXT	0.00	513	44%	0.15	79	7.0%	0.21	1.40	0.15	79	7.0%	0.21	1.40
	0.05	286	18%	0.26	73	8.7%	0.24	0.95	0.26	73	8.7%	0.24	0.95
	0.10	192	16%	0.35	66	16.7%	0.25	0.73	0.35	66	16.7%	0.25	0.73
	0.25	109	21%	0.49	53	67.5%	0.25	0.51	0.49	53	67.5%	0.25	0.51

Table 17-9: Distribution of Copper by Lithology – Mitchell

Lithology	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	46,869	17%	0.15	6,819	2.7%	0.13	0.88	0.14	6,766	2.7%	0.12	0.80
	0.05	38,966	23%	0.17	6,637	12.0%	0.13	0.74	0.17	6,584	12.1%	0.11	0.66
	0.10	28,267	47%	0.21	5,819	52.1%	0.13	0.64	0.20	5,766	52.5%	0.11	0.55
	0.25	6,235	13%	0.36	2,266	33.2%	0.20	0.55	0.35	2,213	32.7%	0.15	0.42
ANDS	0.00	2,175	7%	0.16	340	0.7%	0.12	0.80	0.16	338	0.7%	0.12	0.76
	0.05	2,030	29%	0.17	338	15.0%	0.12	0.74	0.17	336	15.1%	0.12	0.70
	0.10	1,395	49%	0.21	287	47.6%	0.13	0.63	0.20	285	47.9%	0.12	0.59
	0.25	329	15%	0.38	125	36.7%	0.16	0.43	0.37	123	36.4%	0.14	0.37
IVOL	0.00	2,489	9%	0.16	389	1.7%	0.10	0.61	0.16	389	1.7%	0.10	0.61
	0.05	2,266	25%	0.17	382	12.1%	0.09	0.54	0.17	382	12.1%	0.09	0.54
	0.10	1,647	51%	0.20	335	55.0%	0.08	0.40	0.20	335	55.0%	0.08	0.40
	0.25	377	15%	0.32	121	31.2%	0.07	0.23	0.32	121	31.2%	0.07	0.23
PMON	0.00	2,100	35%	0.11	222	8.2%	0.10	0.91	0.11	222	8.2%	0.10	0.91
	0.05	1,363	23%	0.15	204	16.1%	0.09	0.63	0.15	204	16.1%	0.09	0.63
	0.10	878	34%	0.19	168	50.9%	0.09	0.48	0.19	168	50.9%	0.09	0.48
	0.25	155	7%	0.35	55	24.8%	0.10	0.27	0.35	55	24.8%	0.10	0.27
PPFP	0.00	475	63%	0.05	24	15.6%	0.07	1.32	0.05	24	15.6%	0.07	1.32
	0.05	176	21%	0.11	20	30.5%	0.07	0.62	0.11	20	30.5%	0.07	0.62
	0.10	78	15%	0.16	13	44.1%	0.08	0.49	0.16	13	44.1%	0.08	0.49
	0.25	7	1%	0.35	2	9.8%	0.13	0.37	0.35	2	9.8%	0.13	0.37
SARG	0.00	1,143	49%	0.07	74	21.3%	0.06	0.93	0.07	74	21.3%	0.06	0.93
	0.05	578	34%	0.10	59	38.1%	0.07	0.65	0.10	59	38.1%	0.07	0.65
	0.10	189	15%	0.16	30	33.2%	0.09	0.56	0.16	30	33.2%	0.09	0.56
	0.25	13	1%	0.41	6	7.5%	0.16	0.38	0.41	6	7.5%	0.16	0.38
SEDS	0.00	1,015	64%	0.07	70	15.9%	0.10	1.46	0.07	70	15.9%	0.10	1.46
	0.05	366	13%	0.16	59	13.9%	0.12	0.75	0.16	59	13.9%	0.12	0.75
	0.10	230	17%	0.21	49	38.3%	0.12	0.58	0.21	49	38.3%	0.12	0.58
	0.25	60	6%	0.38	22	31.9%	0.14	0.38	0.38	22	31.9%	0.14	0.37
VALT	0.00	4,217	4%	0.16	655	0.9%	0.08	0.53	0.16	655	0.9%	0.08	0.53
	0.05	4,029	23%	0.16	649	11.9%	0.08	0.50	0.16	649	11.9%	0.08	0.50
	0.10	3,040	59%	0.19	570	60.8%	0.07	0.40	0.19	570	60.8%	0.07	0.40
	0.25	550	13%	0.31	172	26.3%	0.06	0.20	0.31	172	26.3%	0.06	0.20
VATF	0.00	6,767	13%	0.15	1,014	1.7%	0.10	0.68	0.15	1,014	1.7%	0.10	0.68
	0.05	5,901	21%	0.17	998	10.7%	0.09	0.56	0.17	998	10.7%	0.09	0.56
	0.10	4,508	52%	0.20	889	55.9%	0.09	0.46	0.20	889	55.9%	0.09	0.46
	0.25	956	14%	0.34	322	31.7%	0.09	0.28	0.34	322	31.7%	0.09	0.28
VAXT	0.00	1,841	21%	0.17	304	2.3%	0.12	0.75	0.17	304	2.3%	0.12	0.75
	0.05	1,462	15%	0.20	297	6.8%	0.11	0.55	0.20	297	6.8%	0.11	0.55
	0.10	1,187	41%	0.23	276	41.8%	0.10	0.44	0.23	276	41.8%	0.10	0.44
	0.25	441	24%	0.34	149	49.0%	0.09	0.26	0.34	149	49.0%	0.09	0.26
VU	0.00	11,447	14%	0.14	1,610	2.7%	0.10	0.68	0.14	1,607	2.7%	0.09	0.66
	0.05	9,896	25%	0.16	1,567	13.5%	0.09	0.57	0.16	1,564	13.5%	0.09	0.55
	0.10	7,019	50%	0.19	1,351	58.0%	0.09	0.45	0.19	1,348	58.1%	0.08	0.43
	0.25	1,263	11%	0.33	417	25.9%	0.11	0.33	0.33	414	25.7%	0.09	0.27
VULT	0.00	2,482	3%	0.19	477	0.4%	0.11	0.59	0.19	477	0.4%	0.11	0.59
	0.05	2,419	15%	0.20	475	6.2%	0.11	0.57	0.20	475	6.2%	0.11	0.57
	0.10	2,041	62%	0.22	445	55.1%	0.11	0.50	0.22	445	55.1%	0.11	0.50
	0.25	508	20%	0.36	183	38.3%	0.12	0.34	0.36	183	38.3%	0.12	0.34
VUTF	0.00	7,011	15%	0.16	1,099	2.8%	0.18	1.16	0.15	1,063	2.9%	0.14	0.93
	0.05	5,927	25%	0.18	1,068	12.4%	0.19	1.04	0.17	1,033	12.8%	0.14	0.82
	0.10	4,148	44%	0.22	933	44.1%	0.21	0.93	0.22	897	45.6%	0.15	0.70
	0.25	1,065	15%	0.42	448	40.8%	0.34	0.80	0.39	413	38.8%	0.21	0.55

Table 17-10: Distribution of Copper by Alteration - Kerr

Alteration	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	25,281	36%	0.32	8,103	4.4%	0.38	1.19	0.32	8,088	4.4%	0.37	1.17
	0.10	16,130	20%	0.48	7,745	10.8%	0.40	0.83	0.48	7,730	10.8%	0.39	0.80
	0.25	10,955	15%	0.63	6,868	14.6%	0.40	0.65	0.63	6,853	14.7%	0.39	0.62
	0.40	7,217	29%	0.79	5,682	70.1%	0.41	0.53	0.79	5,666	70.1%	0.39	0.50
CL	0.00	7,061	27%	0.41	2,892	1.9%	0.41	0.99	0.41	2,891	1.9%	0.41	0.99
	0.10	5,139	15%	0.55	2,836	6.4%	0.39	0.71	0.55	2,835	6.4%	0.39	0.71
	0.25	4,100	17%	0.65	2,651	13.4%	0.38	0.59	0.65	2,651	13.4%	0.38	0.59
	0.40	2,904	41%	0.78	2,265	78.3%	0.38	0.49	0.78	2,264	78.3%	0.38	0.48
IARG	0.00	471	33%	0.26	122	7.9%	0.27	1.05	0.26	122	7.9%	0.27	1.05
	0.10	314	32%	0.36	113	19.6%	0.29	0.80	0.36	113	19.6%	0.29	0.80
	0.25	165	13%	0.54	89	15.0%	0.30	0.55	0.54	89	15.0%	0.30	0.55
	0.40	105	22%	0.67	71	57.6%	0.30	0.45	0.67	71	57.6%	0.30	0.45
MY	0.00	1,964	43%	0.21	405	10.8%	0.24	1.18	0.21	404	10.8%	0.24	1.17
	0.10	1,128	32%	0.32	361	25.1%	0.27	0.84	0.32	361	25.2%	0.26	0.83
	0.25	501	12%	0.52	260	18.4%	0.30	0.58	0.52	259	18.5%	0.29	0.57
	0.40	268	14%	0.69	185	45.6%	0.32	0.47	0.69	184	45.6%	0.31	0.45
QP	0.00	493	3%	1.09	536	0.1%	0.65	0.60	1.07	527	0.1%	0.59	0.55
	0.10	480	4%	1.11	535	0.7%	0.64	0.57	1.10	527	0.7%	0.57	0.52
	0.25	459	4%	1.16	532	1.0%	0.62	0.54	1.14	523	1.0%	0.55	0.48
	0.40	441	90%	1.19	526	98.3%	0.61	0.51	1.17	518	98.2%	0.53	0.46
QS	0.00	3,666	35%	0.28	1,019	5.6%	0.31	1.13	0.28	1,019	5.6%	0.31	1.13
	0.10	2,369	24%	0.41	963	14.6%	0.33	0.80	0.41	963	14.6%	0.33	0.80
	0.25	1,477	16%	0.55	814	18.4%	0.34	0.61	0.55	814	18.4%	0.34	0.61
	0.40	881	24%	0.71	626	61.4%	0.36	0.50	0.71	626	61.4%	0.36	0.50
QSP	0.00	341	34%	0.27	91	5.6%	0.27	1.01	0.27	91	5.6%	0.27	1.01
	0.10	224	27%	0.38	86	16.0%	0.27	0.69	0.38	86	16.0%	0.27	0.69
	0.25	132	9%	0.54	71	10.7%	0.24	0.45	0.54	71	10.7%	0.24	0.45
	0.40	101	30%	0.61	61	67.7%	0.23	0.39	0.61	61	67.7%	0.23	0.39
SE	0.00	8,816	33%	0.31	2,721	4.9%	0.34	1.09	0.31	2,720	4.9%	0.34	1.09
	0.10	5,926	24%	0.44	2,588	13.3%	0.35	0.79	0.44	2,587	13.3%	0.34	0.79
	0.25	3,813	17%	0.58	2,226	17.5%	0.35	0.60	0.58	2,225	17.5%	0.35	0.60
	0.40	2,303	26%	0.76	1,751	64.4%	0.35	0.46	0.76	1,750	64.3%	0.35	0.46
SI	0.00	782	82%	0.13	104	20.4%	0.44	3.29	0.13	99	21.5%	0.29	2.27
	0.10	138	6%	0.60	83	6.5%	0.91	1.51	0.56	78	6.9%	0.49	0.86
	0.25	91	3%	0.84	76	6.0%	1.04	1.24	0.78	71	6.3%	0.47	0.60
	0.40	71	9%	0.98	70	67.1%	1.13	1.16	0.90	65	65.3%	0.45	0.50
UDEF	0.00	1,179	80%	0.12	138	14.2%	0.30	2.60	0.12	138	14.2%	0.30	2.60
	0.10	235	10%	0.50	119	12.7%	0.53	1.05	0.50	119	12.7%	0.53	1.05
	0.25	121	3%	0.84	101	7.4%	0.56	0.67	0.84	101	7.4%	0.56	0.67
	0.40	88	7%	1.03	91	65.7%	0.54	0.52	1.03	91	65.7%	0.54	0.52

Table 17-11: Distribution of Copper by Alteration - Sulphurets

Alteration	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	16,897	36%	0.16	2,766	4.6%	0.22	1.37	0.16	2,757	4.6%	0.22	1.33
	0.05	10,857	20%	0.24	2,640	8.7%	0.25	1.01	0.24	2,631	8.8%	0.24	0.97
	0.10	7,425	23%	0.32	2,398	22.4%	0.26	0.81	0.32	2,389	22.5%	0.25	0.77
	0.25	3,542	21%	0.50	1,778	64.3%	0.28	0.56	0.50	1,769	64.2%	0.26	0.52
CARB	0.00	238	89%	0.02	6	36.6%	0.05	1.86	0.02	6	36.6%	0.05	1.86
	0.05	27	4%	0.14	4	14.2%	0.06	0.42	0.14	4	14.2%	0.06	0.42
	0.10	17	7%	0.17	3	49.2%	0.05	0.30	0.17	3	49.2%	0.05	0.30
	0.25	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
CL	0.00	1,041	32%	0.16	171	3.0%	0.18	1.11	0.16	171	3.0%	0.18	1.11
	0.05	711	14%	0.23	166	6.5%	0.18	0.79	0.23	166	6.5%	0.18	0.79
	0.10	565	32%	0.27	155	31.3%	0.19	0.68	0.27	155	31.3%	0.19	0.68
	0.25	230	22%	0.44	101	59.2%	0.19	0.43	0.44	101	59.2%	0.19	0.43
IARG	0.00	243	39%	0.10	25	5.9%	0.11	1.06	0.10	25	5.9%	0.11	1.06
	0.05	148	24%	0.16	24	18.8%	0.11	0.67	0.16	24	18.8%	0.11	0.67
	0.10	89	28%	0.21	19	45.0%	0.11	0.53	0.21	19	45.0%	0.11	0.53
	0.25	21	9%	0.37	8	30.3%	0.12	0.33	0.37	8	30.3%	0.12	0.33
KP	0.00	280	47%	0.09	25	11.6%	0.09	1.00	0.09	25	11.6%	0.09	1.00
	0.05	149	20%	0.15	22	16.8%	0.09	0.58	0.15	22	16.8%	0.09	0.58
	0.10	92	24%	0.19	18	42.0%	0.08	0.41	0.19	18	42.0%	0.08	0.41
	0.25	24	9%	0.31	7	29.6%	0.04	0.12	0.31	7	29.6%	0.04	0.12
PR	0.00	4,941	40%	0.11	534	8.4%	0.13	1.18	0.11	534	8.4%	0.13	1.18
	0.05	2,957	24%	0.17	489	16.4%	0.14	0.83	0.17	489	16.4%	0.14	0.83
	0.10	1,747	24%	0.23	402	35.2%	0.15	0.64	0.23	402	35.2%	0.15	0.64
	0.25	538	11%	0.40	213	39.9%	0.16	0.40	0.40	213	39.9%	0.16	0.40
PSBX	0.00	335	15%	0.37	125	0.6%	0.35	0.93	0.37	125	0.6%	0.35	0.93
	0.05	286	7%	0.43	124	1.3%	0.34	0.78	0.43	124	1.3%	0.34	0.78
	0.10	263	23%	0.47	123	10.5%	0.33	0.72	0.47	123	10.5%	0.33	0.72
	0.25	185	55%	0.59	110	87.6%	0.32	0.54	0.59	110	87.6%	0.32	0.54
QA	0.00	364	16%	0.28	100	1.6%	0.26	0.93	0.28	100	1.6%	0.26	0.93
	0.05	305	19%	0.32	99	4.9%	0.25	0.79	0.32	99	4.9%	0.25	0.79
	0.10	236	23%	0.40	94	13.9%	0.24	0.62	0.40	94	13.9%	0.24	0.62
	0.25	152	42%	0.52	80	79.6%	0.21	0.41	0.52	80	79.6%	0.21	0.41
QB	0.00	283	2%	0.56	157	0.0%	0.39	0.70	0.55	156	0.0%	0.37	0.67
	0.05	278	4%	0.57	157	0.6%	0.39	0.68	0.56	156	0.6%	0.36	0.65
	0.10	266	13%	0.59	156	4.4%	0.38	0.65	0.58	155	4.5%	0.36	0.61
	0.25	228	80%	0.66	149	94.9%	0.37	0.56	0.65	148	94.9%	0.34	0.53
QP	0.00	2,170	39%	0.15	330	5.8%	0.20	1.31	0.15	329	5.8%	0.19	1.28
	0.05	1,318	20%	0.24	310	9.2%	0.22	0.92	0.23	310	9.3%	0.21	0.90
	0.10	893	21%	0.31	280	22.1%	0.22	0.72	0.31	279	22.2%	0.22	0.69
	0.25	437	20%	0.47	207	62.8%	0.22	0.47	0.47	206	62.7%	0.21	0.44
SI	0.00	1,385	42%	0.18	247	5.2%	0.23	1.30	0.18	247	5.2%	0.23	1.30
	0.05	801	16%	0.29	234	6.2%	0.25	0.85	0.29	234	6.2%	0.25	0.85
	0.10	576	15%	0.38	219	13.4%	0.24	0.64	0.38	219	13.4%	0.24	0.64
	0.25	369	27%	0.50	185	75.1%	0.22	0.44	0.50	185	75.1%	0.22	0.44
SIH	0.00	1,415	26%	0.25	361	2.4%	0.30	1.19	0.25	360	2.4%	0.30	1.19
	0.05	1,050	17%	0.34	352	5.1%	0.32	0.94	0.33	352	5.2%	0.31	0.93
	0.10	803	21%	0.42	334	13.7%	0.32	0.77	0.41	333	13.7%	0.32	0.76
	0.25	503	36%	0.56	284	78.8%	0.32	0.57	0.56	284	78.8%	0.32	0.56
SIL	0.00	1,573	46%	0.15	234	6.7%	0.19	1.28	0.15	234	6.7%	0.19	1.28
	0.05	857	13%	0.25	218	6.4%	0.20	0.80	0.25	218	6.4%	0.20	0.80
	0.10	657	22%	0.31	203	26.5%	0.20	0.66	0.31	203	26.5%	0.20	0.66
	0.25	305	19%	0.46	141	60.4%	0.21	0.45	0.46	141	60.4%	0.21	0.45
UDEF	0.00	2,225	21%	0.18	396	1.7%	0.27	1.52	0.18	390	1.8%	0.23	1.33
	0.05	1,750	31%	0.22	389	10.7%	0.29	1.30	0.22	383	10.9%	0.25	1.12
	0.10	1,055	26%	0.33	347	21.7%	0.33	1.01	0.32	340	22.1%	0.27	0.84
	0.25	483	22%	0.54	261	65.8%	0.40	0.73	0.53	254	65.2%	0.28	0.54

Table 17-12: Distribution of Copper by Alteration - Mitchell

Alteration	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	46,869	17%	0.15	6,819	2.7%	0.13	0.88	0.14	6,766	2.7%	0.12	0.80
	0.05	38,966	23%	0.17	6,637	12.0%	0.13	0.74	0.17	6,584	12.1%	0.11	0.66
	0.10	28,267	47%	0.21	5,819	52.1%	0.13	0.64	0.20	5,766	52.5%	0.11	0.55
	0.25	6,235	13%	0.36	2,266	33.2%	0.20	0.55	0.35	2,213	32.7%	0.15	0.42
CL	0.00	8,192	8%	0.15	1,233	1.7%	0.10	0.64	0.15	1,233	1.7%	0.10	0.64
	0.05	7,566	28%	0.16	1,212	14.0%	0.09	0.59	0.16	1,212	14.0%	0.09	0.59
	0.10	5,288	51%	0.20	1,039	54.3%	0.09	0.46	0.20	1,039	54.3%	0.09	0.46
	0.25	1,107	14%	0.33	369	29.9%	0.09	0.28	0.33	369	29.9%	0.09	0.28
CL2	0.00	6,278	1%	0.17	1,083	0.2%	0.09	0.50	0.17	1,083	0.2%	0.09	0.50
	0.05	6,209	16%	0.17	1,081	7.7%	0.09	0.49	0.17	1,080	7.7%	0.09	0.49
	0.10	5,187	68%	0.19	998	64.1%	0.08	0.43	0.19	997	64.1%	0.08	0.42
	0.25	924	15%	0.33	304	28.0%	0.09	0.28	0.33	303	28.0%	0.09	0.27
IARG	0.00	7,160	23%	0.11	781	5.1%	0.08	0.77	0.11	781	5.1%	0.08	0.77
	0.05	5,534	33%	0.13	741	22.7%	0.08	0.59	0.13	741	22.7%	0.08	0.59
	0.10	3,192	38%	0.18	564	52.8%	0.08	0.46	0.18	564	52.8%	0.08	0.46
	0.25	463	6%	0.33	151	19.3%	0.10	0.30	0.33	151	19.3%	0.10	0.30
KP	0.00	871	25%	0.12	107	5.8%	0.11	0.90	0.12	107	5.8%	0.11	0.90
	0.05	653	31%	0.15	101	19.1%	0.11	0.72	0.15	101	19.1%	0.11	0.72
	0.10	379	33%	0.21	80	44.4%	0.11	0.54	0.21	80	44.4%	0.11	0.54
	0.25	88	10%	0.37	33	30.7%	0.13	0.36	0.37	33	30.7%	0.13	0.36
MTH	0.00	630	5%	0.17	109	1.1%	0.16	0.91	0.17	108	1.1%	0.15	0.87
	0.05	597	41%	0.18	108	18.3%	0.16	0.88	0.18	107	18.5%	0.15	0.84
	0.10	340	32%	0.26	88	28.2%	0.17	0.67	0.26	87	28.5%	0.16	0.63
	0.25	136	21%	0.42	57	52.4%	0.17	0.40	0.41	56	51.9%	0.14	0.34
PR	0.00	5,974	23%	0.12	730	3.8%	0.09	0.75	0.12	730	3.8%	0.09	0.75
	0.05	4,598	21%	0.15	702	13.6%	0.08	0.54	0.15	702	13.6%	0.08	0.54
	0.10	3,324	47%	0.18	603	59.7%	0.08	0.44	0.18	603	59.7%	0.08	0.44
	0.25	513	9%	0.33	168	23.0%	0.09	0.27	0.33	168	23.0%	0.09	0.26
QSP	0.00	6,802	15%	0.19	1,277	2.0%	0.21	1.13	0.18	1,237	2.0%	0.17	0.94
	0.05	5,751	18%	0.22	1,252	7.1%	0.22	1.00	0.21	1,212	7.3%	0.17	0.81
	0.10	4,537	44%	0.26	1,162	39.2%	0.23	0.90	0.25	1,122	40.5%	0.17	0.71
	0.25	1,575	23%	0.42	661	51.7%	0.33	0.78	0.39	621	50.2%	0.23	0.58
QSTW	0.00	3,509	9%	0.20	717	1.0%	0.13	0.65	0.20	713	1.0%	0.12	0.61
	0.05	3,196	6%	0.22	710	2.4%	0.13	0.57	0.22	705	2.4%	0.12	0.52
	0.10	2,974	60%	0.23	693	51.8%	0.12	0.54	0.23	688	52.1%	0.11	0.49
	0.25	878	25%	0.37	321	44.8%	0.16	0.43	0.36	316	44.4%	0.13	0.35
SIH	0.00	3,685	33%	0.10	372	7.3%	0.11	1.05	0.10	371	7.3%	0.10	1.00
	0.05	2,481	31%	0.14	345	23.2%	0.11	0.79	0.14	344	23.3%	0.10	0.74
	0.10	1,350	30%	0.19	259	43.9%	0.13	0.66	0.19	257	44.1%	0.12	0.61
	0.25	243	7%	0.39	95	25.6%	0.18	0.46	0.39	94	25.2%	0.15	0.39
SIL	0.00	1,427	36%	0.10	149	7.0%	0.10	0.93	0.10	149	7.1%	0.09	0.88
	0.05	909	19%	0.15	139	13.9%	0.09	0.60	0.15	138	14.0%	0.08	0.54
	0.10	637	37%	0.19	118	56.5%	0.09	0.49	0.18	117	56.8%	0.08	0.42
	0.25	104	7%	0.32	34	22.5%	0.14	0.43	0.32	33	22.2%	0.08	0.27
UDEF	0.00	638	60%	0.08	49	5.4%	0.18	2.37	0.07	45	5.9%	0.12	1.70
	0.05	255	17%	0.18	47	14.9%	0.26	1.40	0.17	42	16.3%	0.14	0.86
	0.10	147	15%	0.27	39	29.8%	0.31	1.16	0.24	35	32.6%	0.15	0.64
	0.25	52	8%	0.48	25	50.0%	0.45	0.95	0.39	20	45.2%	0.16	0.41

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 17-6 show cumulative probability plots using the cumulative normal distribution function.

Figure 17-1: Kerr Au Assay Cumulative Probability Plot

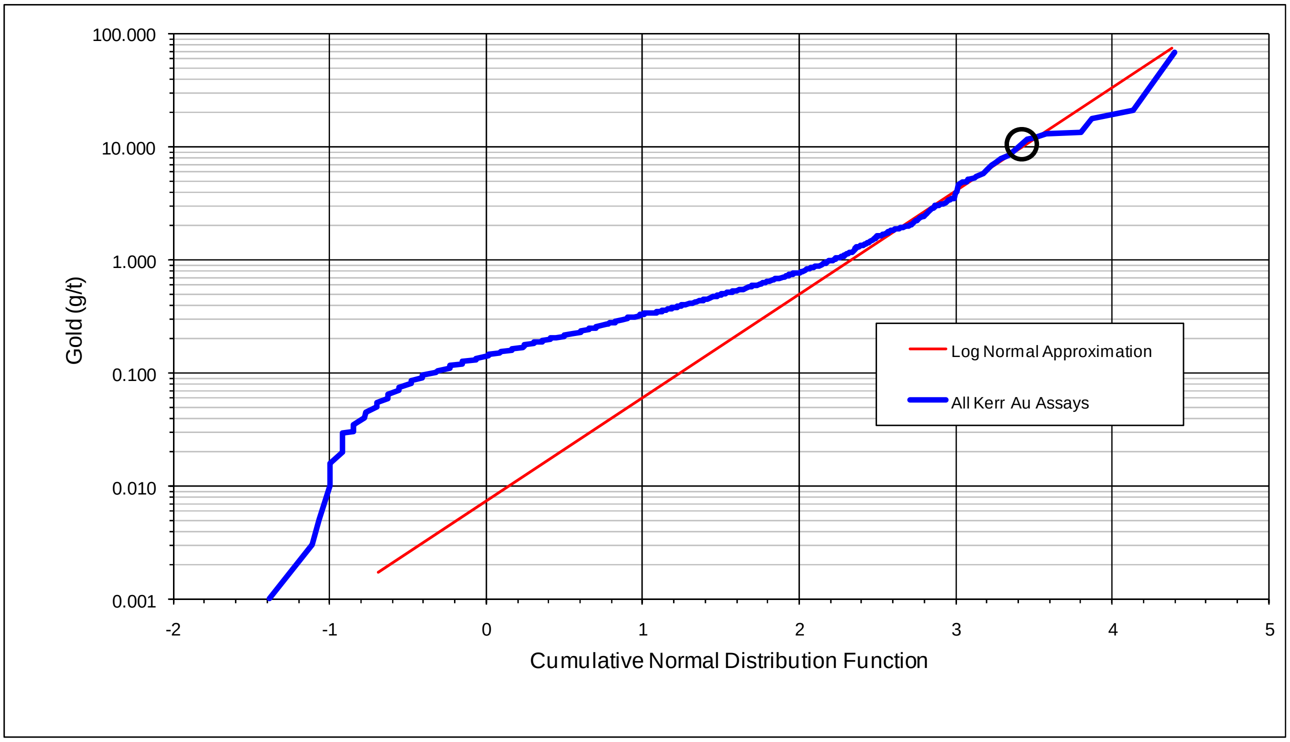


Figure 17-2: Sulphurets Au Assay Cumulative Probability Plot

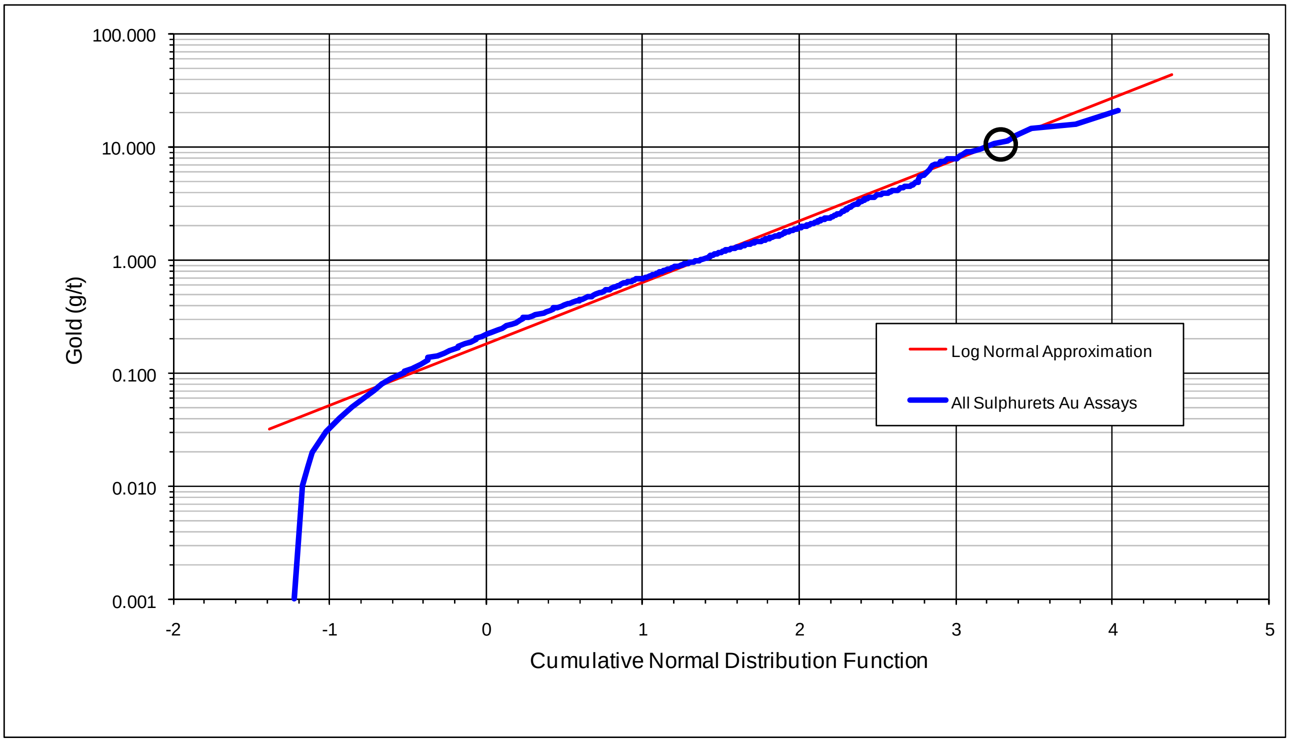


Figure 17-3: Mitchell Au Assay Cumulative Probability Plot

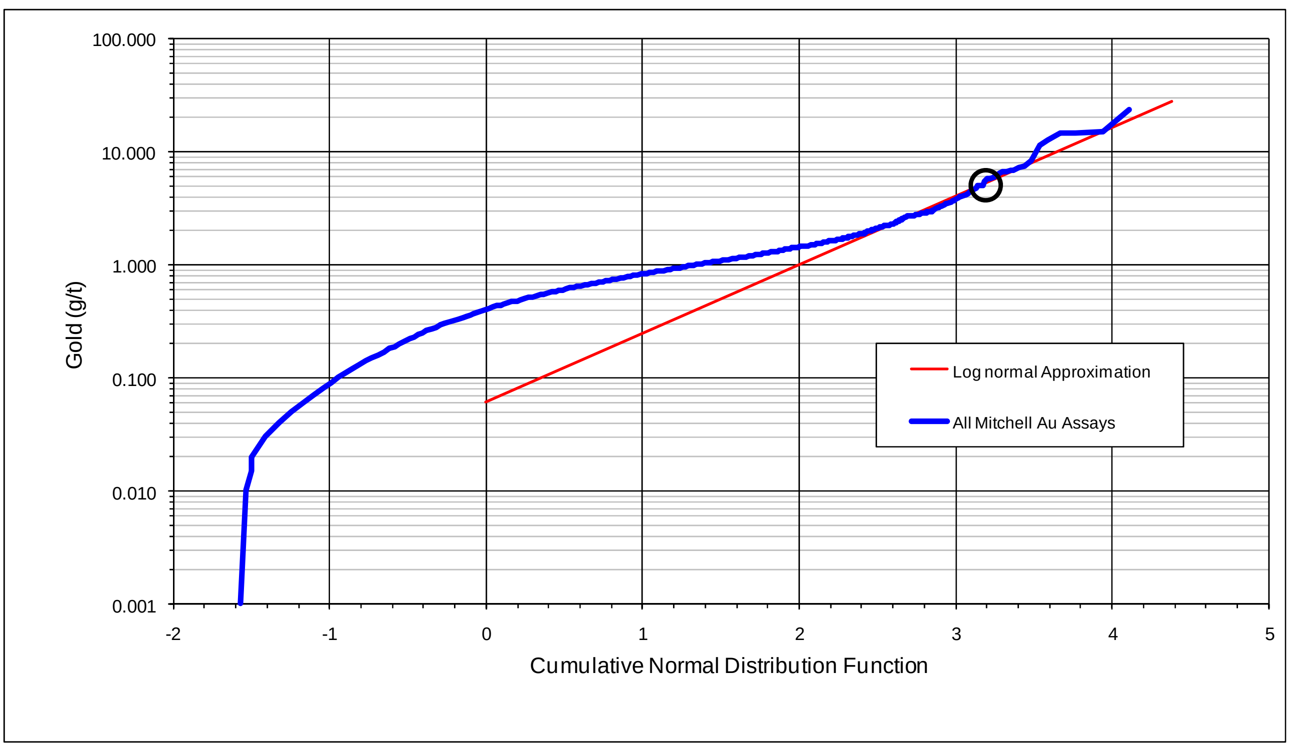


Figure 17-4: Kerr Cu Assay Cumulative Probability Plot

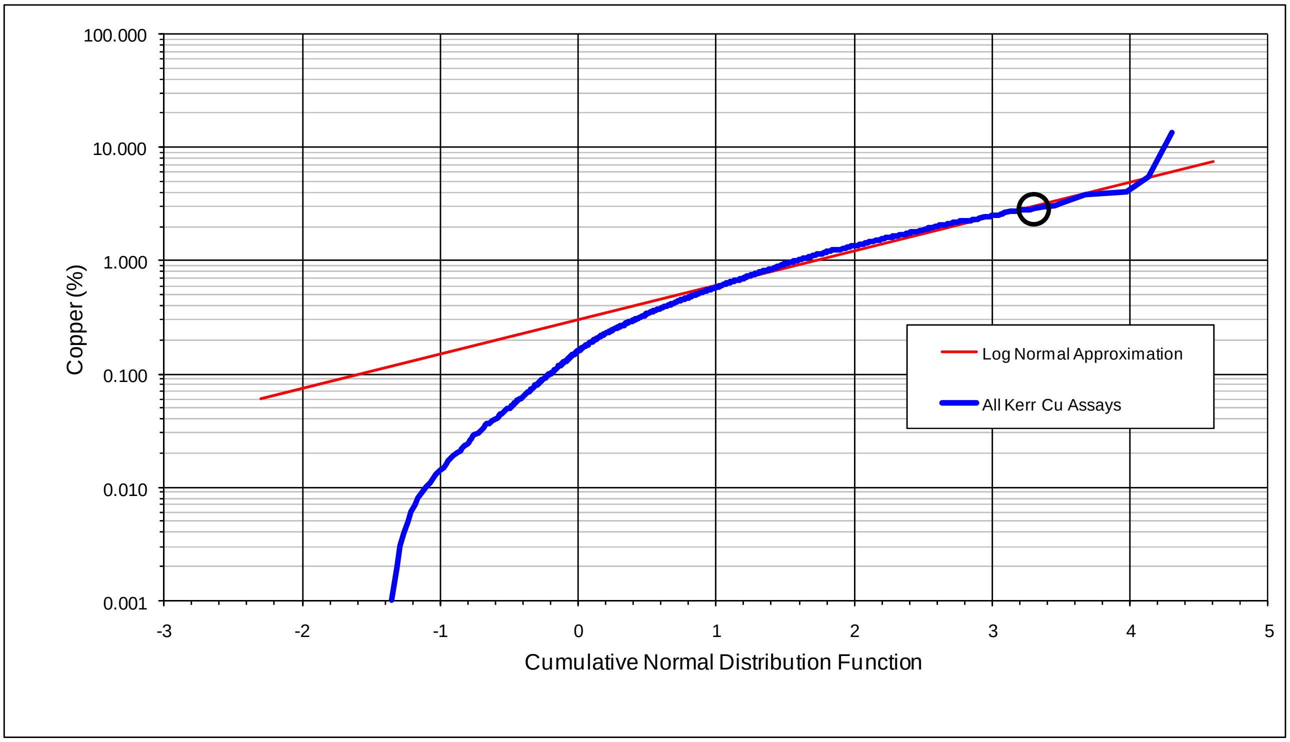


Figure 17-5: Sulphurets Cu Assay Cumulative Probability Plot

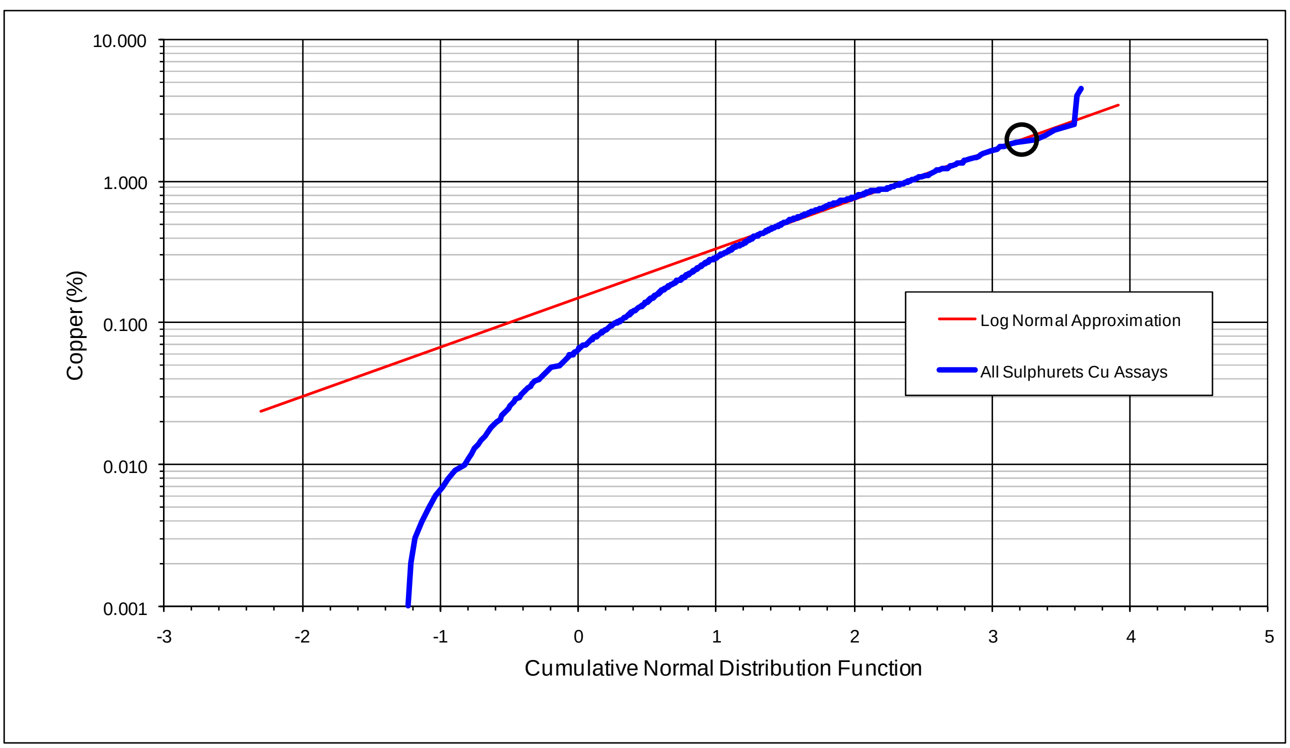
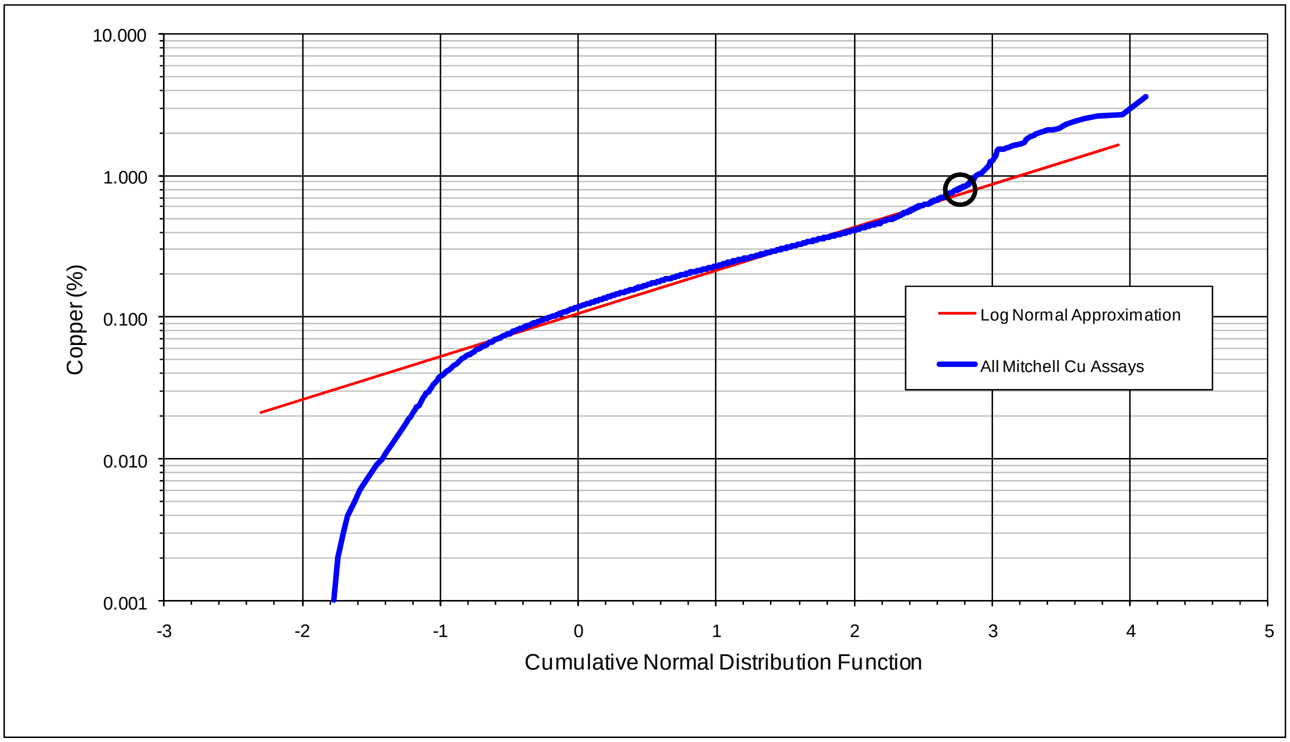


Figure 17-6: Mitchell Cu Assay Cumulative Probability Plot



Based on the information shown in Figures 17-1 through 17-6, the author capped raw gold and copper assays at the area highlighted by the black circle where the distribution of grades becomes erratic.

Table 17-13 summarizes the capping limits that were established for gold, silver, copper, and molybdenum by deposit. In addition to the capping limit for each metal, the number of raw assays that were capped prior to creating 15-meter-long drill hole composites is also provided.

Table 17-13: Grade Capping Limits

Deposit - Zone	Gold (g/t)		Silver (g/t)		Copper (%)		Molybdenum (ppm)	
	Cap Grade	No. Capped	Cap Grade	No. Capped	Cap Grade	No. Capped	Cap Grade	No. Capped
Kerr	10.0	7	500	1	2.75	10	n/a	n/a
Sulphurets	10.0	7	100	3	2.00	5	1000	7
Mitchell (main zone)	5.0	19	180	7	0.90	18	1200	5
Mitchell (bornite breccia)	n/a	n/a	n/a	n/a	1.50	22	n/a	n/a
Mitchell (leach breccia)	n/a	n/a	n/a	n/a	0.35	9	n/a	n/a
Mitchell (undefined Cu Zone)	n/a	n/a	n/a	n/a	0.80	7	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 three deposits are not necessarily confined to distinct geologic units. For this reason, RMI elected to use gold, copper, and molybdenum grade envelopes for constraining the estimate of block grades. The cutoff grades for each metal are summarized in Table 17-14 for each deposit.

Table 17-14: Grade Envelope Cutoffs

Deposit	Au (g/t)	Cu (%)	Mo (ppm)
Kerr	0.15	0.30	n/a
Sulphurets	0.25	0.10	n/a
Mitchell	0.25	0.10	50

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. Down-hole correlograms were generated using the original raw assay data in order to establish the nugget effect for gold and copper.

Figures 17-7 through 17-9 show gold grade correlograms for the Kerr, Sulphurets, and Mitchell deposits, respectively. Figures 17-10 through 17-12 show copper grade correlograms for the Kerr, Sulphurets, and Mitchell deposits, respectively. The correlograms shown in Figures 17-7 through 17-12 are the same as those presented in RMI's March 2009 NI 43-101 Technical Report (Lechner, 2009).

Figure 17-7: Kerr Au Grade Correlogram

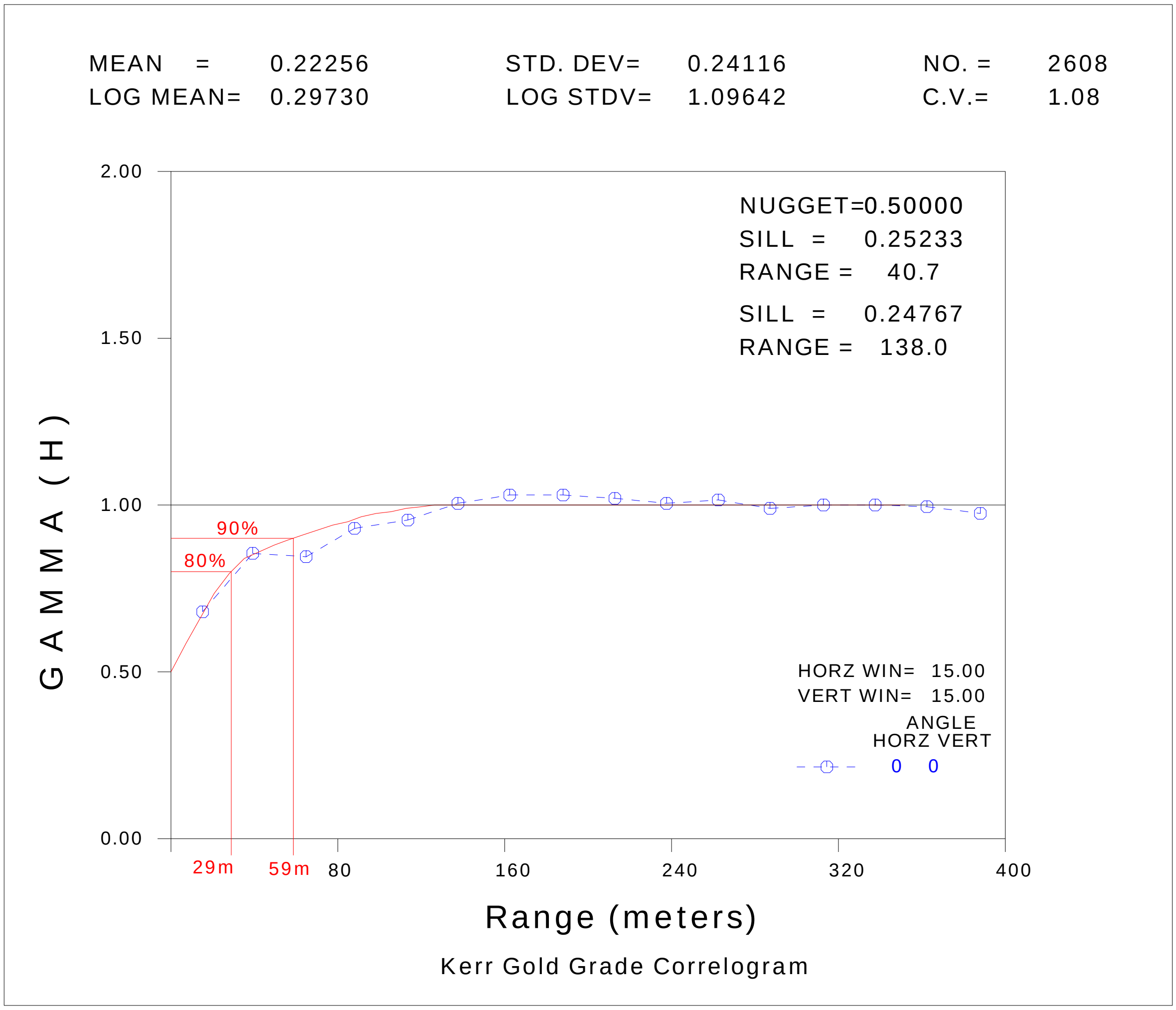


Figure 17-8: Sulphurets Au Grade Correlogram

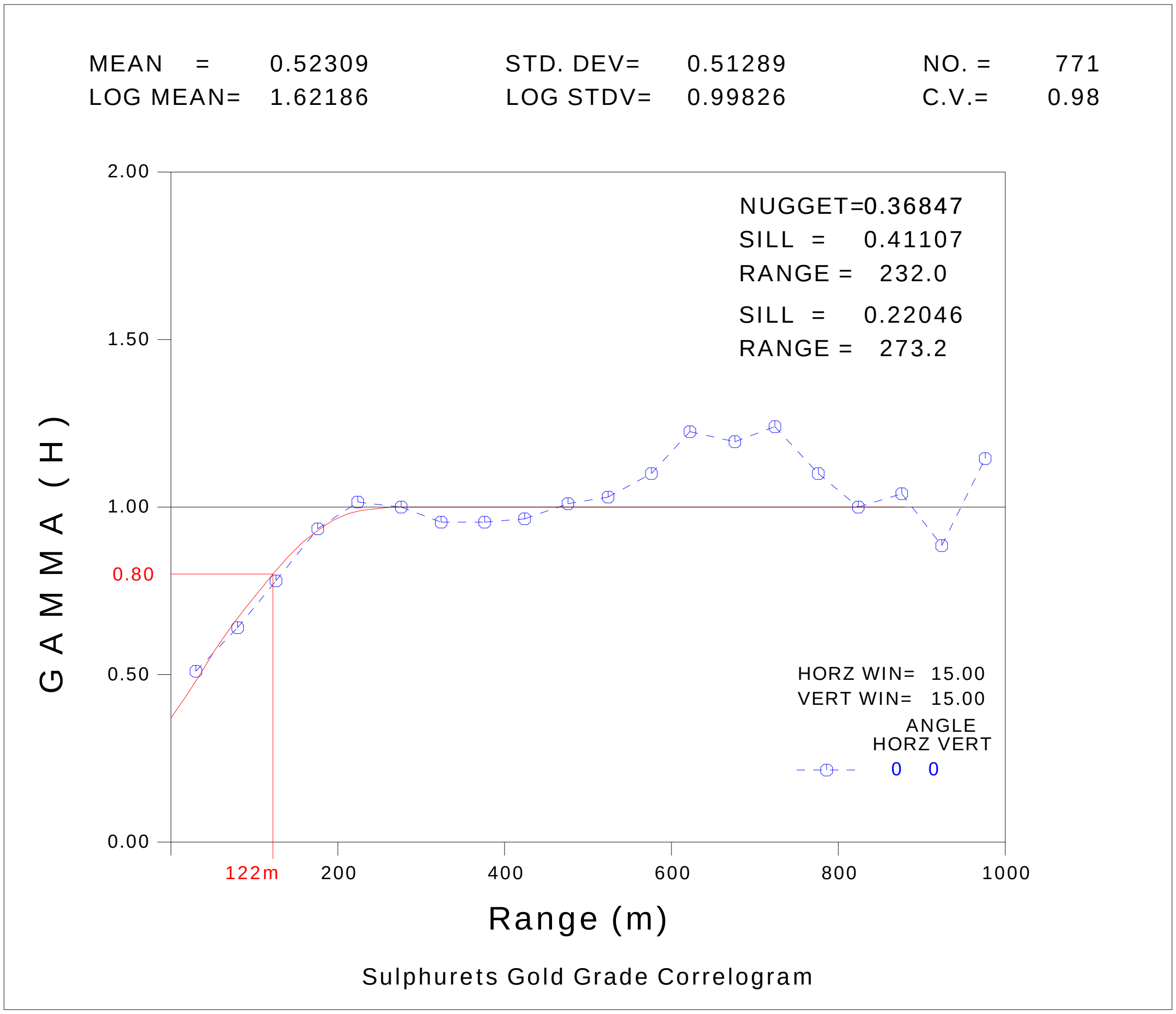


Figure 17-9: Mitchell Au Grade Correlogram

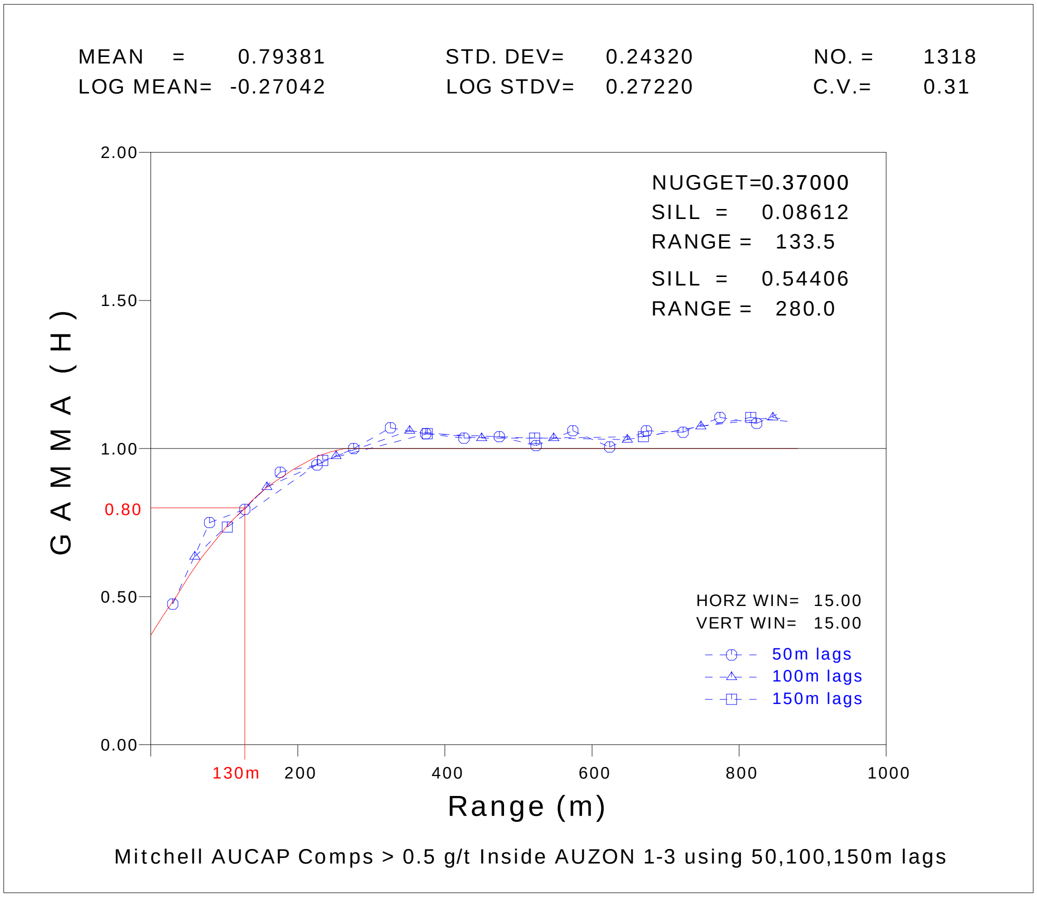


Figure 17-10: Kerr Cu Indicator Correlogram

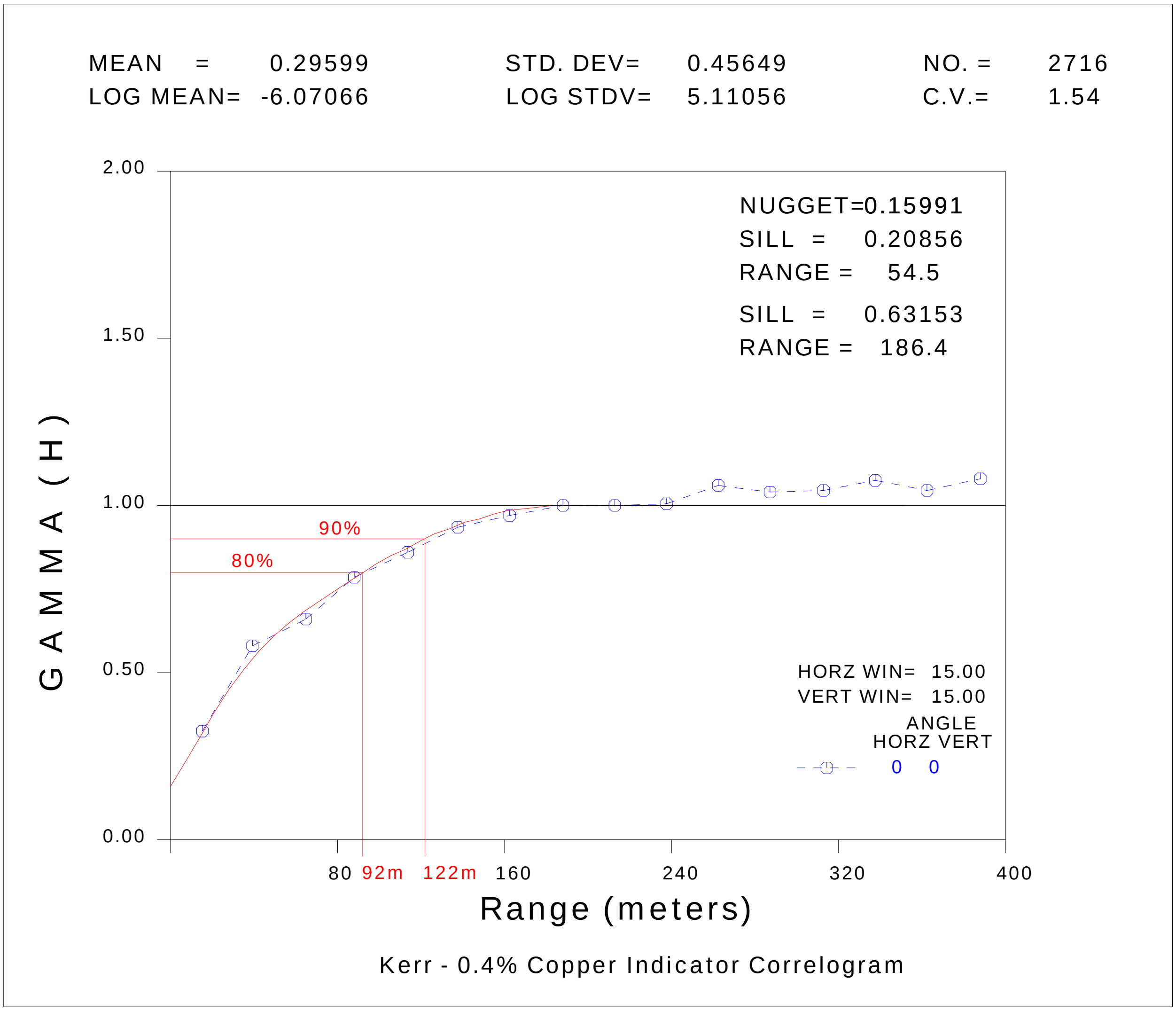


Figure 17-11: Sulphurets Cu Grade Correlogram

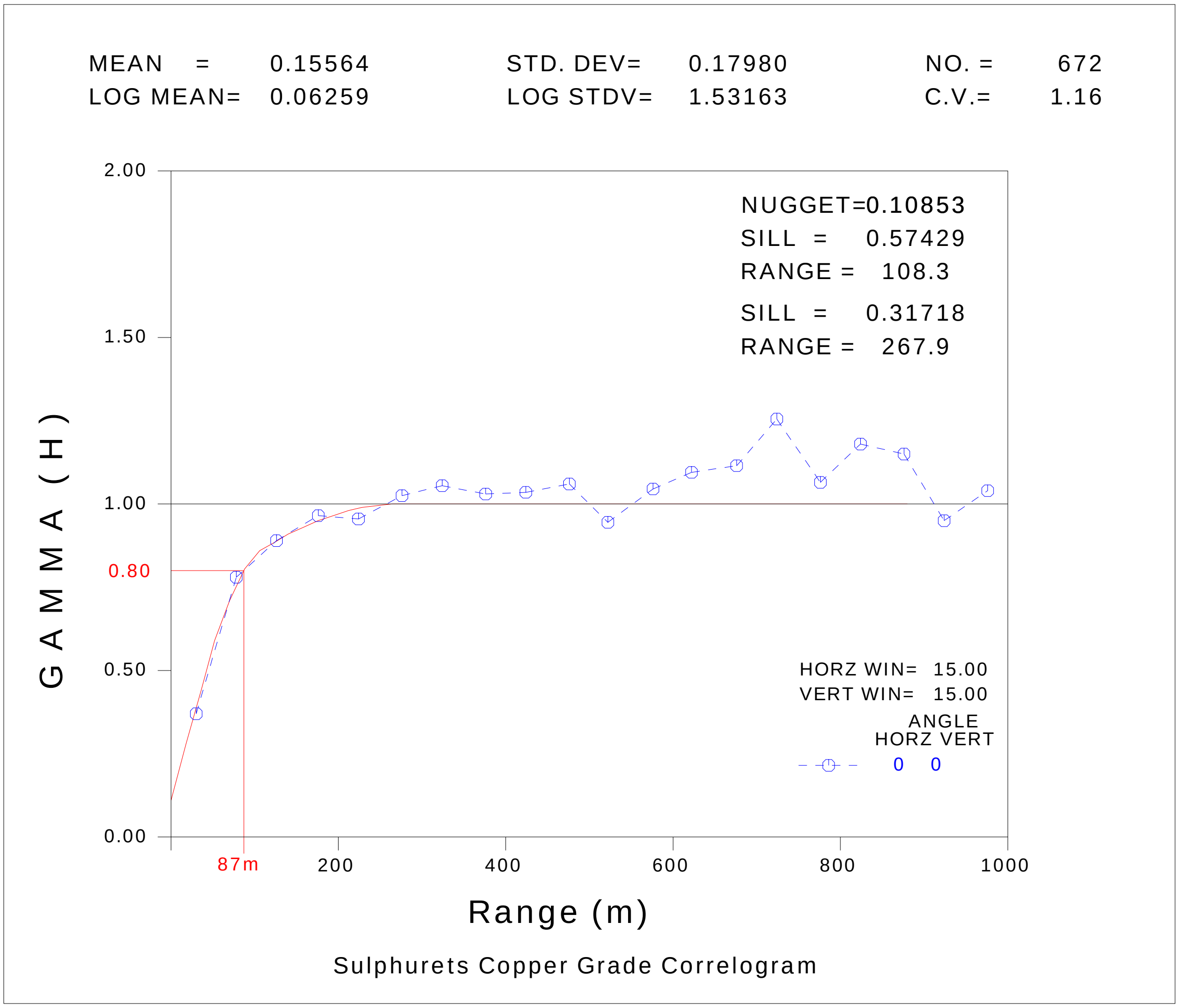
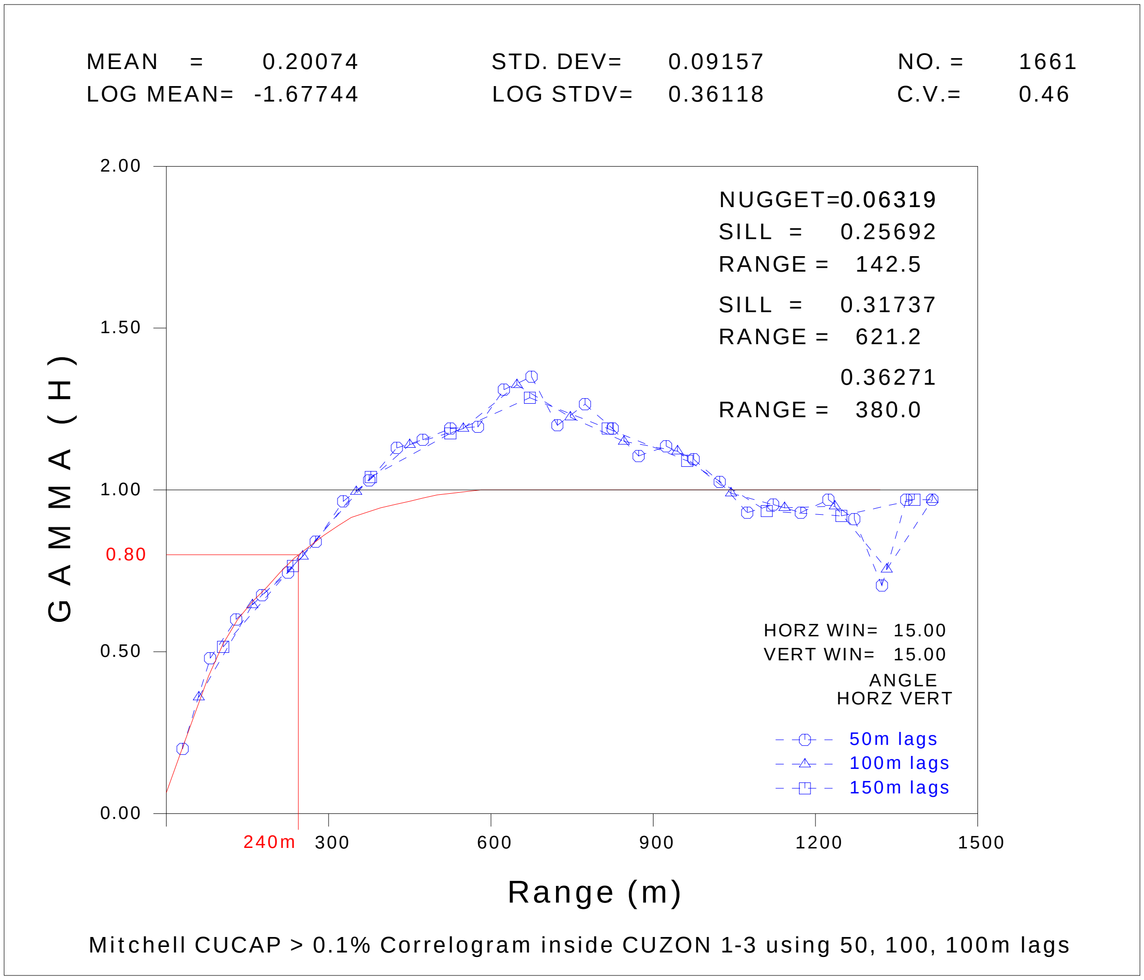


Figure 17-12: Mitchell Cu Grade Correlogram



The correlograms shown in Figures 17-7 through 17-12 were modeled with nested spherical models. Total ranges for gold are 138m, 273m, and 280m for the Kerr, Sulphurets, and Mitchell deposits, respectively. At 80% of the total sill, gold ranges of 29m, 122m, and 130m were interpreted for the Kerr, Sulphurets, and Mitchell, respectively. Total ranges for copper are 186m, 268m, and 380m for the Kerr, Sulphurets, and Mitchell deposits, respectively. At 80% of the total sill, copper ranges of 92m, 87m, and 240m were interpreted for the Kerr, Sulphurets, and Mitchell, respectively.

The Sulphurets and Mitchell deposits show remarkably long ranges for gold, which is related to the style and intensity of mineralization. The Kerr gold grade range is significantly less than the other two deposits but Kerr has significantly lower gold grades and higher copper grades than the other deposits. The copper range at 80% of the total sill value for the Kerr deposit is 92 meters.

17.7 Grade Estimation Parameters

RMI constructed a three-dimensional block model using MineSight®, a widely recognized commercial mine engineering software package. Table 17-15 summarizes various block parameters for this non-rotated model which uses NAD83 UTM coordinates.

Table 17-15: KSM Block Model Dimensions

Parameter	NAD83 Coordinates		Block Size (m)	Number of Blocks	Areal Extent (m)
	Minimum	Maximum			
Easting	420,500	425,300	25	192	4,800
Northing	6,257,800	6,268,000	25	408	10,200
Elevation	-210	2,055	15	151	2,265

Block gold, silver, copper, and molybdenum grades were estimated by three distinct methods: 1) inverse distance weighting 2) ordinary kriging and 3) nearest neighbor. Gold and copper resources summarized in this report are based on inverse distance squared methods.

A multi-pass estimation strategy was used for gold, silver, copper and molybdenum. The first estimation pass required two or more drill holes to estimate block grades while subsequent passes acted as a "cleanup" runs that filled un-estimated blocks by using larger search ellipses and requiring fewer drill holes. The inverse distance, and ordinary kriging estimation plans used block/composite zone matching. For example, blocks located inside of the 0.25 g/t gold envelope were estimated by drill hole composites from that same population.

Tables 17-16 through 17-18 summarizes the key estimation parameters that were used to estimate block gold grades using inverse distance squared methods for the Kerr, Sulphurets, and Mitchell deposits, respectively. Silver grades were estimated for the Sulphurets and Mitchell deposits using the same parameters shown in Tables 17-17 and 17-18.

Table 17-16: Kerr Au Grade Estimation Parameters

Estimation Pass	Population	Block Search Distances (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	Inside 0.15 g/t shape	75	75	15	3	6	2	20	0	60
2	Inside 0.15 g/t shape	125	125	25	3	6	2	20	0	60
3	Inside 0.15 g/t shape	200	200	40	1	3	1	20	0	60
1	Outside 0.15 g/t shape	100	100	20	3	6	2	20	0	60
2	Outside 0.15 g/t shape	100	100	20	1	3	1	20	0	60

Table 17-17: Sulphurets Au Grade Estimation Parameters

Estimation Pass	Population	Block Search Distances (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	Inside 0.25 g/t shape	200	200	30	3	6	2	53	15	35
2	Inside 0.25 g/t shape	200	200	30	1	3	1	53	15	35
3	Inside 0.25 g/t shape	500	500	75	3	8	2	53	15	35
1	Gold breccia	300	300	60	3	6	2	53	15	35
2	Gold breccia	300	300	60	1	3	1	53	15	35
1	Leach breccia	300	300	60	3	6	2	53	15	35
2	Leach breccia	300	300	60	1	3	1	53	15	35
1	Outside 0.25 g/t shape	100	100	15	3	6	2	53	15	35
2	Outside 0.25 g/t shape	100	100	20	1	3	1	53	15	35
1	Upper plate seds	300	300	100	2	6	2	53	15	35

Table 17-18: Mitchell Au Grade Estimation Parameters

Estimation Pass	Population	Block Search Distances (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	Upper Plate Inside Au shape	125	125	30	3	8	2	75	0	40
2	Upper Plate Inside Au shape	250	250	60	3	8	2	75	0	40
3	Upper Plate Inside Au shape	375	375	90	3	8	2	75	0	40
4	Upper Plate Inside Au shape	500	500	120	1	3	1	75	0	40
1	Lower Plate Inside Au shape	125	125	30	3	8	2	75	0	40
2	Lower Plate Inside Au shape	250	250	60	3	8	2	75	0	40
3	Lower Plate Inside Au shape	375	375	90	3	8	2	75	0	40
4	Lower Plate Inside Au shape	500	500	120	1	3	1	75	0	40
1	Bornite breccia	250	250	60	3	8	2	275	0	65
2	Bornite breccia	375	375	90	1	3	1	275	0	65
1	Leach breccia	250	250	60	3	8	2	275	0	65
2	Leach breccia	500	500	120	1	3	1	275	0	65
1	Upper plate seds	150	150	45	3	8	2	75	0	40
3	Upper plate seds	300	300	100	1	3	1	75	0	40
1	Lower Plate Outside Au shape	150	150	45	3	8	2	75	0	40
2	Lower Plate Outside Au shape	300	300	100	1	3	1	75	0	40

Tables 17-19 through 17-21 summarizes the key estimation parameters that were used to estimate block copper grades using inverse distance squared methods for the Kerr, Sulphurets, and Mitchell deposits, respectively.

Table 17-19: Kerr Cu Grade Estimation Parameters

Estimation Pass	Population	Block Search Distances (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	Inside 0.30% shape	75	75	15	1	3	1	20	0	60
2	Inside 0.30% shape	125	125	25	1	3	1	20	0	60
3	Inside 0.30% shape	200	200	40	1	3	1	20	0	60
1	Outside 0.30% shape	100	100	20	2	6	2	20	0	60

Table 17-20: Sulphurets Cu Grade Estimation Parameters

Estimation Pass	Population	Block Search Distances (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	Upper Plate Inside Cu shape	200	200	30	3	6	2	53	15	35
2	Upper Plate Inside Cu shape	200	200	30	1	3	1	53	15	35
1	Lower Plate Inside Cu shape	200	200	30	3	6	2	53	15	35
2	Lower Plate Inside Cu shape	200	200	30	1	3	1	53	15	35
3	Lower Plate Inside Cu shape	500	500	75	3	8	2	53	15	35
1	Lower Plate Outside Cu shape	100	100	15	3	6	2	53	15	35
2	Lower Plate Outside Cu shape	100	100	15	1	3	1	53	15	35
1	Upper plate seds	300	300	100	2	6	2	53	15	35

Table 17-21: Mitchell Cu Grade Estimation Parameters

Estimation Pass	Population	Block Search Distances (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	Upper Plate Inside Cu Shape	250	250	60	3	8	2	75	0	40
2	Upper Plate Inside Cu Shape	500	500	120	1	3	1	75	0	40
1	Lower Plate Inside Cu Shape	250	250	60	3	8	2	75	0	40
2	Lower Plate Inside Cu Shape	500	500	120	1	3	1	75	0	40
1	Bornite breccia	250	250	60	3	8	2	275	0	65
2	Bornite breccia	375	375	90	1	3	1	275	0	65
1	Leach breccia	250	250	60	3	8	2	275	0	65
2	Leach breccia	500	500	120	1	3	1	275	0	65
1	Upper plate seds	150	150	45	3	8	2	75	0	40
2	Upper plate seds	75	75	15	1	3	1	75	0	40
1	Upper Plate Outside Cu Shape	300	300	100	1	3	1	75	0	40
1	Lower Plate Outside Cu Shape	150	150	45	3	8	2	75	0	40
2	Lower Plate Outside Cu Shape	150	150	45	1	3	1	75	0	40

Table 17-22 summarizes the key estimation parameters that were used to estimate block molybdenum grades using inverse distance squared methods for the Sulphurets and Mitchell deposits.

Table 17-22: Molybdenum Grade Estimation Parameters

Estimation Pass	Block Search Distances (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
	X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	300	300	300	1	3	1	53	15	35
2	250	250	60	3	8	2	53	15	35

Notes:

ROTN = major axis rotation in degrees using “left hand” rule about the Z-axis
DIPN = dip angle of major axis (negative means downward)
DIPE = semi-major axis rotation using “left hand” rule about the Y-axis

The number of composites and drill holes used to estimate block gold and copper grades were captured during the estimation process along with the distance to the closest composite that was used to estimate each block. These criteria among others were used to classify resources (see Section 17-10). The majority of the Mineral Resources that are subject to this report are based on blocks that were estimated by initial estimation passes that required two or more drill holes.

17.8 Grade Model Verification

Estimated block grades were verified by visual and statistical methods. The author visually compared estimated block gold and copper grades versus 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 17-13 and 17-4 are east-west cross sections through the Kerr block model drawn at northing coordinate 6,258,800. These figures show estimated block/composite gold grades (Figure 17-3) and block/composite copper grades (Figure 17-4). Figures 17-15 and 17-16 are block model level maps drawn at the 1500 elevation through the Kerr model showing estimated block/composite gold and copper grades, respectively. Figures 17-17 and 17-18 are northwest-southeast cross sections through the Sulphurets block model drawn at Section 23. These figures show estimated block/composite gold grades (Figure 17-17) and block/composite copper grades (Figure 17-8). Figures 17-19 and 17-20 are block model level maps drawn at the 1275 elevation through the Sulphurets model showing estimated block/composite gold and copper grades, respectively. Figures 17-21 and 17-22 are northeast-southwest cross sections through the Mitchell block model drawn at Section 11. These figures show estimated block/composite gold grades (Figure 17-21) and block/composite copper grades (Figure 17-22). Figures 17-23 and 17-24 are block model level maps drawn at the 660 elevation through the Mitchell model showing estimated block/composite gold and copper grades, respectively.

Figure 17-13: Kerr Au Block Model Section 6,258,800 North

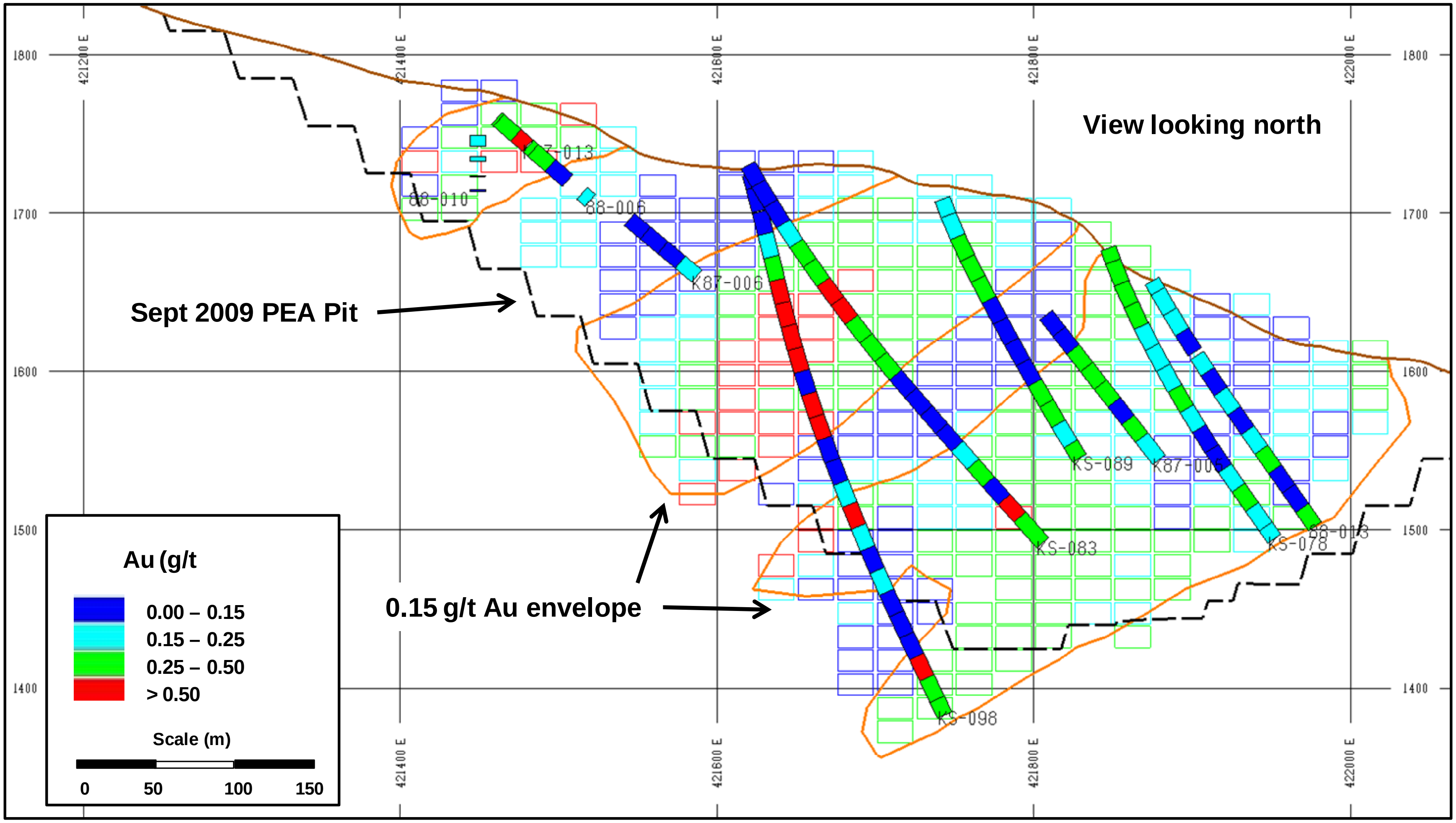


Figure 17-14: Kerr Cu Block Model Section 6,258,800 North

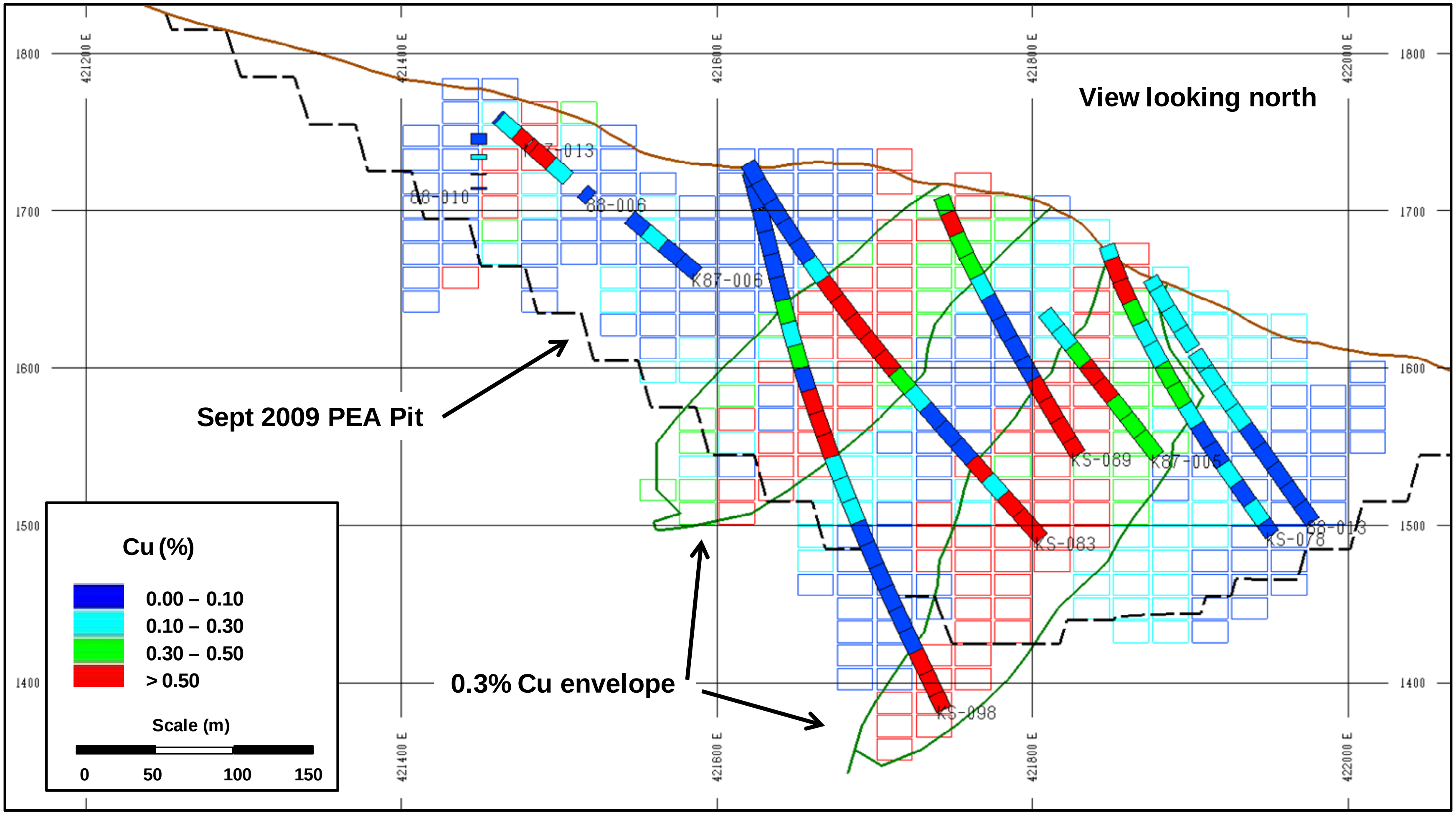


Figure 17-15: Kerr Au Block Model - 1500 Level

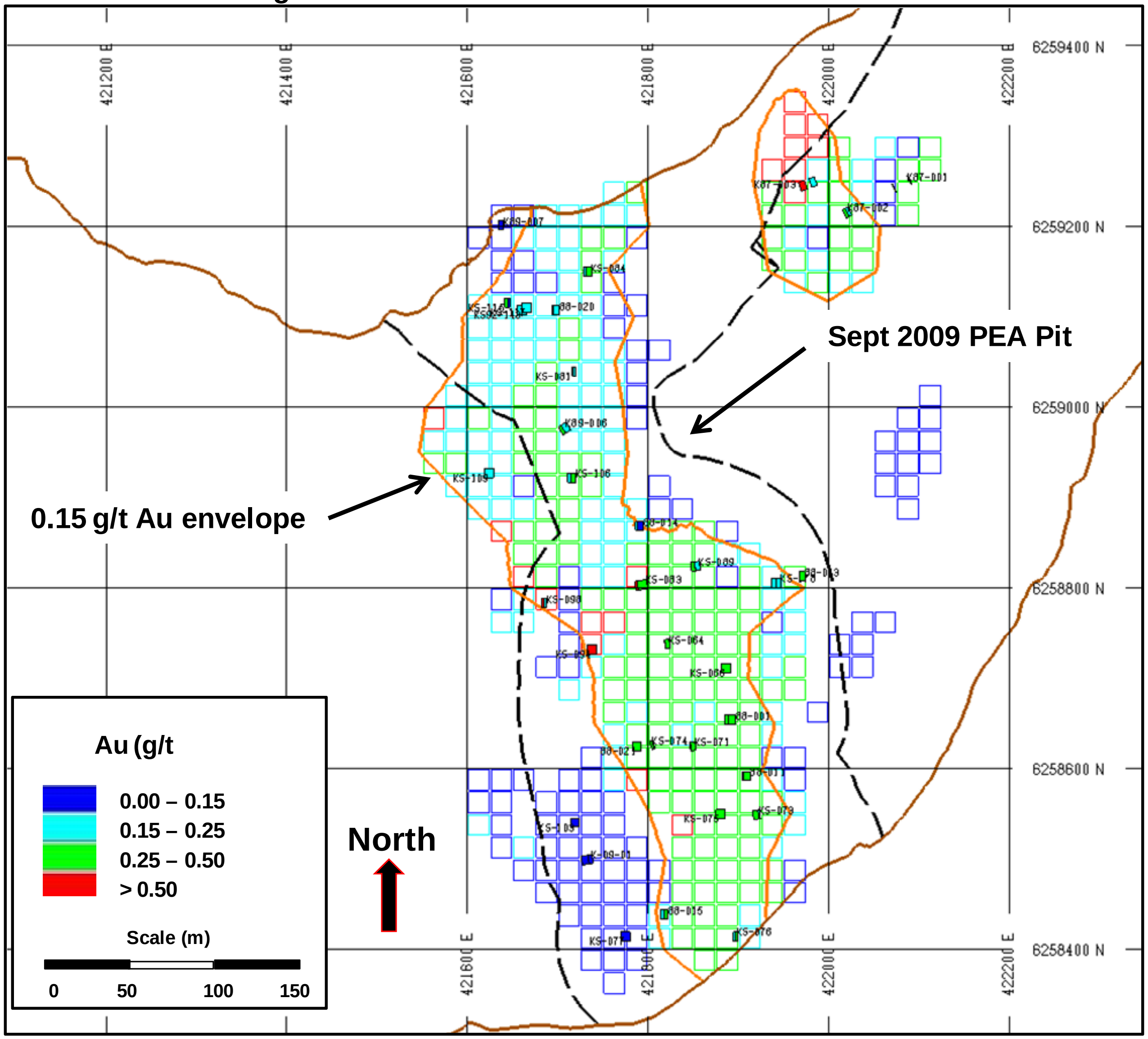


Figure 17-16: Kerr Cu Block Model - 1500 Level

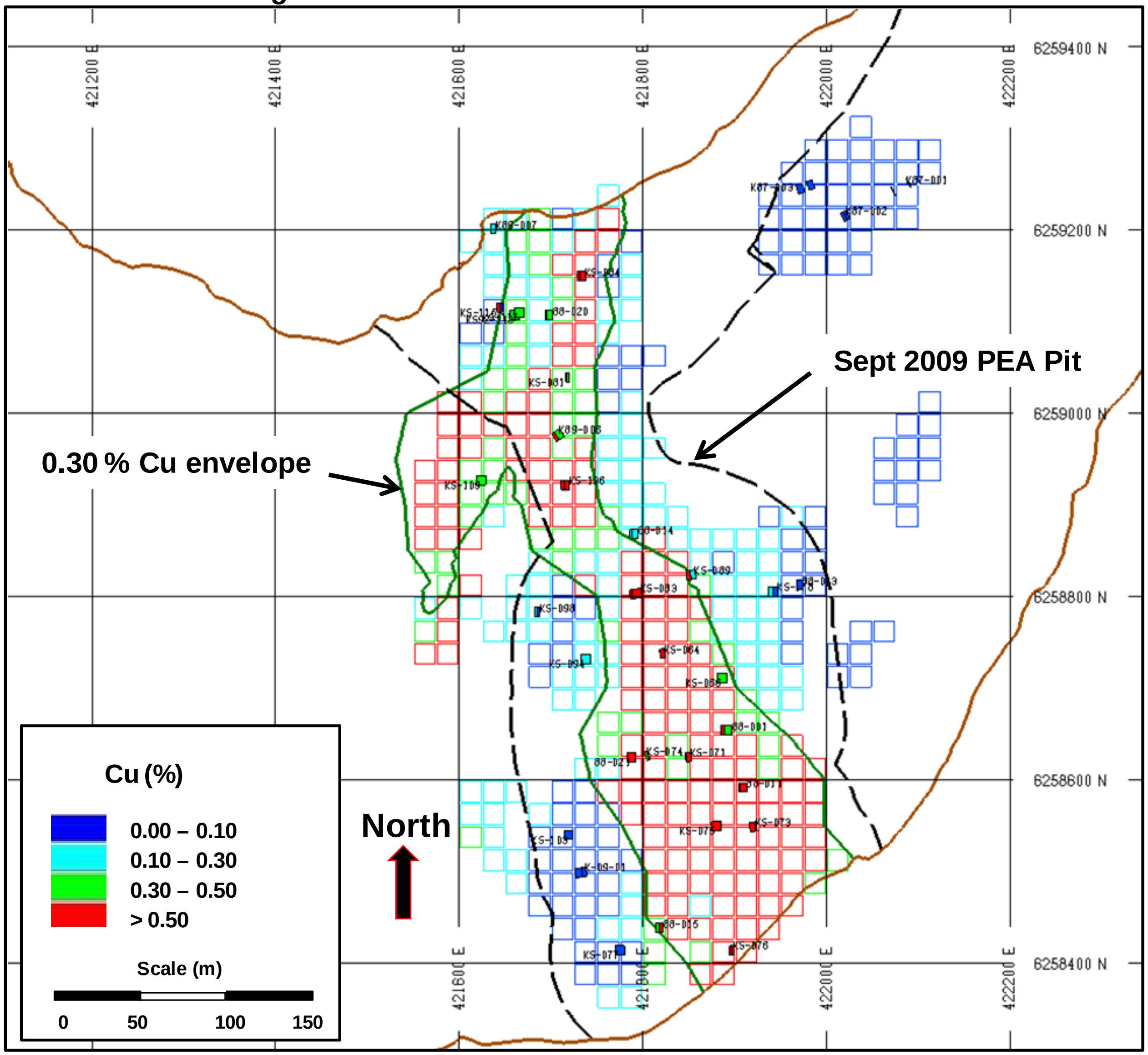


Figure 17-17: Sulphurets Au Block Model Cross Section 23

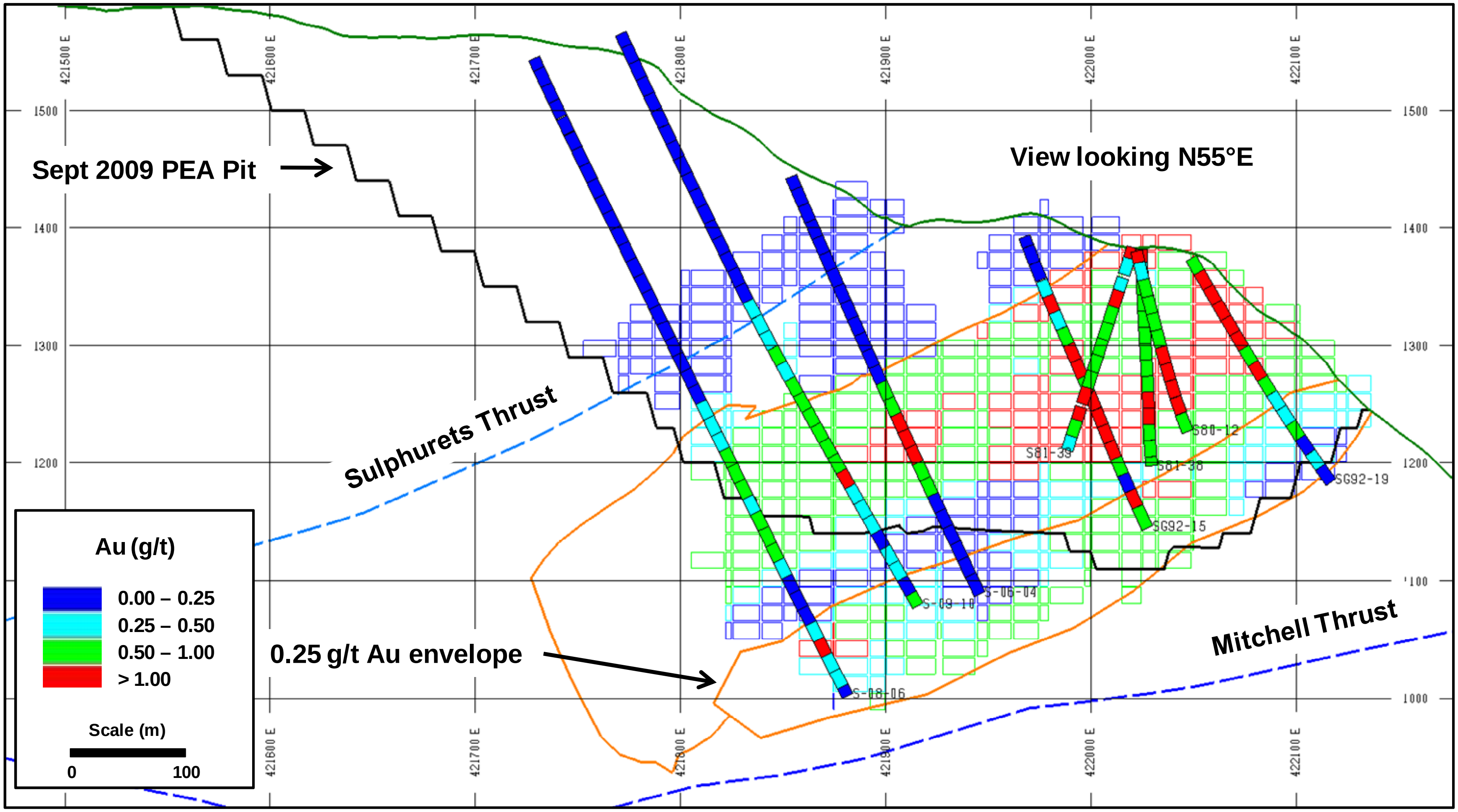


Figure 17-18: Sulphurets Cu Block Model Cross Section 23

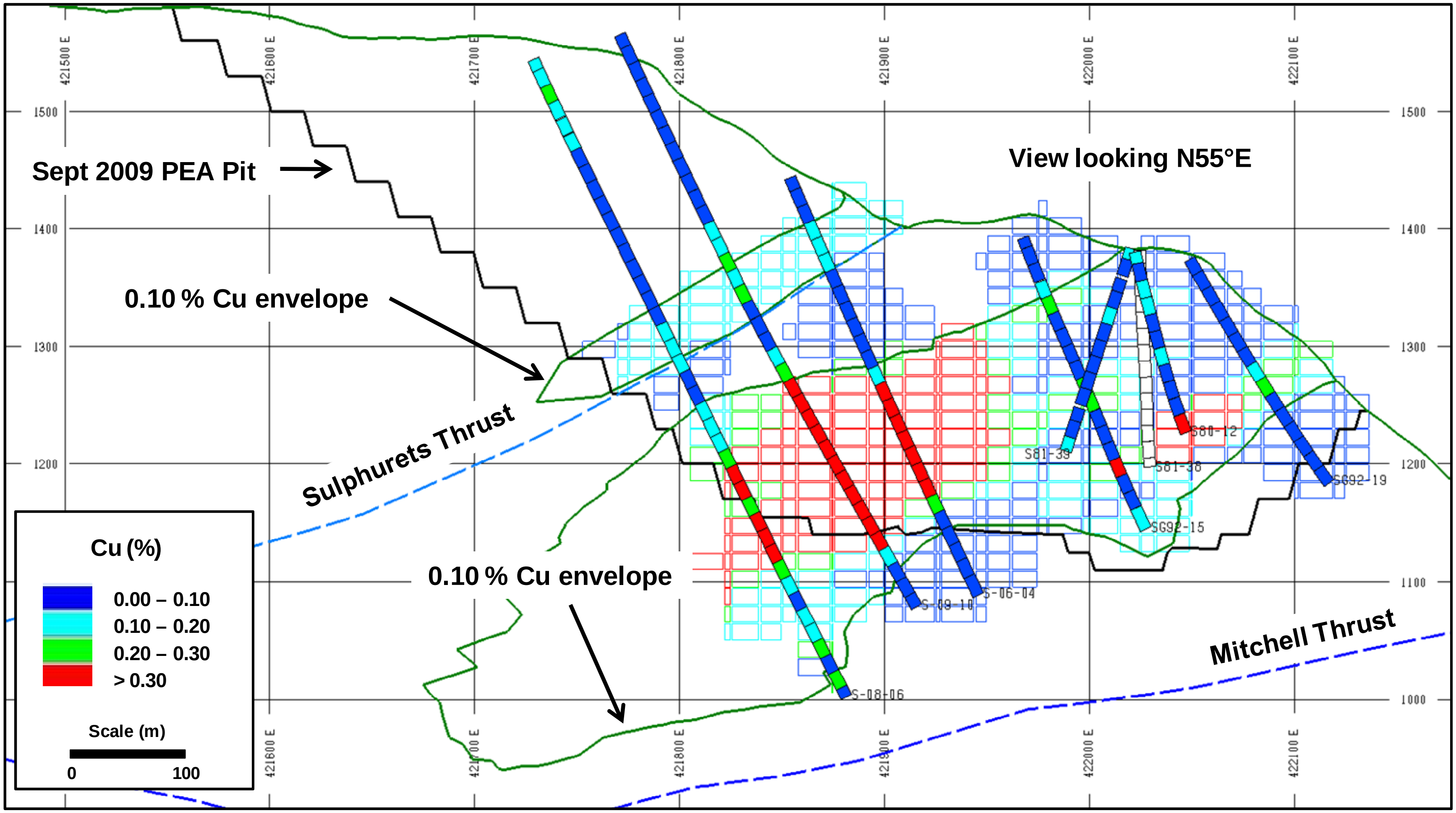


Figure 17-19: Sulphurets Au Block Model - 1275 Level

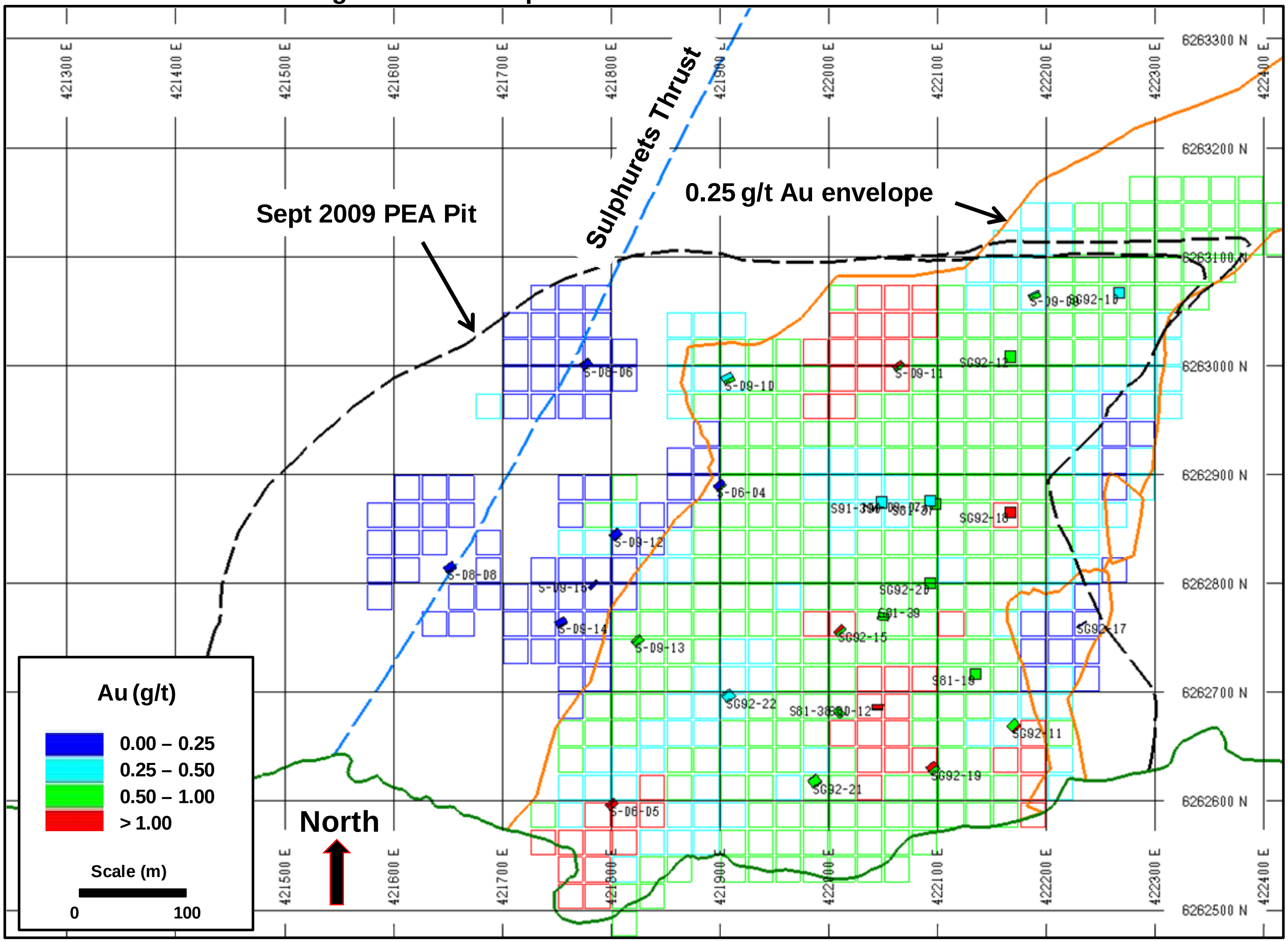


Figure 17-20: Sulphurets Cu Block Model - 1275 Level

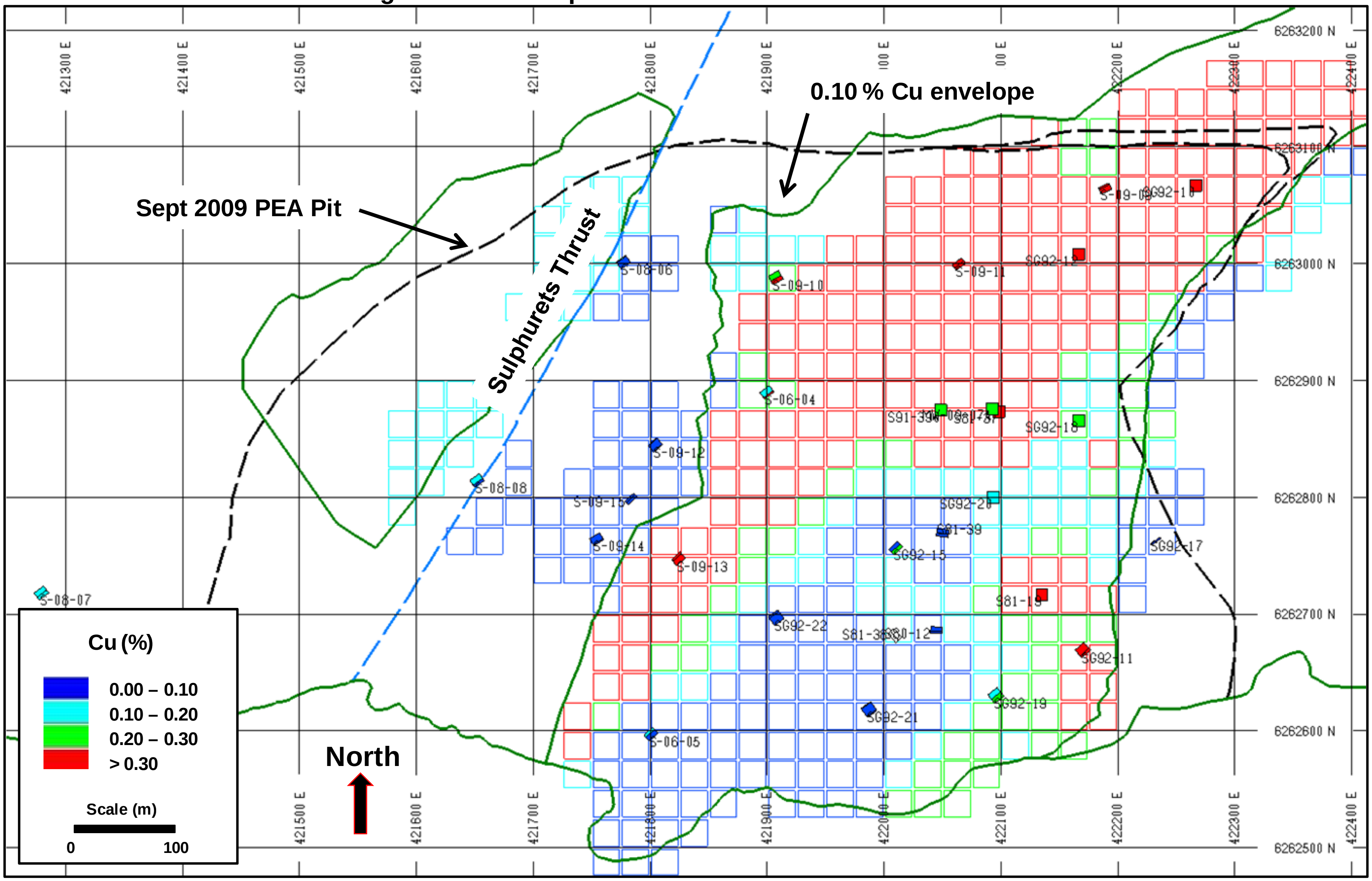


Figure 17-21: Mitchell Au Block Model Cross Section 11

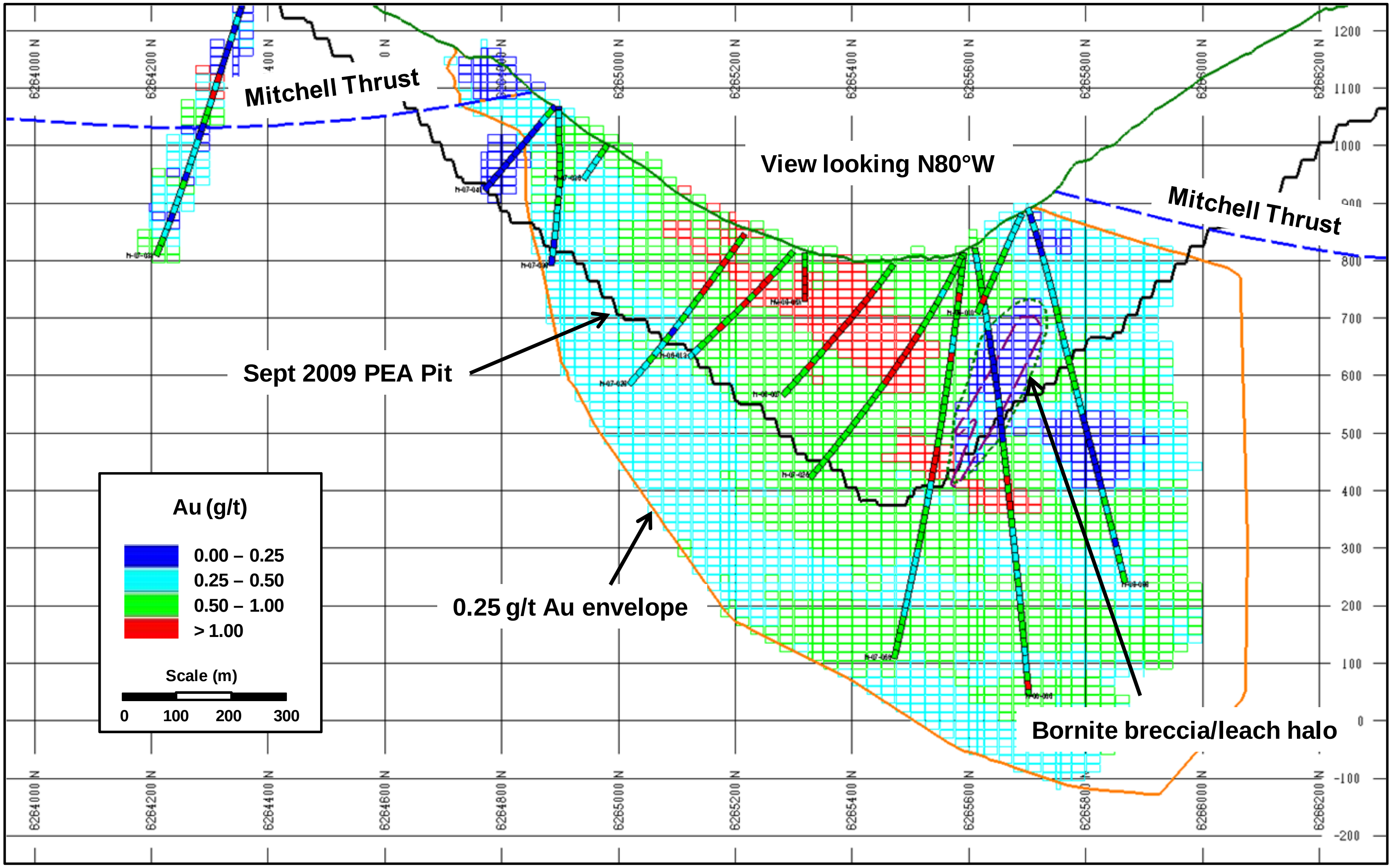


Figure 17-22: Mitchell Cu Block Model Cross Section 11

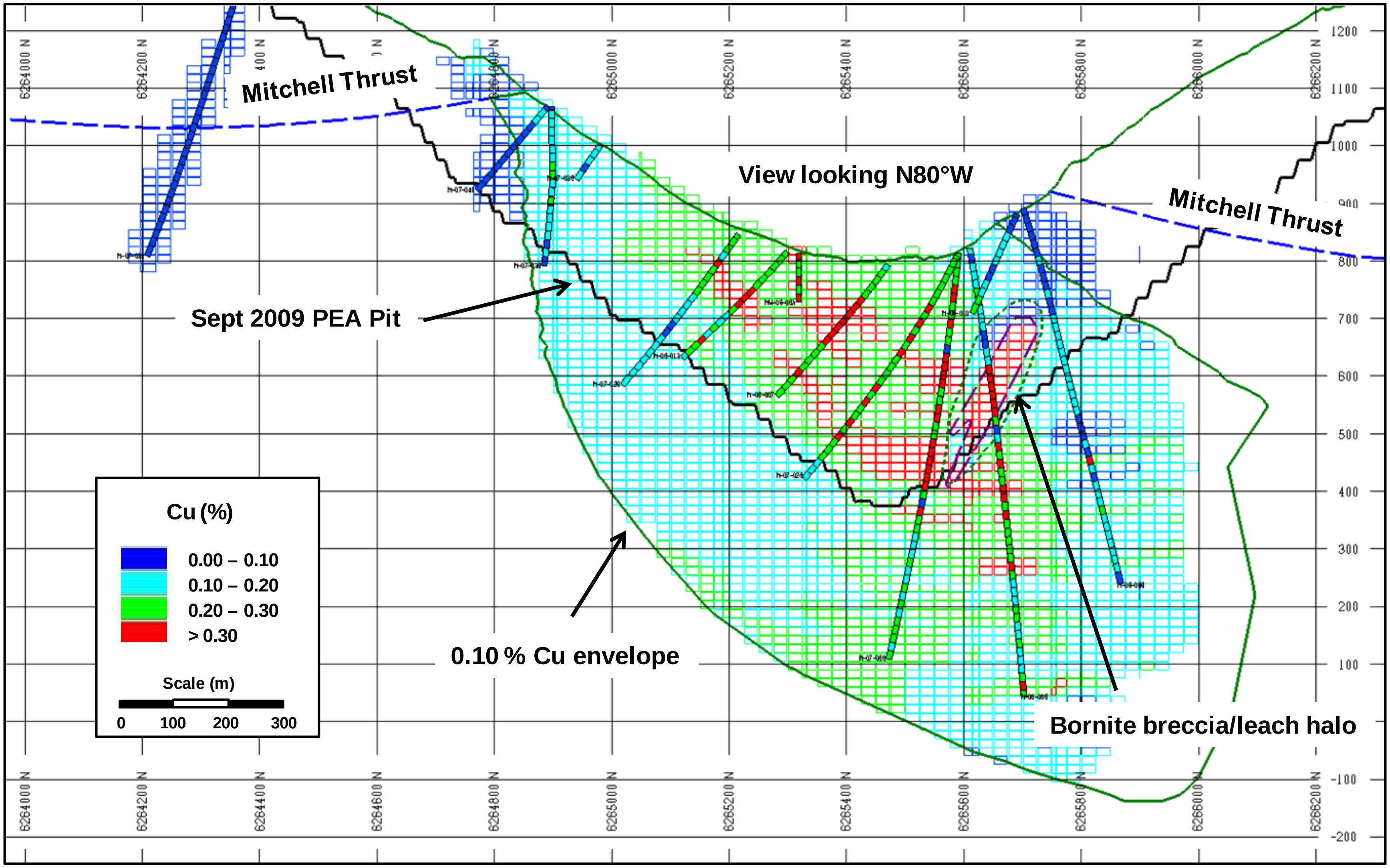


Figure 17-23: Mitchell Au Block Model - 660 Level

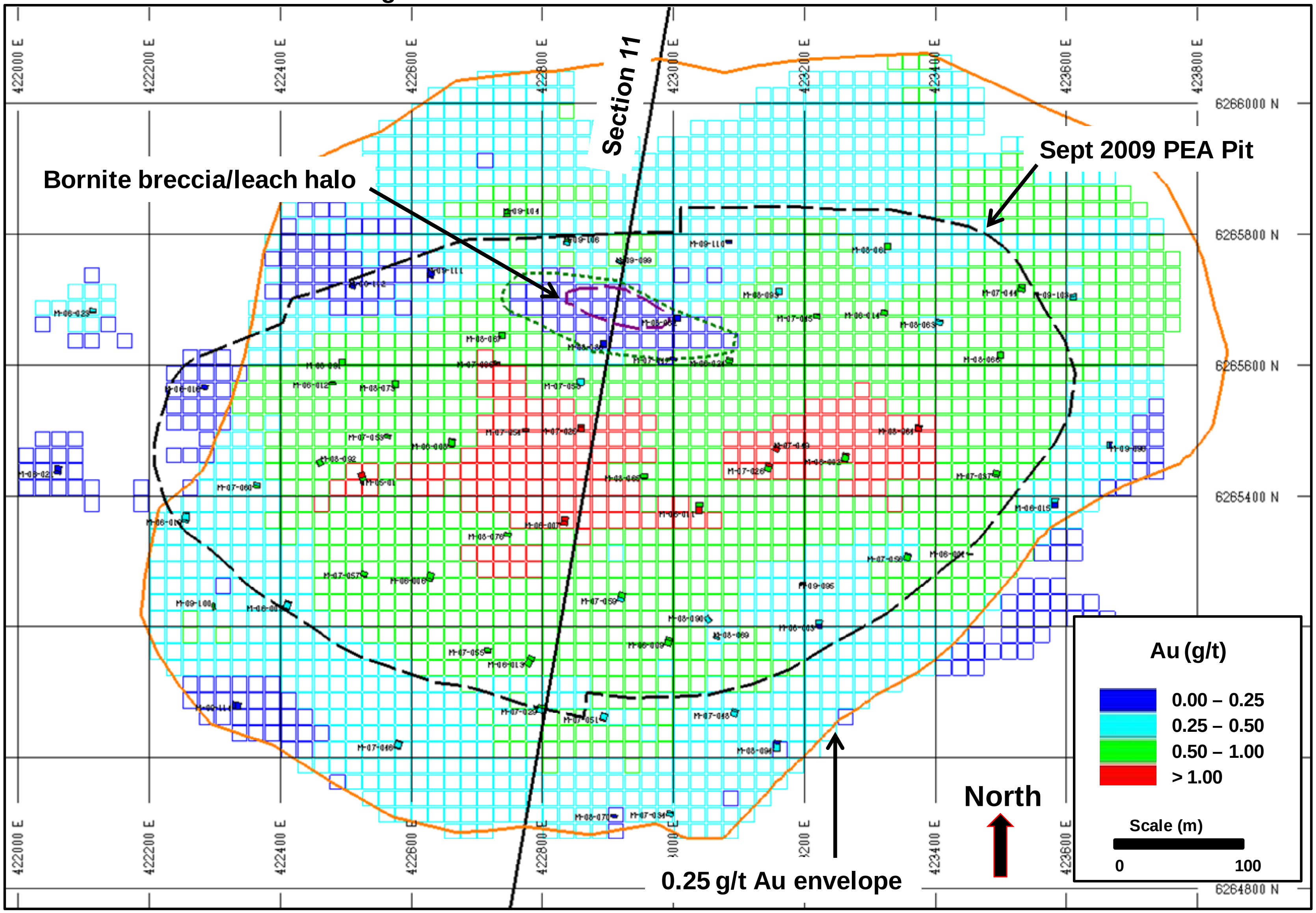
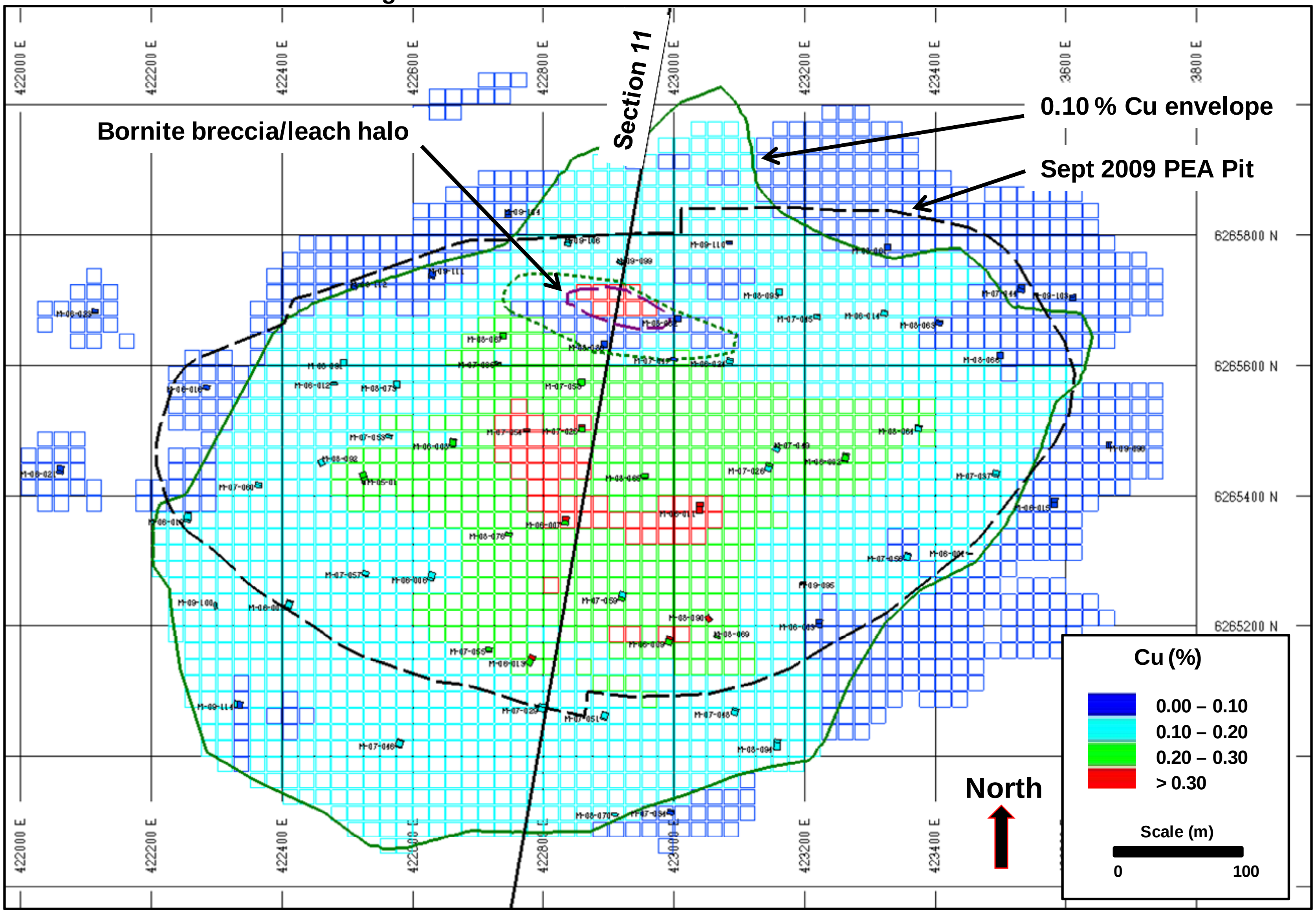


Figure 17-24: Mitchell Cu Block Model - 660 Level



The author generated nearest neighbor models for both gold and copper in order to check for potential global biases in the estimated block grades. Table 17-23 compares mean nearest neighbor and inverse distance gold and copper grades at a zero cutoff grade for Kerr, Sulphurets and Mitchell Measured (Mitchell only) and Indicated Mineral Resource blocks. Inferred resource comparisons are also tabulated in Table 17-23.

Table 17-23: Grade Model Bias Checks

Kerr Grade Models	Gold Model		Copper Model	
	Indicated	Inferred	Indicated	Inferred
	Mean Au (g/t)	Mean Au (g/t)	Mean Cu (%)	Mean Cu (%)
Nearest Neighbor	0.2625	0.1609	0.4439	0.1376
Inverse Distance	0.2626	0.1596	0.4482	0.1402
% Diff vs. NN Model	0.04%	-0.81%	0.97%	1.89%

Sulphurets Grade Models	Gold Model		Copper Model	
	Indicated	Inferred	Indicated	Inferred
	Mean Au (g/t)	Mean Au (g/t)	Mean Cu (%)	Mean Cu (%)
Nearest Neighbor	0.6205	0.3820	0.2800	0.1206
Inverse Distance	0.6116	0.3737	0.2751	0.1210
% Diff vs. NN Model	-1.43%	-2.17%	-1.75%	0.33%

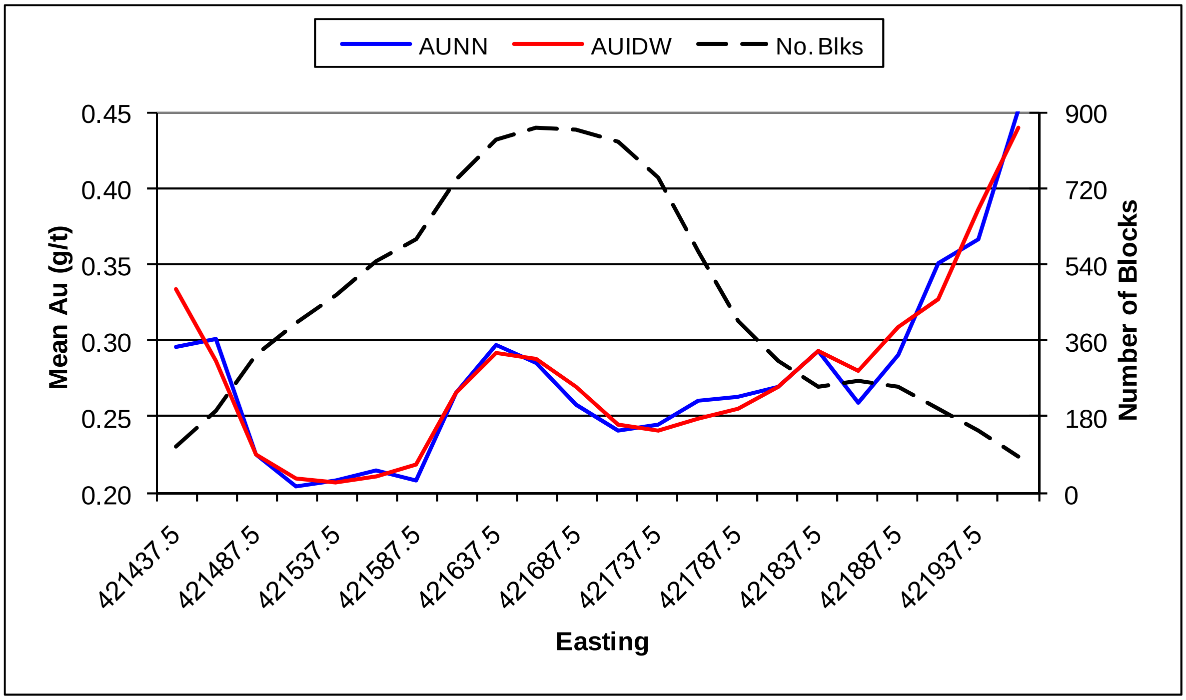
Mitchell Grade Models	Gold Model		Copper Model	
	M+I	Inferred	M+I	Inferred
	Mean Au (g/t)	Mean Au (g/t)	Mean Cu (%)	Mean Cu (%)
Nearest Neighbor	0.5821	0.3274	0.1640	0.1045
Inverse Distance	0.5827	0.3410	0.1644	0.1104
% Diff vs. NN Model	0.10%	4.15%	0.24%	5.65%

The results shown in Table 17-23 show that the inverse distance weighted (IDW) models compare very well with the nearest neighbor grades for the Measured+Indicated category. 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 (AUIDW and CUIDW) with nearest neighbor (AUNN and CUNN) estimates by block model columns (eastings), rows (northings), and levels (elevation). Gold and Copper swath plots for are shown in Figures 17-25 through 17-27 for the Kerr deposit by easting, northing, and elevation, respectively. Similar swath plots are shown for the Sulphurets and

Mitchell deposits as Figures 17-28 through 17-33. The number of blocks by the rows, columns, and levels are shown by the dashed black line and the units are read from the Y-axis on the right side of the plots.

Figure 17-25: Kerr Au-Cu Swath Plots by Eastings



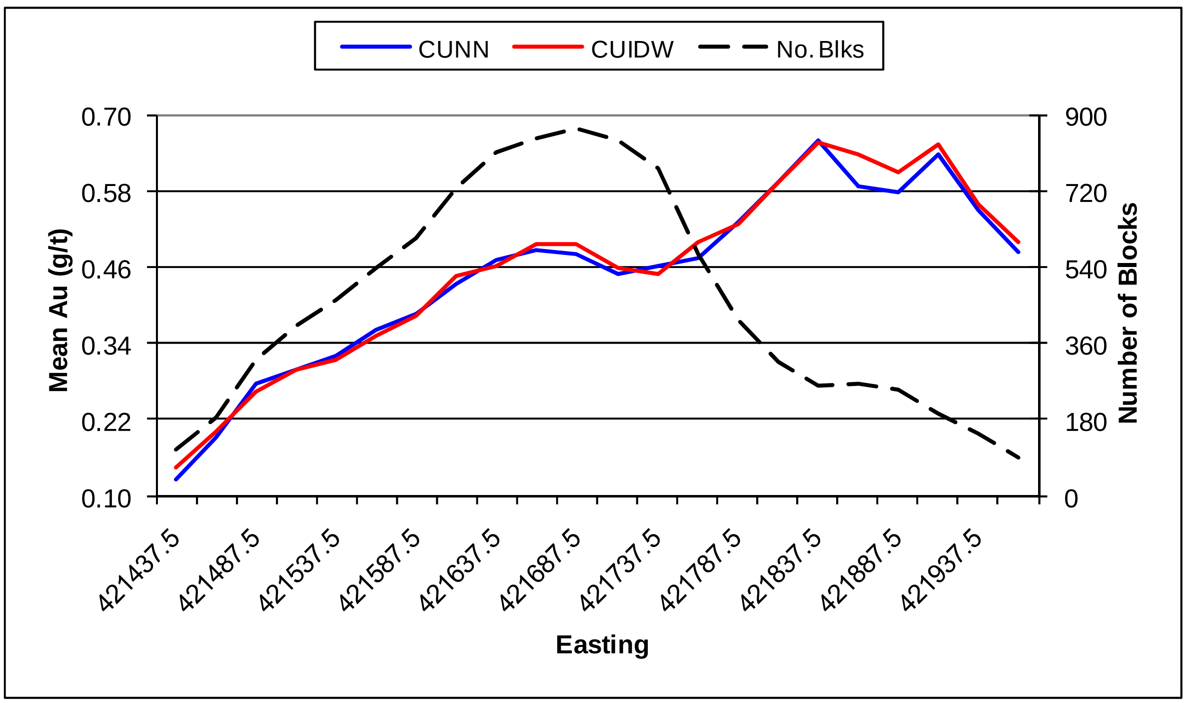
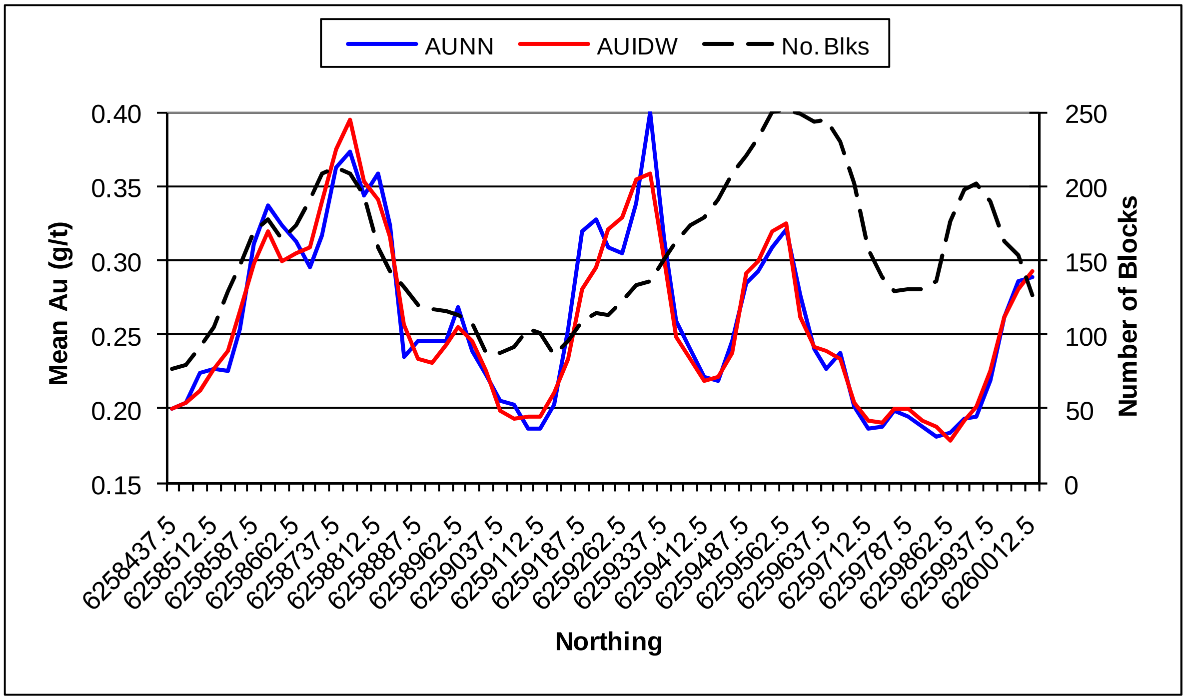


Figure 17-26: Kerr Au-Cu Swath Plots by Northings



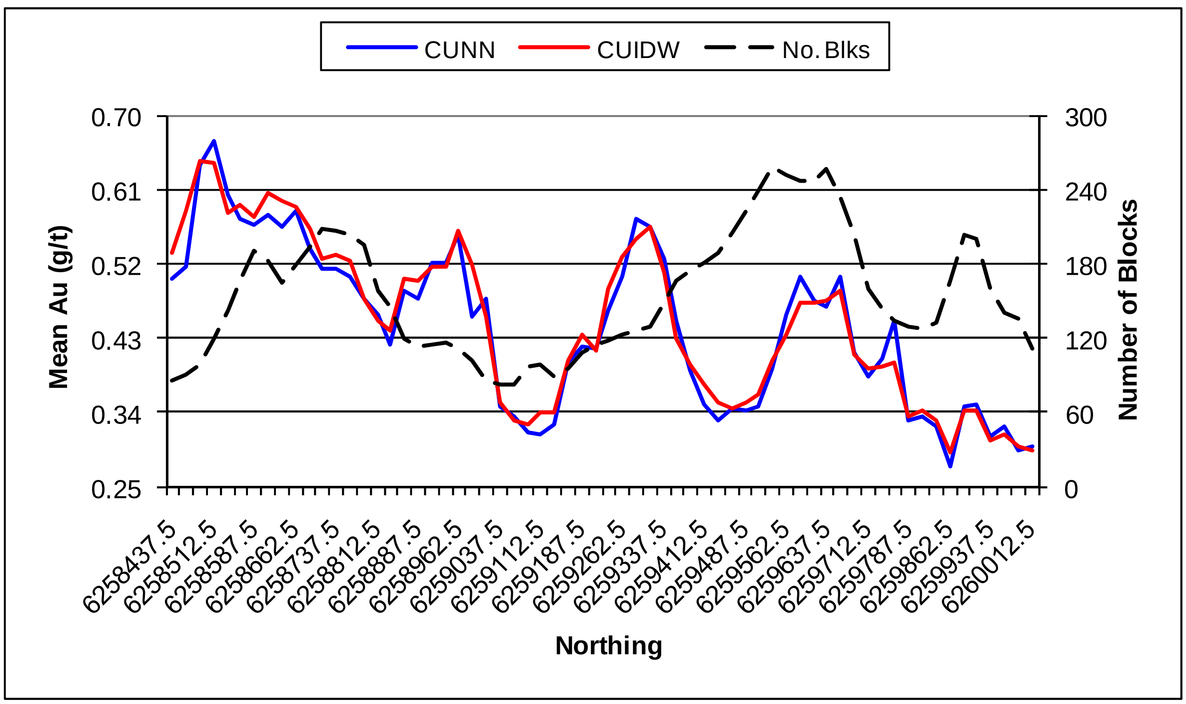
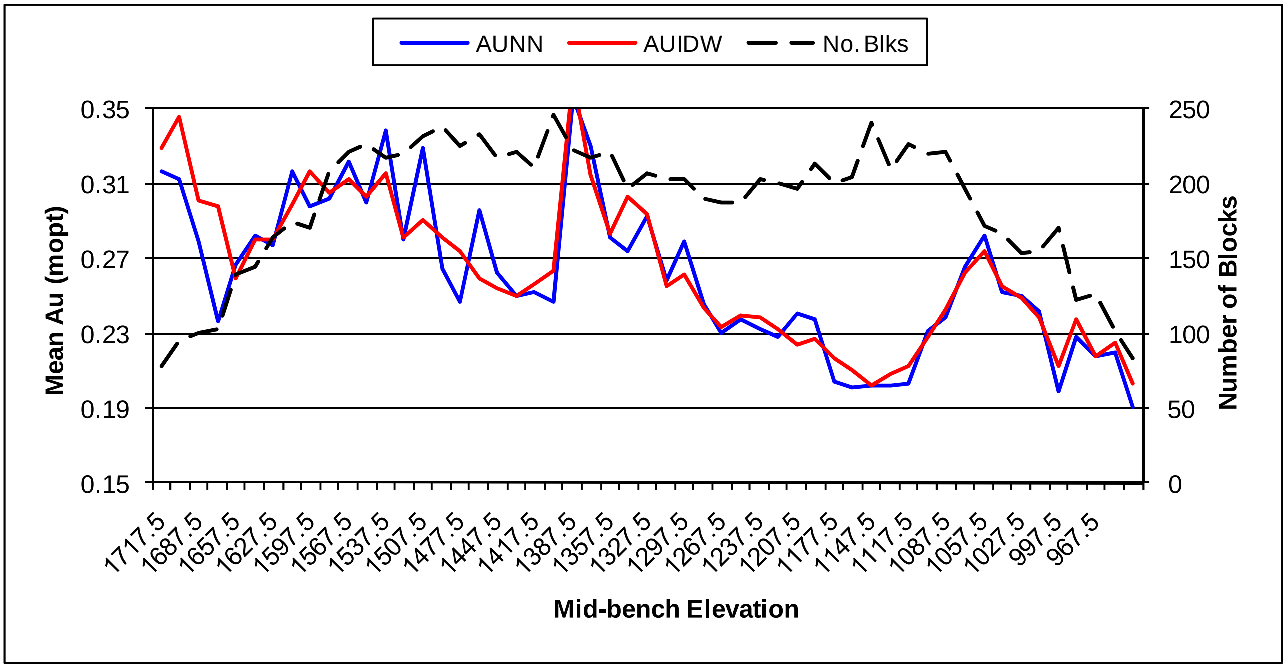


Figure 17-27: Kerr Au-Cu Swath Plots by Elevation



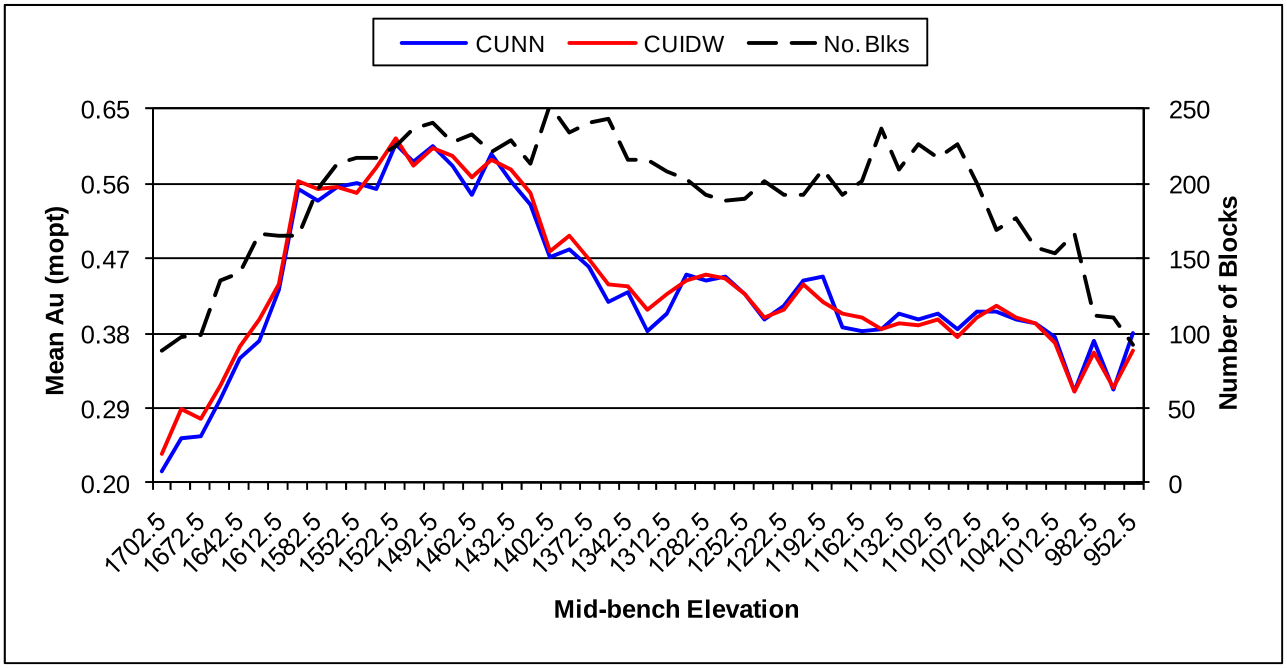
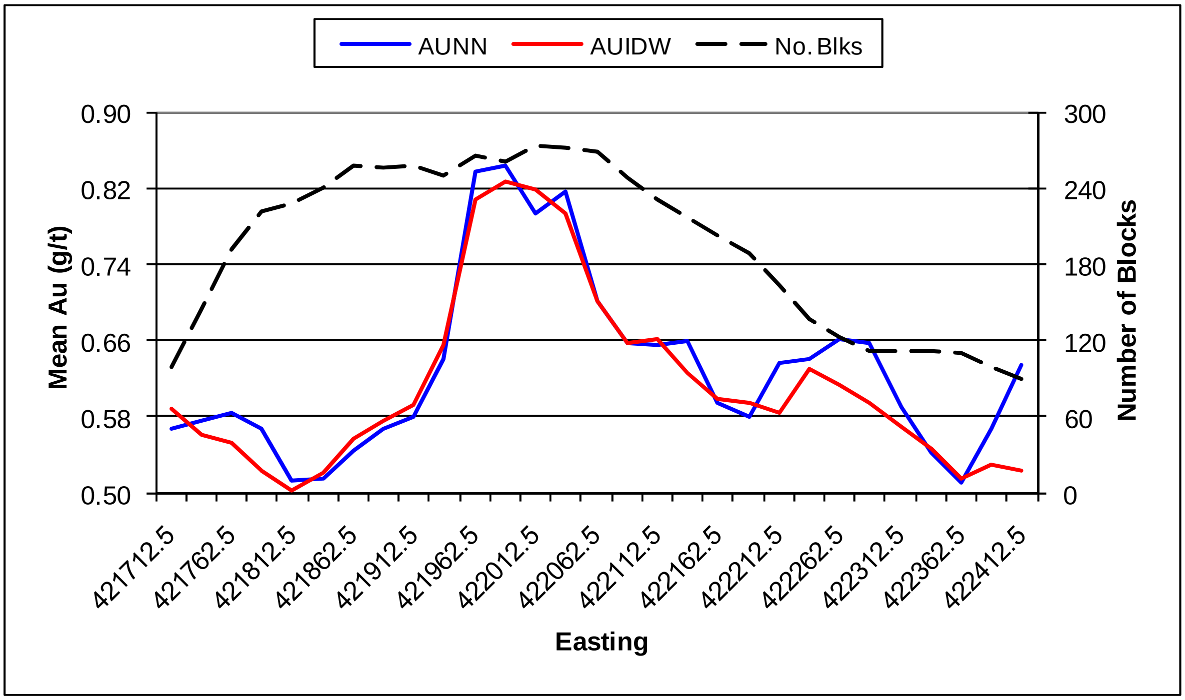


Figure 17-28: Sulphurets Au-Cu Swath Plots by Eastings



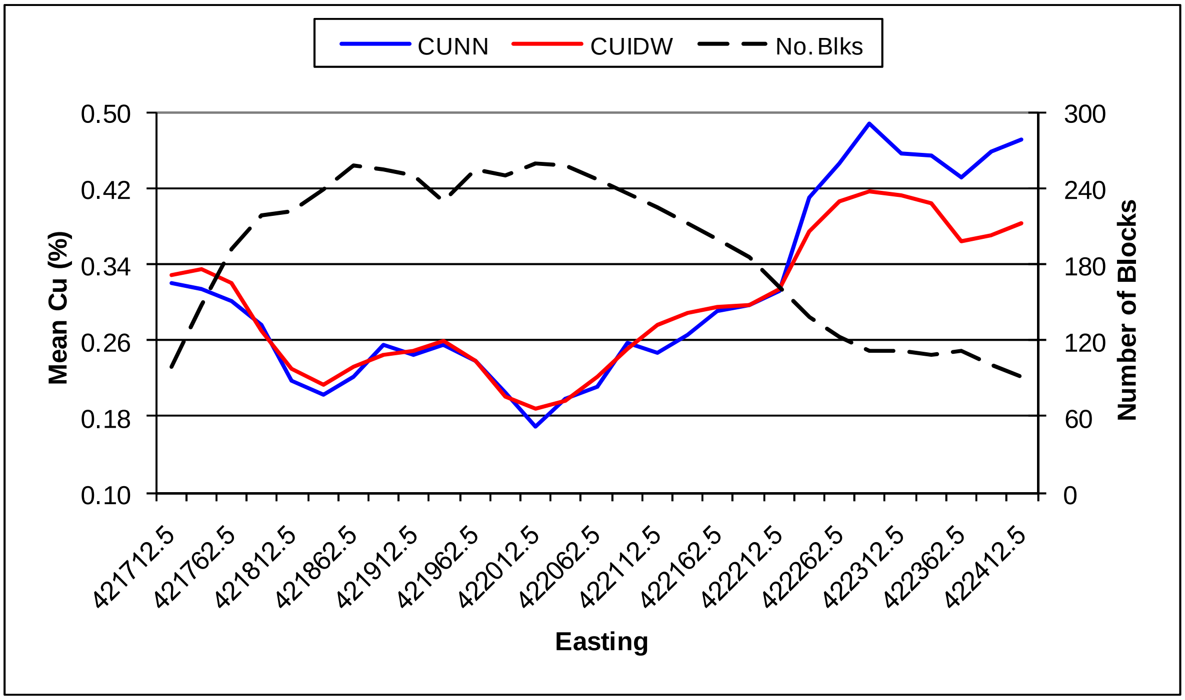
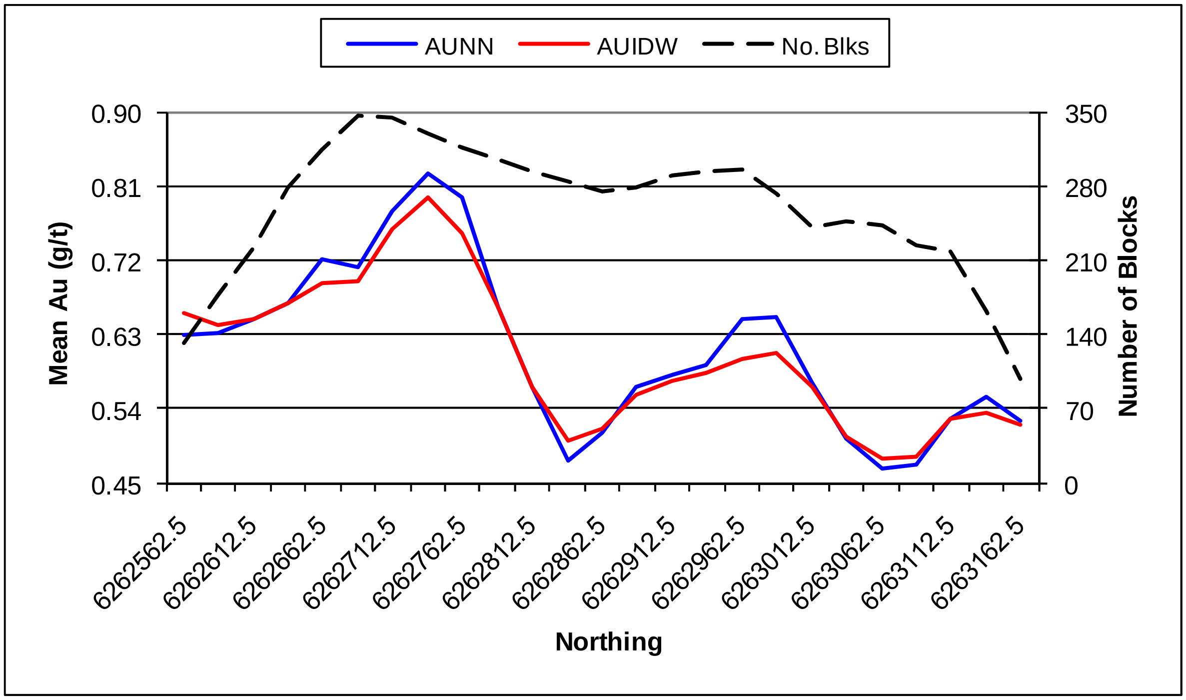


Figure 17-29: Sulphurets Au-Cu Swath Plots by Northings



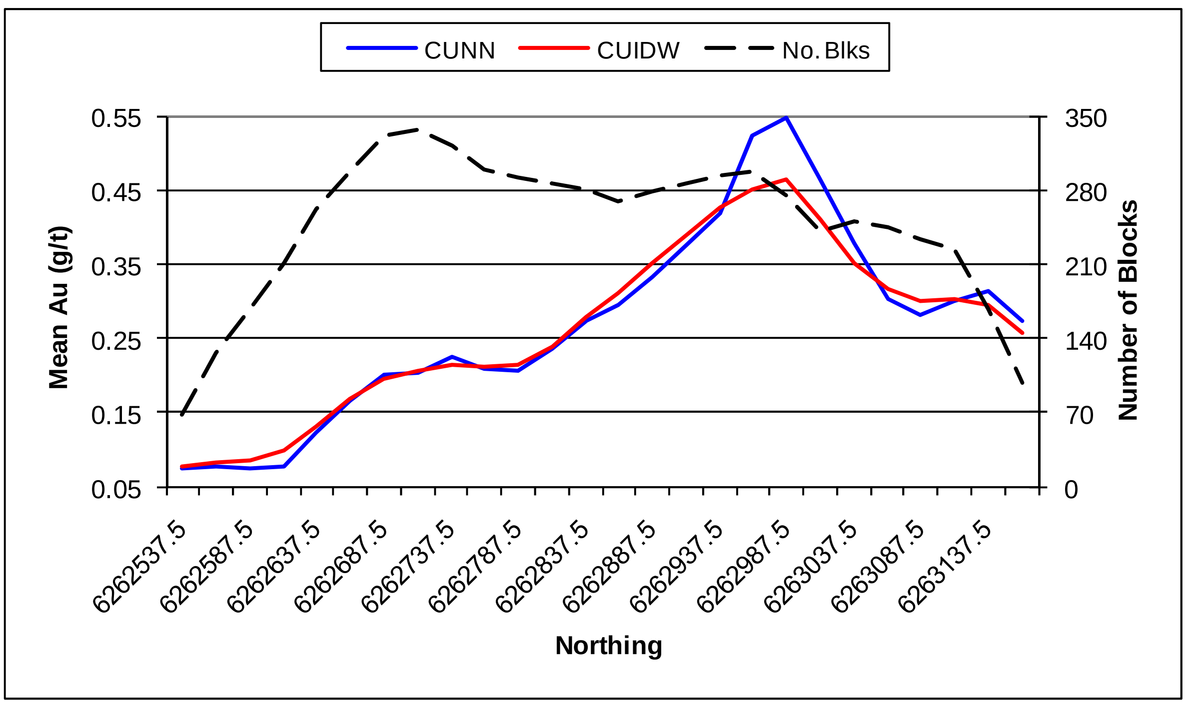
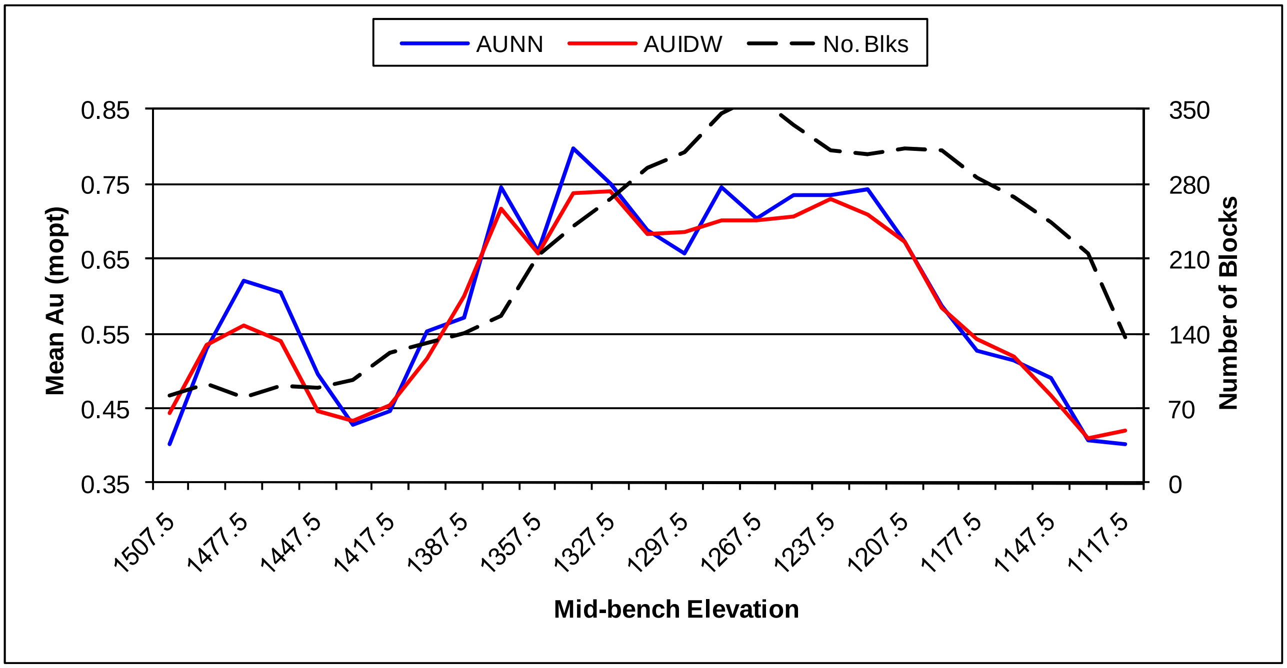


Figure 17-30: Sulphurets Au-Cu Swath Plots by Elevation



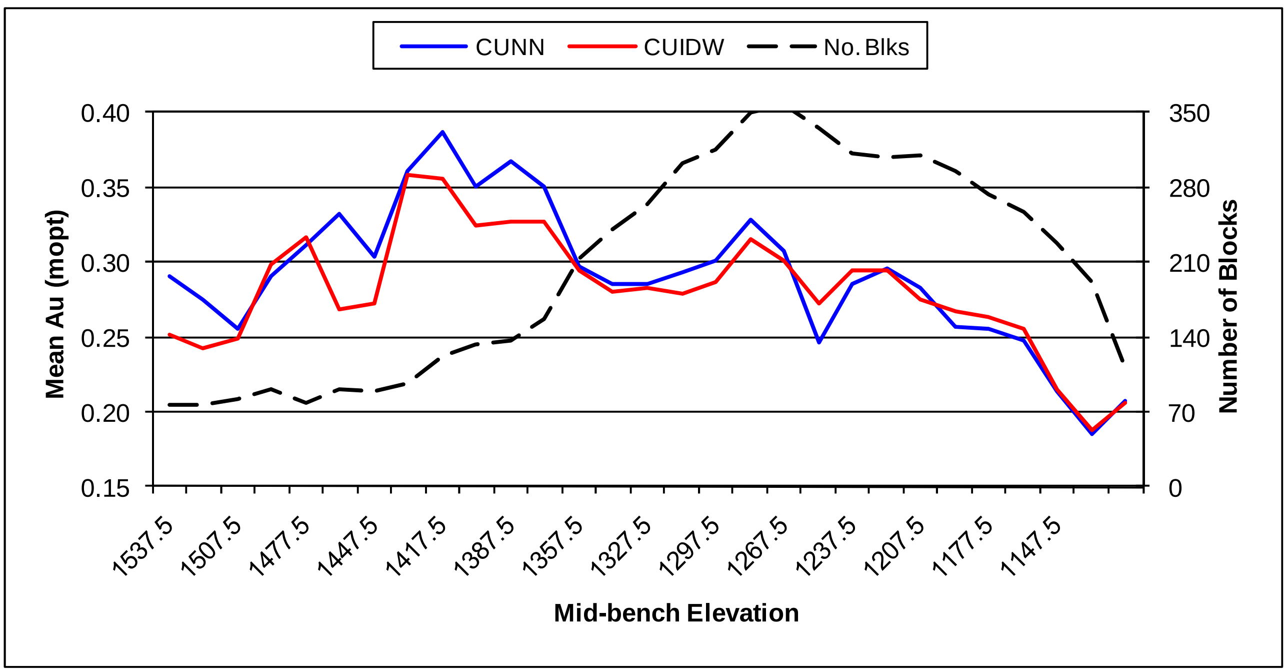
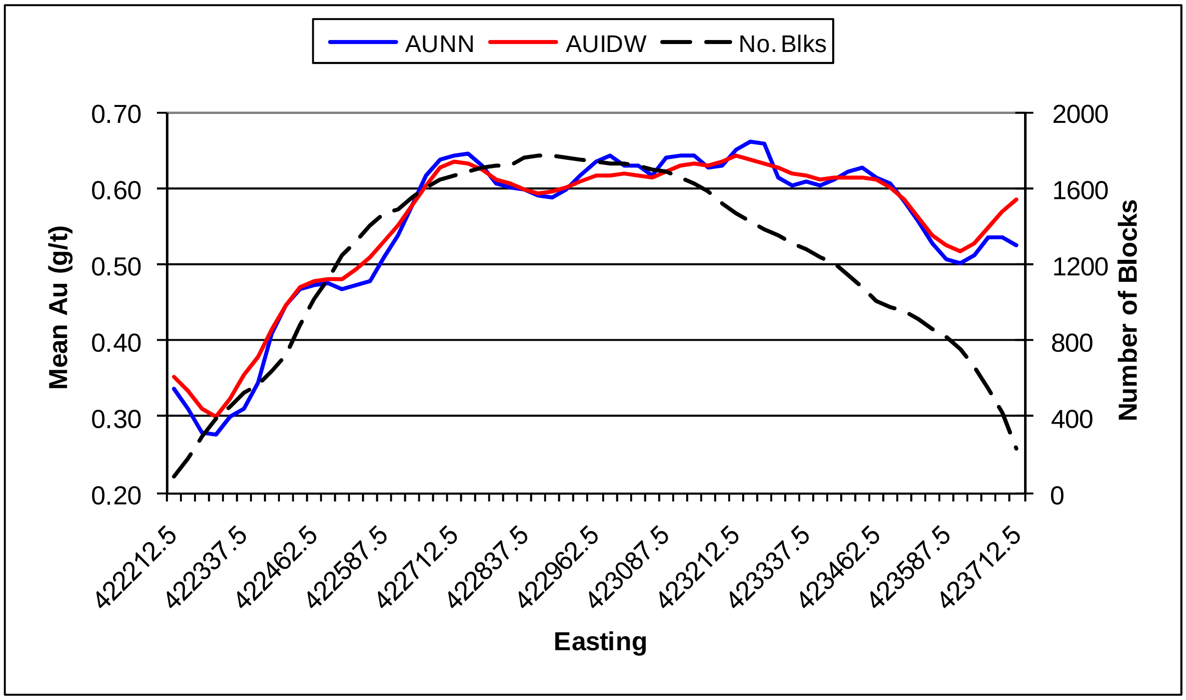


Figure 17-31: Mitchell Au-Cu Swath Plots by Eastings



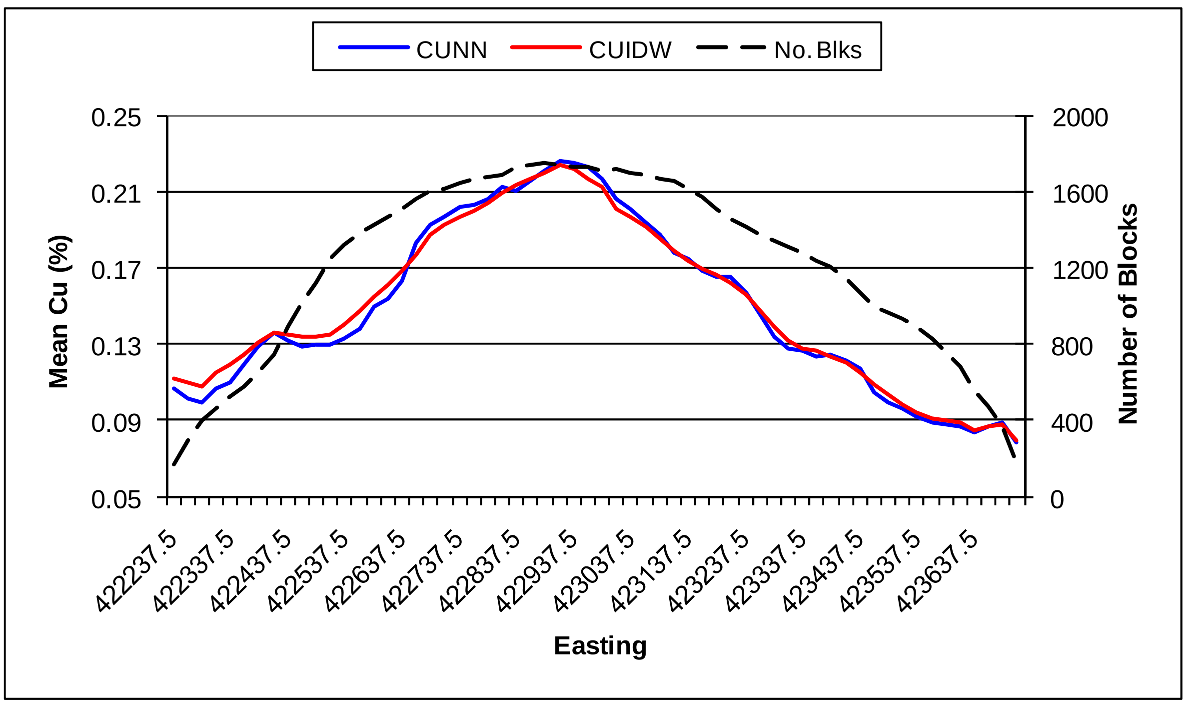
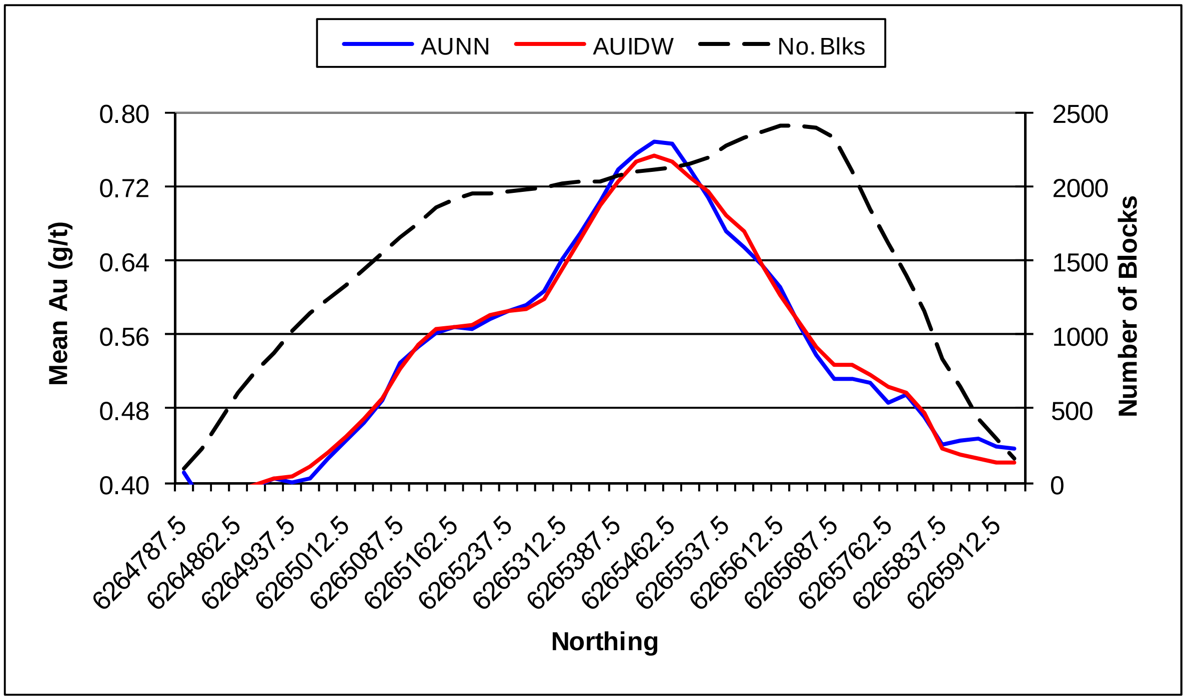


Figure 17-32: Mitchell Au-Cu Swath Plots by Northings



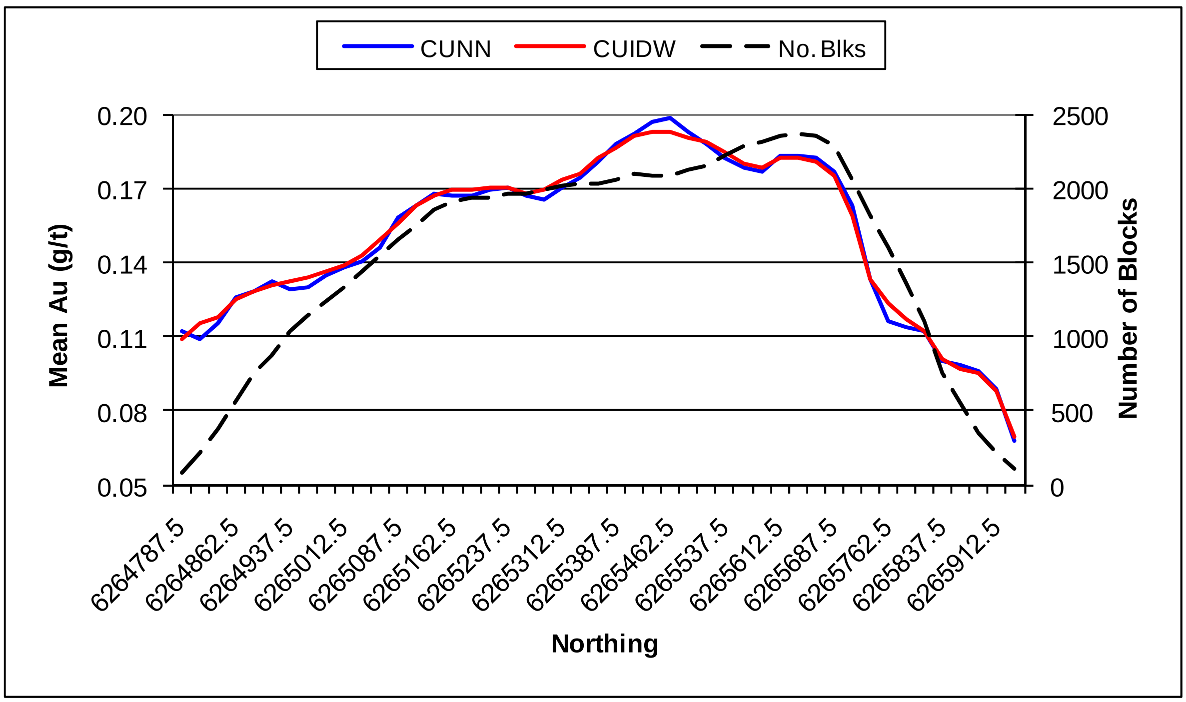
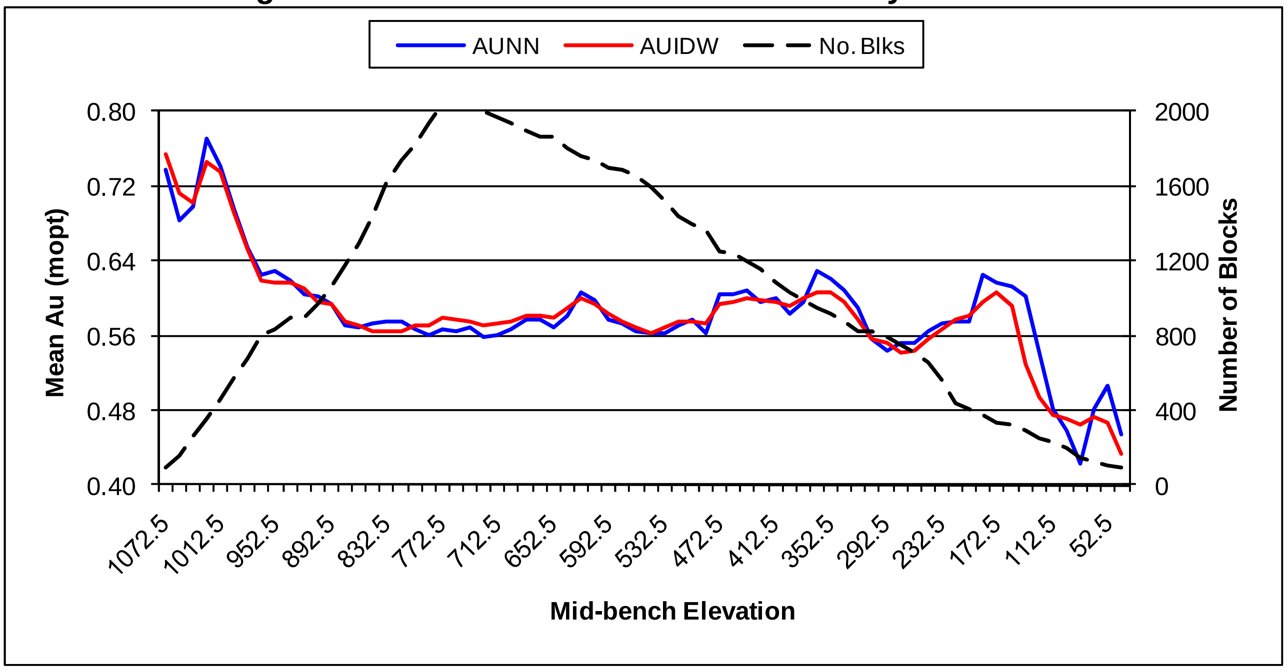
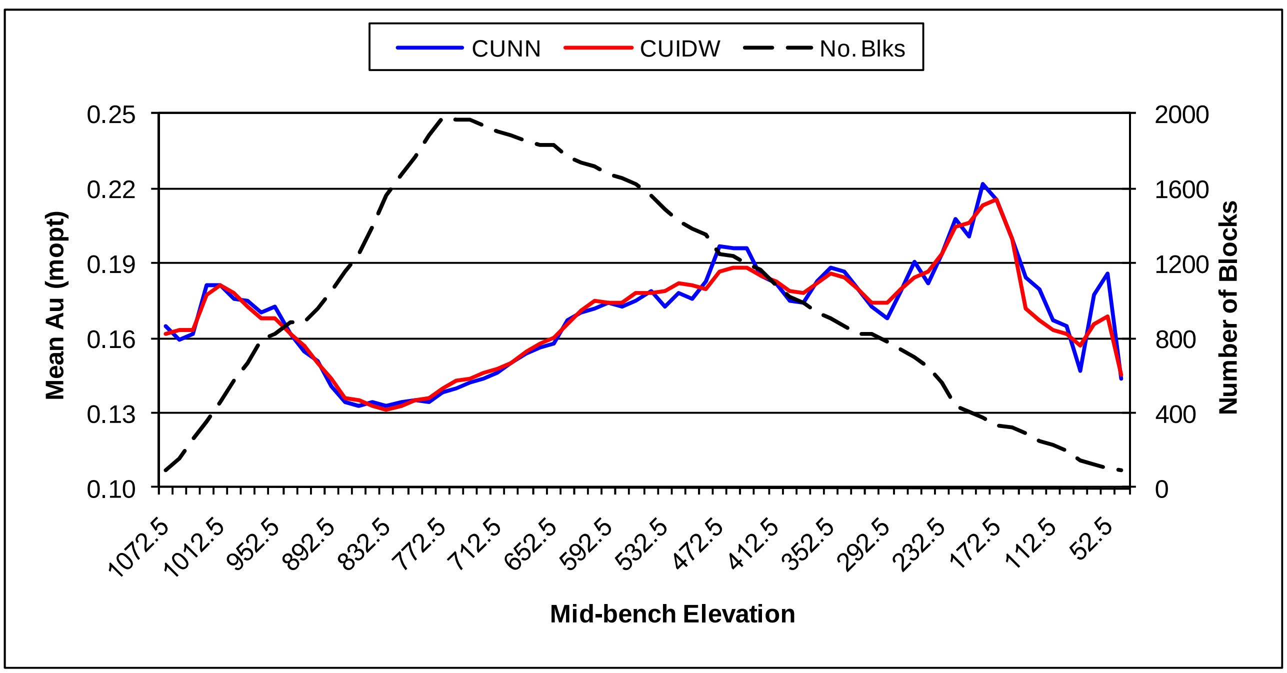


Figure 17-33: Mitchell Au-Cu Swath Plots by Elevation





In the author's opinion, the swath plots shown in Figures 17-13 through 17-8 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 and Mitchell models are globally unbiased and represent reasonable estimates of insitu block grades.

17.9 **Resource Classification**

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

RMI digitized shapes around mineralized drill holes for the Kerr, Sulphurets and Mitchell deposits. These shapes define areas where mineralized continuity has been well established by logging and sample results. To aid in the construction of the mineralized continuity shapes, RMI constructed probabilistic gold equivalent models using 0.38 g/t and 0.50 g/t cutoff grades. Blocks with an estimated probability is excess of 50% were considered to demonstrate mineralized continuity. The model blocks were coded with these manually constructed shapes and used to identify possible Measured (Mitchell only) and Indicated resources. Other criteria such as distance to data and the number of holes also had to be met before the blocks inside of the digitized shapes were classified as Indicated.

Indicated Mineral Resources were assigned to the Kerr and Sulphurets zones if the blocks were located inside of the digitized mineralized shape and if the blocks were estimated by two or more holes with the closest being within 75 meters of the block. For the Mitchell deposit, blocks were considered to be Indicated Mineral Resources if they were inside of the manually constructed shape and if they were estimated by two or more holes with the closest being within 125 meters of the block.

Measured Mineral Resources were 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 (i.e. not previously classified as Indicated or Measured) blocks for the Kerr, Sulphurets, and Mitchell zones. Distance to data and number of holes used to estimate grades were used in conjunction with whether the blocks were located inside of the gold or copper envelopes that were used in the estimation process.

For the Kerr deposit Inferred Resources were assigned to:

- unclassified blocks located inside of the gold or copper zones estimated by 2 or more holes, one of which is within 125 meters of the block
- unclassified blocks located outside of the gold or copper zones estimated by 1 or more holes within 75 meters

For the Sulphurets deposit Inferred Resources were assigned to:

- unclassified blocks located inside of the gold or copper zones estimated by 2 or more holes, one of which is within 75 meters of the block

- unclassified blocks located outside of the gold or copper zones estimated by 1 or more holes within 50 meters of the block

For the Mitchell deposit Inferred Resources were assigned to:

- unclassified upper plate blocks located inside of the gold or copper zones estimated by 2 or more holes, one of which is located within 175 meters of the block or 1 hole within 75 meters of the block

- unclassified upper plate blocks located outside of the gold or copper zone estimated by 1 or more holes, one of which is within 50 meters of the block

- lower plate blocks located inside of the gold or copper zones estimated by 2 or more holes, one of which is within 175 meters of the block or 1 hole within 75 meters of the block

- lower plate blocks located outside of the gold or copper zone estimated by 2 or more holes one of which is located within 75 meters of the block or 1 hole within 50 meters of the block

Two final passes were made to cleanup isolated Indicated or Inferred blocks. Indicated blocks surrounded by six (four full sides and four corners) or more Inferred blocks were downgraded to Inferred. Similarly, Inferred blocks surrounded by 6 or more Indicated blocks were upgraded to Indicated.

17.10 **Summary of Mineral Resources**

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

$$\text{Gold Equivalent Grade} = \text{Au (g/t)} + (\text{Cu (\%)} * 2.562)$$

Mineral Resources are summarized at a variety of gold equivalent cutoff grades in Tables 17-24 through 17-26 for the Kerr, Sulphurets, and Mitchell deposits, respectively. A gold equivalent cutoff grade of 0.50 g/t has been selected for disclosing Mineral Resources as highlighted in yellow in Tables 17-24 through 17-26.

Table 17-24: Summary of Mineral Resources - Kerr

AuEQV Cutoff (g/t)	Indicated Mineral Resources						Inferred Mineral Resources					
	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)
0.3	256,300	1.42	0.26	0.45	2,142	2,542	139,800	0.70	0.17	0.20	764	616
0.4	248,400	1.45	0.26	0.46	2,076	2,518	101,500	0.83	0.19	0.25	620	559
0.5	**237,500**	**1.50**	**0.26**	**0.48**	**1,985**	**2,513**	**76,100**	**0.96**	**0.20**	**0.30**	**489**	**503**
0.6	221,600	1.56	0.27	0.51	1,924	2,491	59,500	1.07	0.20	0.34	383	446
0.7	208,700	1.62	0.27	0.53	1,812	2,438	46,300	1.20	0.21	0.38	313	388
0.8	193,800	1.69	0.27	0.55	1,682	2,349	37,500	1.30	0.22	0.42	265	347
0.9	178,900	1.76	0.28	0.58	1,610	2,287	29,800	1.42	0.23	0.46	220	302
1.0	164,400	1.83	0.28	0.60	1,480	2,174	24,000	1.53	0.23	0.51	177	270

Table 17-25: Summary of Mineral Resources - Sulphurets

AuEQV Cutoff (g/t)	Indicated Mineral Resources						Inferred Mineral Resources					
	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)
0.3	166,200	1.31	0.61	0.27	3,260	989	203,900	0.76	0.42	0.13	2,729	593
0.4	163,500	1.33	0.62	0.28	3,259	1,009	173,700	0.83	0.46	0.14	2,577	547
0.5	**159,000**	**1.35**	**0.63**	**0.28**	**3,221**	**981**	**144,000**	**0.91**	**0.50**	**0.16**	**2,317**	**511**
0.6	148,800	1.41	0.65	0.30	3,110	984	113,700	1.00	0.55	0.18	2,006	446
0.7	139,100	1.46	0.66	0.31	2,952	950	85,800	1.11	0.62	0.20	1,701	371
0.8	129,700	1.51	0.68	0.32	2,836	915	64,100	1.24	0.66	0.22	1,364	316
0.9	118,600	1.57	0.70	0.34	2,669	889	50,600	1.34	0.71	0.25	1,151	281
1.0	107,600	1.63	0.72	0.36	2,491	854	40,700	1.45	0.74	0.27	972	244

Table 17-26: Summary of Mineral Resources - Mitchell

AuEQV Cutoff (g/t)	Measured Mineral Resources						Indicated Mineral Resources					
	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)
0.3	710,400	1.04	0.62	0.17	14,161	2,662	1,158,300	0.97	0.56	0.16	20,855	4,085
0.4	695,600	1.05	0.62	0.17	13,866	2,606	1,135,200	0.99	0.57	0.16	20,804	4,003
0.5	**659,700**	**1.09**	**0.64**	**0.17**	**13,574**	**2,472**	**1,080,900**	**1.01**	**0.58**	**0.17**	**20,156**	**4,050**
0.6	608,200	1.13	0.67	0.18	13,101	2,413	994,000	1.05	0.60	0.18	19,175	3,943
0.7	547,400	1.18	0.70	0.19	12,320	2,292	871,400	1.11	0.64	0.19	17,930	3,649
0.8	487,100	1.24	0.73	0.20	11,432	2,147	740,900	1.17	0.67	0.20	15,960	3,266
0.9	420,800	1.30	0.77	0.21	10,417	1,948	601,700	1.25	0.71	0.21	13,735	2,785
1.0	352,600	1.37	0.81	0.22	9,182	1,710	472,700	1.33	0.76	0.22	11,550	2,292

AuEQV Cutoff (g/t)	Measured+Indicated Mineral Resources						Inferred Mineral Resources					
	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)	Tonnes (000)	AuEQV (g/t)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (M)
0.3	1,868,700	1.00	0.58	0.16	35,015	6,746	860,100	0.64	0.37	0.11	10,232	2,085
0.4	1,830,800	1.01	0.59	0.16	34,669	6,609	706,500	0.71	0.40	0.12	9,086	1,869
0.5	**1,740,600**	**1.04**	**0.60**	**0.17**	**33,730**	**6,522**	**537,000**	**0.79**	**0.44**	**0.14**	**7,597**	**1,657**
0.6	1,602,200	1.08	0.63	0.18	32,276	6,356	423,000	0.85	0.47	0.15	6,392	1,398
0.7	1,418,800	1.14	0.66	0.19	30,250	5,941	325,100	0.92	0.50	0.16	5,226	1,146
0.8	1,228,000	1.20	0.69	0.20	27,392	5,413	226,300	0.99	0.54	0.18	3,929	898
0.9	1,022,500	1.27	0.73	0.21	24,152	4,733	140,800	1.08	0.58	0.19	2,626	590
1.0	825,300	1.35	0.78	0.22	20,733	4,002	81,300	1.17	0.61	0.22	1,594	394

17.11 **Conceptual Pit Results**

The Mineral Resources summarized in Tables 17-24 through 17-26 were tabulated as “global resources” using gold equivalent cutoff grades. As a preliminary test to determine “reasonable expectations for economic viability”, the author generated three conceptual pits for the Kerr, Sulphurets, and Mitchell deposits using the Lerchs-Grossmann algorithm. In addition to a "base case" pit a "pessimistic" and "optimistic" case was also run. Measured, Indicated, and Inferred Mineral Resources were used for all three cases. For the base case conceptual pits, gold and copper prices of US $900 per ounce and US $2.50 per pound were used. Other key parameters that were used to generate the conceptual pits are summarized in Table 17-27.

Table 17-27: Conceptual Pit Parameters

Parameters	Base Case	Pessimistic Case	Optimistic Case
Au Price (US$/troy ounce)	$900	$720	$1,080
Cu Price (US$/pound)	$2.50	$2.00	$3.00
Ag Price (US$/troy ounce)	$17	$14	$20
Mo Price (US$/pound)	$15.00	$12	$18
Au Recovery	78%	70%	80%
Cu Recovery	85%	80%	90%
Ag Recovery	73%	65%	75%
Mo Recovery	50%	40%	60%
Mining Cost (US$/tonne mined)	$1.50	$1.80	$1.20
Processing Cost (US$/ore tonne)	$4.96	$5.95	$3.97
G&A Cost (US$/ore tonne)	$0.89	$1.07	$0.71
Tailings & Water Management (US$/ore tonne)	$0.70	$0.84	$0.56
Slope Angle (degrees)	45	40	45

The categorized Measured, Indicated and Inferred Mineral Resources inside of the Kerr, Sulphurets, and Mitchell conceptual pits are summarized using a 0.50 g/t Au equivalent cutoff grade in Tables 17-28 through 17-30, respectively.

Table 17-28: Kerr Conceptual Pit Results

LG Pit	Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)
Base Case	232,500	0.26	0.49	1,944	2,511	63,500	0.21	0.30	429	420
Pessimistic Case	204,600	0.27	0.50	1,776	2,255	43,200	0.24	0.30	333	286
Optimistic Case	235,800	0.26	0.48	1,971	2,495	68,700	0.20	0.30	442	454

Table 17-29: Sulphurets Conceptual Pit Results

LG Pit	Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)
Base Case	158,600	0.63	0.28	3,212	979	130,800	0.49	0.16	2,061	461
Pessimistic Case	148,200	0.64	0.29	3,049	947	84,000	0.45	0.18	1,215	333
Optimistic Case	158,800	0.63	0.28	3,216	980	138,200	0.49	0.16	2,177	487

Table 17-30: Mitchell Conceptual Pit Results

LG Pit	Measured Mineral Resources					Indicated Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)
Base Case	651,300	0.65	0.17	13,611	2,440	1,056,800	0.59	0.17	20,046	3,960
Pessimistic Case	469,400	0.69	0.18	10,413	1,862	556,400	0.64	0.17	11,449	2,085
Optimistic Case	659,400	0.64	0.17	13,568	2,471	1,080,100	0.58	0.17	20,141	4,047

LG Pit	Measured + Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)	Tonnes (000)	Au (g/t)	Cu (%)	Au Ozs (000)	Cu Lbs (millions)
Base Case	1,708,100	0.61	0.17	33,657	6,400	349,200	0.42	0.14	4,715	1,077
Pessimistic Case	1,025,800	0.66	0.17	21,862	3,947	118,900	0.31	0.14	1,185	367
Optimistic Case	1,739,500	0.60	0.17	33,709	6,518	457,800	0.44	0.14	6,476	1,413

The author notes that in general, the base case conceptual pits capture nearly all of the Measured and/or Indicated Mineral Resources in each of the models (98% for Kerr, nearly 100% for Sulphurets, and 98% for Mitchell). The base case Kerr, Sulphurets, and Mitchell LG pits captured 83%, 91%, and 65% of the Inferred material, respectively. The optimistic Kerr, Sulphurets, and Mitchell LG pits captured 90%, 96%, and 85% of the Inferred material. Ongoing work will refine mining and processing costs for future conceptual pits that will be developed later this year.

17.12 Risks and Uncertainties

At this juncture, the author is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the estimate of Mineral Resources. A Preliminary Economic Assessment (PEA) was updated for this project in September 2009 (Wardrop, 2009). The updated Mineral Resources that are subject of this report will be used to prepare a pre-feasibility study that is anticipated to be completed by the end of the first quarter of 2010.

This project is still relatively early in the pre-development stage with numerous studies that are ongoing with significant additional work that will be required in defining the limits of the deposit, identifying possible processing methods, and if warranted, permitting the project into a developing/producing property. Mining and processing costs, metal recovery, and permitting could have a material effect on the viability and possible size of this deposit. It is too early to determine the potential impact of these topics on the ultimate size or viability of this project.

18.0 OTHER RELEVANT DATA AND INFORMATION

Seabridge has retained Rescan, a leading environmental permitting consulting company located in Vancouver, B.C., to begin the process of collecting baseline data and preparing all of the documents necessary for obtaining various permits.

The following information was excerpted from the September 2009 PEA (Wardrop, 2009) 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.

Baseline environmental data are not publically available for the area. Seabridge has engaged Rescan Environmental Services Ltd., a Vancouver-based consulting firm with extensive mining-related environmental assessment experience in BC, to undertake the baseline studies required for an environmental assessment of the Project

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). Two field seasons of baseline data collection have been completed as of December 2009 and specific programs will continue thru 2010.

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 licenses and permits required for the KSM Project. Figure 18-2 outlines the approval schedule for the Project up to the issuance of high level federal and provincial approvals in principle.

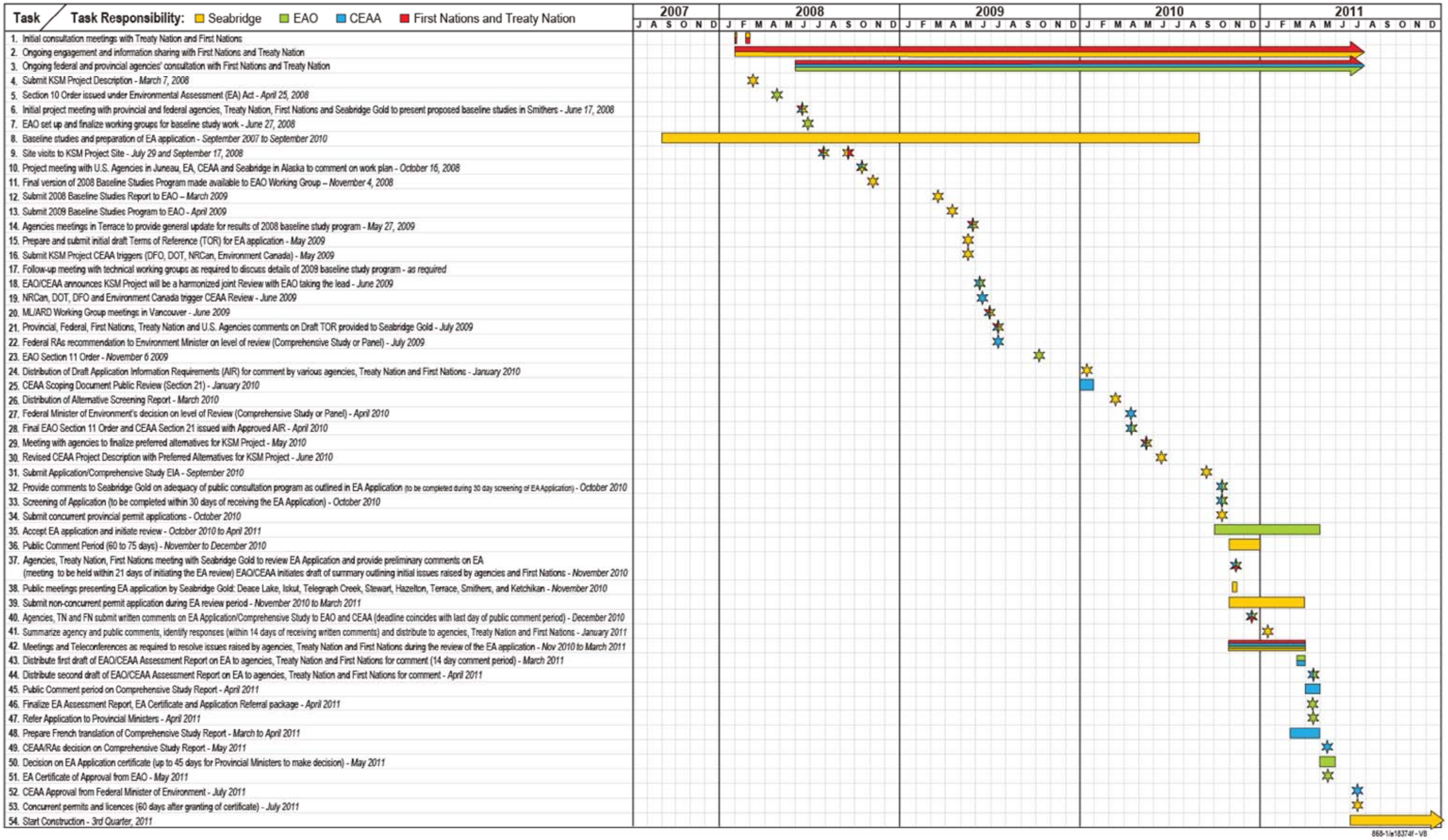
British Columbia Environmental Assessment Act Process

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 require 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.

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

Lists of 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 list of government approvals represents 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, Licences 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 Environmental Assessment Certificate. At this time it is too early to ascertain whether Seabridge will seek concurrent approvals under the BCEAA process. 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 British Columbia 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 areas 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*

19.0 INTERPRETATION AND CONCLUSIONS

19.1 **2009 Data Collection and Results**

In 2009, Seabridge completed a 14,000 meter core drilling program at KSM with the primary objective to upgrade resources to a level required for a reserve calculation. A portion of the 2009 drilling program was dedicated to geotechnical and hydrologic studies. The author combined the 2009 drilling data (43 holes totaling about 12,500 meters) with previously collected data to update the KSM resource estimate.

Seabridge investigated drill hole collar locations in 2009 by re-surveying pre-2009 collar locations. In addition to their own re-surveying program, an independent survey contractor (McGladrey & Associates). The results from these surveys confirm minor differences in the collar easting, northing, and elevations which are not thought to be material given the scale of the deposit.

Additional bulk density determinations collected by Seabridge from the 2009 core holes supports the density values that have been used for the past several years.

Various geotechnical data were collected in 2009 for determining pit wall angles and various infrastructure stability (tailings, waste disposal areas, etc). Similarly several holes were drilled adjacent to each mineralized zone for hydrologic monitoring and sampling. Baseline environmental data collection continued in 2009. The results from these studies are currently being evaluated and will be used in the development of a Pre-feasibility Study that is scheduled to be completed at the end of the first quarter of 2010.

The geologic interpretation of the Sulphurets and Mitchell deposits was updated by Seabridge personnel incorporating the new drilling information. In general the overall interpretation of the deposits has not changed. Independent gold and copper grade envelopes were developed for each mineralized zone and used as the primary control for estimating block grades.

Various exploratory data analyses were completed by the author to establish modeling populations, grade capping limits, and spatial relationships. Gold, silver and copper grades were estimated by inverse distance and nearest neighbor methods by the author. The estimated grades were validated by visual and statistical methods and in the opinion of the author, are globally unbiased and locally consistent with the current drill hole data. Molybdenum grades were also estimated for the Sulphurets and Mitchell zones using inverse distance methods.

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.

19.2 **Data Density and Reliability**

Seven holes were drilled by Seabridge within the Kerr zone in 2009 totaling about 1,150 meters. These holes were primarily drilled in the northern and southern ends of the deposit to infill areas of wider spaced drilling and to confirm prior Placer Dome drill results. The drill hole spacing at Kerr varies between 50 and 75 meters within the main zone of recognized mineralization.

Eleven holes were drilled by Seabridge in 2009 within the Sulphurets zone totaling about 3,500 meters. The majority of these holes were drilled along the western flank of the west dipping mineralized zone below the Sulphurets thrust fault in order to upgrade Inferred resources to an Indicated category. Several holes also tested the Main Copper zone, which is located immediately above the Sulphurets thrust fault. The drill hole spacing at Sulphurets is quite variable. Within the currently recognized "core" of mineralization, drill hole spacing varies between 50 and 75 meters. Outside of the more tightly drilled area the drill hole spacing is much wider and will require additional infill drilling in order to increase the overall confidence level for upgrading the Mineral Resources.

Twenty-five core holes were drilled by Seabridge in 2009 within the Mitchell zone totaling about 7,800 meters. The majority of these holes were drilled along the northern and southern flanks of the mineralized zone in order to: 1) test the north dipping extension of the system 2) upgrade Inferred material to Indicated and 3) test upper plate mineralization (material above the Mitchell thrust fault). 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 1,000 meters by 500 meters.

It is the opinion of the author that the data density within the Kerr, Sulphurets, and Mitchell 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 **Areas of Uncertainty**

As was mentioned in Section 17.12, the author is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the estimate of Mineral Resources. This project is still in the early pre-development stage with numerous studies that are ongoing with significant additional work that will be required regarding defining the limits of the deposit, identifying possible processing methods, and if warranted, permitting the project into a developing/producing property. Mining and processing costs, metal recovery, and permitting could have a material effect on the viability and potential size of this deposit. It is too early to determine the potential impact of these topics on the ultimate size or viability of this project.

19.4 **2009 Project Objectives**

It is the opinion of the author that Seabridge Gold successfully completed their stated 2009 project objectives by upgrading significant portions of Mitchell and Sulphurets Inferred Mineral Resources to either Measured (Mitchell only) or Indicated categories. Significant progress has also been made regarding metallurgy, preliminary mine planning exercises, along with a host of various permitting activities.

20.0 RECOMMENDATIONS

20.1 General Discussion

Seabridge Gold's 2006 through 2009 drilling programs have confirmed the presence of a large, disseminated, gold-copper system known as the Mitchell zone, with average grades in the order of 0.60 to 0.70 g/t Au and 0.2% Cu. The geology, dimensions and metal distribution of the Mitchell deposit are consistent with those of a gold-enriched, low-grade copper porphyry model. There are no apparent significant hard grade boundaries except the Mitchell thrust fault which separates more permissive lower plate units from upper plate lithologies with less extensive mineralization. The Mitchell deposit still remains open in the down-dip direction and to a lesser extent to the south and southeast. Pit optimization studies should be carried out to determine what type of economics may be required to chase the mineralized system to depth.

The Sulphurets zone, while smaller than the Mitchell zone, represents an attractive target due to its proximity to the Mitchell zone (and possible shared revenues), higher copper grades, and near surface exposures. Additional drilling will be required to close off the deposit and to upgrade the current Inferred Mineral Resources to higher confidence categories.

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 copper grades at Kerr provide an opportunity for blending and upgrading the copper concentrate that could be derived from the Mitchell and Sulphurets zones.

20.2 Specific Recommendations

The recommendations described below in should be carried out during the 2010 field season. These recommendations are not necessarily contingent on positive results from previous phases.

- The Sulphurets deposit remains open along strike from the "Canyon Zone" at the southwest end of the deposit northeasterly towards the main zone of mineralization. Drilling should target permissive geometry along strike or down-dip from existing gold intercepts. This program should be carefully designed with contingencies for dropping or adding holes based on the drilling results. The cost for such a program depends on the number of meters drilled but could easily exceed several million dollars.

- Additional drilling should be completed within the Main Copper Zone at Sulphurets to determine its limits and to upgrade currently defined Inferred Mineral Resources. The cost for such a program depends on the number of meters drilled but could range between $500,000 and $1,000,000.

- Seabridge drilled 7 holes into the Kerr deposit in 2009 and in general confirmed prior Placer Dome results. Additional drilling is recommended to 1) provide new samples for metallurgical testwork, 2) examine core recovery/geotechnical issues that were reported with the early 1990 drilling. This drilling should be conducted with triple tube core barrels and an adequate mud program to enhance core recovery. The "rubble breccia" zones (anhydrite/gypsum) that were identified by Placer Dome should be targeted and 3) drill test the down-dip extension of the deposit on the south end to confirm continuity along strike to the south. The cost for such a program depends on the number of meters drilled but could easily exceed several million dollars.

- If possible, test the continuity of mineralization between the Mitchell and Iron Cap deposits by drilling methods. Little is understood about the Iron Cap zone other than quartz-sericite-pyrite alteration is more intense than at Mitchell and there appears to be more base metal mineralization, particularly in narrow veins. The 2005 Falconbridge holes intersected low-grade gold mineralization near the surface. Offset holes from existing known mineralization should be designed to aid in determining the possible geometry of mineralization and its possible relationship to the nearby Mitchell deposit and potential influence on the location of the main haulage tunnels. The cost for such a program depends on the number of meters drilled.

- Continue with geotechnical studies for determining possible pit slope angles for the Mitchell deposit. Similar studies will be required for the Sulphurets and Kerr deposit. Seabridge has been working with a geotechnical consulting company to recommend pit slope angles and other infrastructure geotechnical design parameters. The author is unaware of the magnitude of costs associated with these activities.

- Continue ongoing metallurgical studies to determine potential recoveries and process flowsheets. Seabridge has contracted TJS Mining-Met Services Inc. to manage and direct a number of metallurgical consulting firms to complete these activities. The author is unaware of the costs associated with these activities.

- 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.

21.0 REFERENCES

Aero Geometrics, 2009, Adjustment of Local Grid to NAD27, 4 pages

Alldrick, D.J., and Britton, J.M., 1991, Sulphurets Area Geology, Openfile 1991-21 Geology Maps, British Columbia Geological Surveys Branch.

Brassard, J., 2009, Unpublished spreadsheets from The Claim Group containing pertinent KSM mining claim information,

AMEC (Martin, T. et al) 2004, Kerr-Sulphurets Project, Scoping Study – Tailings and Waste Rock Management, Proprietary Report Commissioned by Noranda Inc.

Ditson, G.M., 1997, Kerr/Sulphurets Resource Estimates, Internal Placer Dome Canada Memorandum dated January 23, 1997.

Ditson, G.M., Wells, R.C., and Bridge, D.J., 1995, Kerr: The Geology and Evolution Of A Deformed Porphyry Copper-Gold Deposit, Northwestern British Columbia, in Porphyry Deposits of the Northwestern Cordillera of North America, Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46, p. 509-523.

Fowler, B.P., and Wells, R.C., 1995, The Sulphurets Gold Zone, Northwestern British Columbia, in Porphyry Deposits of the Northwestern Cordillera of North America, Canadian Institute of Mining, Metallurgy and Petroleum, Special Volume 46, p. 484-498.

Huard, A. and Savell, M., 2005, Report On The 2004 Exploration Program Kerr-Sulphurets Property, Proprietary Report Commissioned by Falconbridge Ltd.

JKTech Pty Ltd, 2008, SMC Test Report on Five Samples from Seabridge Gold, report prepared for Seabridge Gold, 13 pages

Lechner, M.J., 2007, Mitchell Creek Technical Report, Northern British Columbia, NI 43-101 Technical Report prepared for Seabridge Gold, 104 pages

Lechner, M.J., 2008a, Kerr-Sulphurets Technical Report, Northern British Columbia, NI 43-101 Technical Report prepared for Seabridge Gold, 126 pages

Lechner, M.J., 2008b, Updated Mitchell Creek Technical Report, Northern British Columbia, NI 43-101 Technical Report prepared for Seabridge Gold, 104 pages

Lechner, M.J., 2009, Updated KSM Mineral Resources, NI 43-101 Technical Report prepared for Seabridge Gold, 154 pages

G & T Metallurgical Services, Ltd., 2007, Preliminary Assessment of Mitchell Zone Samples, Kerr-Sulphurets, British Columbia, Canada, report prepared for Seabridge Gold, 19 pages

Lewis, P.D. Toma, A., Tosdal, R.M. (compilers), 2001, Metallogenesis of the Iskut River Area, Northwestern British Columbia, in Mineral Deposit Research Unit Special Publication Number 1.

Lewis, P.D., Thompson, J.F.H., Nadaraju, G., Anderson, R.G., and Johannson, G.G., 1993, Lower and Middle Jurassic Stratigraphy in the Treaty Glacier Area and the Geological Setting of the Treaty Glacier Alteration System, Northwestern British Columbia, In Current Research, Part A: Geological Survey of Canada Paper 93-1A, p. 75-86.

Gray, M.D., and Munguia, J, 2006, Field Review Summary Report, Kerr-Sulphurets Project, Iskut River Region, British Columbia, Canada, Proprietary Report Commissioned by Seabridge Gold Inc.

P&E Mining Consultants Inc., 2009, Technical Report and Resource Estimate on the Snowfield Property, Skeena Mining Division, British Columbia, Canada, Latitude 56°29' N by Longitude 130° W, NI 43-101 Technical Report filed January 31, 2009, 87 pages

Savell, M., and Huard, A., 2005, Report On Diamond Drilling Mineral Claims 516241, 516242, 516245, 516248, 516251, 516252 and 516253 Skeena Mining Division NTS 104B08, 104B09 56.52N, 130.25W owned by Seabridge Gold Inc., Work performed by Falconbridge Limited July 7 – September 4, 2005, Proprietary Report Commissioned by Falconbridge Ltd.

Stantec Consulting Ltd., Environmental Evaluation of Kerr-Sulphurets Property, Northwestern B.C., 2003, Proprietary Report Commissioned by Noranda Inc.

Wardrop, 2009a, Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008, report written for Seabridge Gold and later filed as NI 43-101 Technical Report, 513 p.

Wardrop, 2009b, Kerr-Sulphurets-Mitchell (KSM) Preliminary Economic Assessment Addendum 2009, report written for Seabridge Gold and later filed as NI 43-101 Technical Report, 466 p.

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.

5. That I 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.

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, and Mitchell deposits. 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.

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 this 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 report was prepared for Seabridge Gold Corporation 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 Corporation. 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"

January 25, 2010

23.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES