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Associated Mining Consultants Ltd.



Preliminary Assessment of the Schaft Creek Deposit, British Columbia

Project Status Report No. 1

prepared for

955528 (Alberta) Ltd.
Calgary, Canada

submitted by

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955528 (Alberta) Ltd.
c/o McLeod & Company, LLP,
3rd Floor, 14505 Bannister Road S.E.
Calgary, Alberta,
CANADA, T2X 3J3

Attention: Guillermo Salazar S.

Dear Sirs:

Re: Project Status Report for the Preliminary Assessment (Scoping Study) of the Schaft Creek Mineral Deposit, British Columbia, Canada.

Associated Mining Consultants Ltd. (AMCL) and HATCH™ have been retained by 955528 (Alberta) Ltd. to conduct a preliminary assessment (Scoping Study) of the viability of the Schaft Creek project. AMCL is the Project Manager for this study.

As well, AMCL has provided an expert opinion on the valuation (subject of a separate report) of the Option Agreement to acquire Teck Cominco Limited's interest in the property. The Option Agreement forms the principal asset to be vended in to a TSX-Venture Exchange (the "Exchange") Capital Pool Corporation (CPC), Copper Fox Metals Inc. (the "Company").

This letter provides a project update report on the status of the preliminary assessment as well as a revised work program and budget. The Project Manager for the preliminary assessment is Mr. Keith McCandlish, P.Geol., Manager of Mineral Services of AMCL who will also act as the Company's independent Qualified Person.

PROJECT STATUS UPDATE

Introduction

The Schaft Creek copper (± gold, ± silver, ± molybdenum) porphyry deposit is located in the Liard Mining Division of northwestern British Columbia. The deposit is located 63 km north and west of Bob Quinn Camp on the Stewart-Cassier highway.

The preliminary assessment has been underway since April 2004, subsequent to the preparation of a NI 43-101 compliant technical report on the property. AMCL has relied on the information provided in the technical report as the starting point for planning and executing the preliminary assessment. This report and others used to support the listing of the Company are referenced as follows:

> *Summary Report, Status and Resource Estimate, Schaft Creek Property, Northwestern British Columbia. Prepared by Gary H. Giroux, M.A.Sc., P.Eng., Giroux Consultants Ltd. and Erik Ostensoe, P.Geo. and Dated June 30, 2003, Amended May 20, 2004.* (The "technical report").
>
> *Schaft Creek Property, Reclamation and Environmental Assessment-Camp & Exploration Drill Sites/Roads. Prepared by E.W. Beresford, P.Eng. and Dated July 31, 2003.* (The "supporting report").

AMCL has also been provided with all of the historic exploration and technical data generated by Asarco, Hecla Mining Company and TeckCominco Ltd.

Some of the data generated during the early stages of the preliminary assessment could be considered to be "material information" and have resulted in changes to the timing of certain expenditures in the proposed work program included in the technical report.

The Exchange has requested that a project status update be provided to provide information on the work completed since the date of the technical report.

In defining the scope of work for the preliminary assessment it become apparent that the development of an appropriate metallurgical process flow sheet would be key to the advancement of the project. In fact, this was recognized as being so key, that the inability to develop an appropriate recovery process could be considered to be a potential "fatal flaw" in the project. With this in mind, the decision was taken to proceed with an early metallurgical test program.

METALLURGICAL TEST WORK

It has been noted that the core from previous drilling programs, which is stored on site, exhibits a remarkable degree of preservation with limited visible weathering. The possibility of using the existing core for preliminary metallurgical testing was considered as a very cost-effective approach that could significantly advance the project. This would require a determination that the historical core inventory was unweathered to any great degree.

In April of 2004 a number of drill hole locations were selected to provide two composite metallurgical samples from a deeper "fresh" zone and a more shallow, potentially oxidized zone. The core boxes were shipped to the offices of Process Research Associates Ltd. (PRA) in Vancouver. PRA has been retained by the Company to act as the consultant metallurgical engineers to the project. A metallurgical specialist, Hoe Teh, P.Eng. from HATCH™ was requested to independently monitor the test procedures.

Selected samples were sent to Andres Skupinski for petrographic examination including macroscopic descriptions, determination of mineral content and alteration type, textural description and a determination of the degree of oxidation and weathering. It was noted that mineral alterations did not suggest the presence of any substantive surficial, meteoric weathering.

Only the composite sample from the deeper "fresh" zone has been tested to date. Testing of the oxidized sample is now underway.

The following summary discussion is sourced from the PRA report on the metallurgical test work entitled:

> *Metallurgical Validation Tests on Selected Drill Hole Core Samples, Schaft Creek Project, B.C. Prepared by Gie Tan, Ph.D., Senior Metallurgist, Process Research Associates Ltd. and Dated August 25, 2004.*

Process Research Associates (PRA) of Vancouver, B.C. has completed a program on selected core samples from the historical Schaft Creek drill core inventory. The inventory had been created by Hecla Mining Company during the 1970's and Teck Corporation during 1981. The primary objectives of the program were to assess the integrity of the samples and their suitability for metallurgical purposes and in turn validate the historic data base and the drill core inventory.

The samples for the program have been selected based on spatial distributions in the deposit, lithology and historic assays. Sixteen samples were selected for assay data base validation, from which five samples were selected for further metallurgical validation. The samples were assayed and tested by flotation using industry standard procedures and reagents.

The current assays have validated the historic data base. The current copper (Cu) and gold (Au) assays compared favourably with the data base; but molybdenum (Mo) and silver (Ag) assays were lower than historic. The favourable assay correlation of Cu, which is the main mineral, suggests that the historic Cu assays may be used as a basis for development considerations. The reasons for the differences in Mo and Ag assays are not evident.

The selected samples responded well to flotation. The metallurgy observed in the PRA work compares well with that obtained by Lakefield Research in 1981, based on a similar flowsheet and mineralization with similar Cu head grade.

PRA achieved rougher flotation recoveries of 91% to 96% for Cu, 70% to 92% for Au, and 90% to 99% for Ag. Mo recovery was lower at 38% to 85% and variable with head grade. A cleaner concentrate grading 30% Cu at a recovery of 72.2% was achieved. The cleaner flotation was not optimized nor locked cycle tests conducted.

The mineralogy is unlikely to pose acid generation concerns based on the analysis of the 16 selected core samples. Acid-Base accounting results indicated an excess neutralization potential of over twice the estimated acid potential in all cases and the paste pH ranged from neutral to alkaline. With the low head sulphide content in the samples to start and high flotation recoveries, the total sulphur in the tailings was reduced to below 0.03% to further reduce concerns on environmental impact.

Metallurgical Testwork Details

The historic samples were provided by Mr. Guillermo Salazar in 165 core boxes, labelled according to drill hole number in 10-foot interval splits. Following a visual assessment of the integrity of the samples, 16 samples were selected for assay validation based on prior documentation of assays, lithology and spatial distributions. Subsequently, 5 samples were selected from the suite of 16 for metallurgical validation using standard batch grinding and rougher flotation procedures. Current standard reagents for flotation of sulphides were used.

Initial scoping grinding and flotation were conducted to standardize the grind and flotation parameters. Standard open-cycle rougher flotation tests were then made at a typical grind of 80% passing 75 µ. One cleaner flotation was conducted to further assess the metallurgical condition.

The 5 samples selected for metallurgical validation represented drill holes H61, T182, T186, T172 and T176.

The 16 samples selected were also subjected to standard Acid-Base accounting procedures to assess any acid generation and environmental impact concerns.

Assay Validation Results

The initial visual assessment was that the samples were dry, competent rock which have been well preserved.

A comparison between the assays obtained by PRA and historical data base has been tabulated in Table 1.

Table 1: Sample Assay Comparison

Hole Number Starting Level, ft	Current PRA Assays				Historical Data Base			
	% Cu	% Mo	g/t Au	g/t Ag	% Cu	% Mo	g/t Au	g/t Ag
H44 - 750	0.24	0.048	0.10	0.4	0.29	0.143	0.09	<0.3
H61 - 770	0.46	0.009	0.18	0.7	0.42	0.014	0.22	2.05
H61 - 780	0.10	0.007	0.08	<0.3	0.12	0.009	0.07	1.03
T140 - 860	0.31	0.004	0.43	2.1	0.26	0.008	0.26	2.74
T182 - 770	0.46	0.051	0.23	1.6	0.49	0.123	0.35	2.88
T182 - 800	0.15	0.010	0.08	0.3	0.17	0.018	0.08	1.85
T183 - 540	0.46	0.064	0.41	3.8	0.45	0.103	0.35	4.25
T186 - 650	0.36	0.017	0.30	1.7	0.35	0.025	0.32	2.43
T186 - 660	0.39	0.015	0.33	2.2	0.33	0.026	0.27	2.54
T188 - 300	0.40	0.014	0.41	1.8	0.41	0.011	0.34	2.57
T188 - 320	0.26	0.005	0.33	0.5	0.26	0.008	0.35	1.44
T172 - 870	0.31	0.004	0.15	1.8	0.33	0.010	0.13	3.53
T172 - 880	0.42	0.023	0.13	2.5	0.46	0.045	0.15	4.01
T176 - 530	0.88	0.004	0.61	1.6	0.91	0.006	0.60	2.78
T176 - 490	0.24	0.004	0.16	0.3	0.28	0.009	0.21	1.17
T125 - 350	0.35	0.003	0.10	1.2	0.37	0.006	0.11	2.33

The current assays corresponded reasonably well with the historic data base for Cu and Au, while the current assays for Mo and Ag were consistently lower. The reasons for this are not evident, but the consistency suggests that it may be due to systematic differences in analytical techniques. Furthermore, the PRA assays were confirmed by an independent laboratory, Acme Analytical Laboratories Ltd., on the same samples.

The gold and silver analyses were done with a standard fire-assay pre-concentration and instrumental determination by atomic absorption spectroscopy (AAS). The special metals of interest, principally copper and molybdenum, were determined by an inductively coupled plasma-mass spectroscope (ICP-MS) or AAS methods after a multi-acid digestion. Other species were determined by ICP-MS. Sulfide sulphur was determined by wet chemical gravimetric methods and total sulphur using a LECO furnace.

The acceptable correspondence of Cu, the main element, suggests that its historical data would be reliable and may be used for development considerations.

Metallurgical Validation Results

Variable hardness has been observed based on the grinding times required to achieve the same target grind. The quantitative measurement of hardness, based on Bond Work Index, was not an objective in this exploratory validation program.

The standard batch rougher flotation tests produced high recoveries of Cu, Au and Ag at acceptable kinetics using typical reagent regimes for sulphide flotation. These observations and the ability to upgrade the concentrate to 30% Cu grade in one cleaner flotation test suggest that the historic drill core inventory is suitable for exploratory process development purposes.

As shown in Table 2, the rougher recoveries were 92% to 96% for Cu, 70% to 92% for Au, and 90% to 99% for Ag. Mo recovery was lower and more variable between 38% and 85%, depending on head grade.

Table 2: Batch Rougher Flotation at P_{80} of 75 µ

Hole Number	Rougher Concentrate Grades				Rougher Recoveries				
	% Cu	% Mo	g/t Au	g/t Ag	Mass, %	Cu, %	Mo, %	Au, %	Ag, %
T186	1.57	0.070	1.05	8.4	23.2	93.5	68.0	88.8	89.5
T182	1.34	0.167	0.68	4.3	25.1	93.5	84.9	88.3	99.3
T176	2.85	0.024	1.80	5.1	20.5	95.8	37.8	92.1	99.2
T172	1.93	0.066	0.65	11.4	19.5	95.1	61.6	88.8	93.3
H61	1.10	0.027	0.36	1.8	28.2	92.3	51.3	70.3	98.6

One batch open-circuit cleaner flotation conducted further confirmed the metallurgical integrity of the samples. The three-stage cleaning with re-grinding results for sample T186 have been tabulated in Table 3.

Table 3: Batch Cleaner Flotation of Sample T186

Product	Product Grades				Product Recoveries				
	% Cu	% Mo	g/t Au	g/t Ag	Mass, %	Cu, %	Mo, %	Au, %	Ag, %
Ro + Sc Conc	2.39	0.112	1.7	15.9	15.7	91.7	76.8	79.4	59.6
1st Clnr Conc	8.14	0.361	6.1	48.5	4.3	84.9	67.2	73.9	49.4
2nd Clnr Conc	24.8	0.995	18.3	136.6	1.3	76.7	54.9	66.4	41.3
3rd Clnr Conc	30.1	1.150	22.6	158.6	1.0	72.2	49.2	63.5	37.1

The final concentrate grade of 30% Cu is reasonable for a chalcopyrite type ore.

All the flotation tests were not optimized. Optimization would generally improve recovery and grade.

Acid-Base Accounting

Acid-Base accounting tests using US-EPA standard procedures were conducted on the 16 ore samples to determine the potential for acid-generation and environmental impact due to the presence of sulphur.

The results showed that the samples from the selected drillholes would not pose acid generation problems. This may not apply to the deposit as a whole as other areas have a greater amount of pyrite mineralization and further study is required.

Sulphur contents in the samples were low, given the low levels of contained metals. As a result, the acid generation potential was low and the neutralization potential was high. The average NP:AP ratios exceeded 6 to indicate that the potential for acid generation is remote. This was also confirmed by the neutral to alkaline paste pH determinations. These results have been tabulated in Table 4.

Table 4: Acid-Base Accounting: Results of 16 Samples

	Sulphide Sulphur %	Paste pH	AP	NP	NP/AP	(NP-AP)
Averages	0.43	8.8	13.4	75.5	7.36	62.2
Range	0.1 - 0.9	7.5 - 9.3	3.4 - 28.6	53 - 114	3.0 - 16.9	45 - 91

With the high flotation recoveries of the metals, the total sulphur content in the tailings was reduced to very low levels at below 0.03% in all cases. Coupled with the fact that flotation would be run under alkaline conditions, it is safe to assume that the potential for acid generation from flotation tailings would be even more remote.

Conclusions

The assay data base for the deposit, particularly Cu and Au assays, may be used for project development purposes.

The high ratio of sulphide sulphur to total sulphur assay in the samples and the favorable flotation responses confirmed the integrity of the drill core inventory and that it may be used for exploratory metallurgical and process development work.

PIT OPTIMIZATION STUDIES

AMCL has completed a series of pit optimization runs to identify areas at Schaft Creek that may provide higher initial operating profits under a scenario where high volume surface mining is employed. These potential “starter pits” will be used to focus areas where additional metallurgical test work based on the existing drill core should be completed. The pit optimization studies were completed by Peter Lacroix, P.Eng., Chief Mining Consultant, AMCL.

This optimization study is preliminary in nature and includes inferred mineral resources that may be considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary assessment will be realized and the reader is cautioned not to infer that the project has advanced to the stage that a mineral reserve can be estimated. Mineral resources which are not mineral reserves do not have demonstrated economic viability.

Mineral Resource Estimates

The most recent mineral resource estimate was prepared by Giroux Consultants Ltd. (Giroux) of Vancouver in June 2003. The resource model is based on the original 3D block model constructed by Teck (now TeckCominco) in the 1980's. Other than addressing some inconsistencies in the some areas of the model, Giroux retained Teck's assignment of block codes for geology.

Grades were interpolated into individual 100 ft. x 100 ft. by 40 ft. high blocks utilizing ordinary Kriging of drill hole assay data composited into 20 ft +/- 10 ft. intervals. A minimum of 4 and a maximum of 16 composites were used for each block estimate. No limit was set for the maximum number of composites per drill hole. Three interpolation passes were employed with successively larger 3D search ellipse dimensions equal to one-third, two-thirds and the full range of the variogram models produced for each geologic domain. The interpolation pass in which each block received a grade estimate was tracked by assigning additional blocks codes of 1, 2 and 3 respectively.

Individual resource blocks were classified according to the CIM guidelines as measured, indicated or inferred based on whether the grade was assigned during pass 1, 2 or 3 provided a maximum estimation error was not exceeded for each category.

In AMCL's opinion, the method employed by Giroux does not truly address the density of data in a given area, assigning a portion of the model surrounding each drill hole as measured and indicated, regardless of drill spacing. A more appropriate approach would be to classify resources in respective areas based on whether or not they meet a minimum threshold in data density for each of the categories. In this case a maximum drill spacing commensurate with a distance on the variogram model where the variance does not exceed two thirds the total variance (less the nugget effect) would be appropriate for measured, provided this is supported by a detailed bulk sampling program confirming grade estimates. None of the resource would be classified in the measured resource category based on this approach.

Table 5 provides a summary of the Giroux estimate stated at a cutoff grade of copper equivalent value of 0.35%. It should be noted that the determination of equivalent grade by Giroux is based on metal prices and assumed recoveries only. Smelter terms and other off-property costs such as freight essential in determining true equivalent value have not been addressed in the Giroux calculations. Inclusion of these factors would result in equivalent copper values approximately 30% higher, or conversely, grades 30% lower at any given copper equivalent cutoff.

Table 5: Mineral Resource Estimate, Schaft Creek: 0.35% Copper Equivalent Cutoff (Giroux and Ostensoe, June 2003)

Category	**Tonnes x 1,000**	**Cu (%)**	**MoS2 (%)**	**Au (g/t)**	**Ag (g/t)**	**CuEq (%)**
Measured	91,800	0.370	0.037	0.26	1.89	0.495
Indicated	373,000	0.356	0.041	0.24	2.02	0.481
Measured plus Indicated	464,700	0.359	0.040	0.25	1.99	0.484
Inferred	169,300	0.358	0.045	0.26	2.19	0.481

Note: Tonnage calculations are based on a specific gravity of 2.67 (12 cubic feet per ton).

Economic Model

In order to facilitate the economic calculations and subsequent pit optimization runs, the block model was imported in ASCII format to Minesight® and equivalent values re-estimated using appropriate prices, metallurgical recoveries, smelter terms and transportation costs. Equivalent copper values have been calculated by estimating the net smelter return for gold and silver and back calculating what the copper grade would have to be for the revenue to be generated from copper alone. Smelter terms have been derived from long-term industry average prices known to AMCL.

Molybdenum prices have spiked upwards, sharply, since mid 2002 but we have used a more conservative average over 1998-2001.

In simple terms, the calculation is as follows:

Cu Eq %= Cu% + (NSR Au US$ + NSR Ag US$+ NSR Mo)/(NSR Cu US$ per %Cu)

Assumed process performance, prices and smelter terms used in these calculations are summarized in Table 6. Equivalent copper grades net of process recovery and smelter deductions have also been calculated to facilitate subsequent optimization runs.

Table 6: Net Smelter Returns

Metal Prices	
Cu US$/lb	0.90
Mo US$/lb	2.40
Au US$/oz	375
Ag US$/oz	5.50
Copper Concentrate	
Concentrate Grade Cu%	30
Cu Recovery %	(Cu%-0.065)/Cu%, Min 50, Max 95
Au Recovery %	(Au g/t-0.14)/Au g/t, Min 30, Max 80
Ag Recovery %	80
TC/RC US$/lb	0.192
Freight US $/lb	0.128
Cu Payable %%	96.7
Au Payable %	93
Ag Payable %	90
Au Refining US$/oz	5.50
Ag Refining US$/oz	0.40
MoS_2 Concentrate	
Concentrate Grade Mo%	50
Mo Recovery %	70
%Mo per %MoS_2	0.5994
TC (Roasting) US$/lb	0.50
Freight US $/lb	0.075
Mo Payable %	100

Table 7 and Table 8 provide summaries of the results stated at increasing cut-off values. Since the model format provided was in imperial units, the table reflects this standard. Note that while measured and indicated have been lumped into indicated commensurate with the above comments, no attempt has been made to redistribute resources between indicated and inferred. Future modelling efforts should address this.

Table 7: Re-stated Indicated Mineral Resource Estimate, Schaft Creek

Cutoff CuEq (%)	Tons X 1,000	Cu (%)	MoS_2 (%)	Au (opt)	Ag (opt)	CuEq (%)	Net Cu Eq (%)
0.20	1,344,148	0.248	0.026	0.0049	0.0487	0.414	0.292
0.25	1,149,954	0.268	0.029	0.0054	0.0504	0.446	0.322
0.30	933,207	0.292	0.032	0.0059	0.0524	0.485	0.359
0.35	739,013	0.316	0.035	0.0065	0.0547	0.528	0.399
0.40	567,137	0.341	0.037	0.0071	0.0576	0.574	0.442
0.45	431,058	0.364	0.040	0.0078	0.0608	0.622	0.485
0.50	325,503	0.384	0.042	0.0086	0.0649	0.670	0.528
0.60	183,686	0.426	0.047	0.0100	0.0723	0.767	0.617
0.70	101,918	0.472	0.054	0.0112	0.0794	0.866	0.708
0.80	57,881	0.514	0.061	0.0124	0.0849	0.956	0.793
0.90	31,226	0.565	0.068	0.0134	0.0886	1.052	0.883
1.00	16,914	0.610	0.076	0.0144	0.0946	1.141	0.969

Table 8: Re-stated Inferred Mineral Resource Estimate, Schaft Creek

Cutoff CuEq (%)	Tons X 1,000	Cu (%)	MoS2 (%)	Au (opt)	Ag (opt)	CuEq (%)	Net Cu Eq (%)
0.20	824,984	0.220	0.027	0.0032	0.0367	0.356	0.240
0.25	608,538	0.246	0.031	0.0039	0.0401	0.403	0.281
0.30	429,456	0.277	0.036	0.0046	0.0432	0.458	0.331
0.35	302,084	0.309	0.039	0.0053	0.0475	0.514	0.383
0.40	206,405	0.340	0.042	0.0063	0.0535	0.580	0.442
0.45	145,621	0.369	0.044	0.0074	0.0591	0.646	0.500
0.50	110,292	0.389	0.044	0.0084	0.0644	0.701	0.548
0.60	65,988	0.416	0.045	0.0107	0.0743	0.804	0.636
0.70	40,434	0.433	0.047	0.0131	0.0842	0.906	0.721
0.80	26,389	0.455	0.050	0.0148	0.0914	0.990	0.795
0.90	16,013	0.469	0.048	0.0170	0.0974	1.082	0.869
1.00	8,774	0.509	0.046	0.0193	0.1113	1.186	0.971

As discussed on page 5, current analytical work has resulted in consistently lower molybdenum and silver values than the historical database, albeit on a very limited sample population. The reason for this is not evident and will require further investigation. The re-stated resource estimates utilize the historic data base.

Pit Optimizations

A series of pit optimizations were completed to provide guidelines for future technical and economic studies as well as give targets for the next round of metallurgical test work. Results are summarized in Table 9 and Table 10 while plans and sections can be found in Appendix 1 and 2. The optimizations, which utilized the Lerchs-Grossmann algorithm, were based on net copper equivalent values calculated by estimating the gold, silver and molybdenum revenues for each block and back calculating what the equivalent payable or net copper grade would have been to generate the revenue entirely from copper. These values, along with grid matrices for topographic, bedrock and intact rock surfaces were incorporated in a condensed block model suitable for use with the optimization software. The use of a net equivalent value was necessary to facilitate the use of price variations in the revenue model without having to recalculate the condensed model for each variation in price or cost. Only whole blocks were used in the condensed model.

A total of 11 nested pit shells were generated utilizing copper prices from US$0.70 to US$1.20 per pound copper in steps of US$0.05 per pound. The price steps allowed the definition of smaller, lower-strip pits with higher unit values and grades within the ultimate pit outline that could be used to identify some of the near-surface higher-grade, higher value material that could be mined earlier in the mine life for accelerated payback.

Table 9: Optimization Results: Incremental

Pit	Price US$/lb CuEq	>0.22% Net EqCu		Waste/OB ktons	Strip Ratio
		ktons	Grade Cu Eq%		
1	0.70	2,467	0.966	1,133	0.46
2	0.75	4,666	0.720	3,700	0.79
3	0.80	4,934	0.663	3,534	0.72
4	0.85	5,933	0.586	4,133	0.70
5	0.90	17,200	0.528	10,233	0.59
6	0.95	27,733	0.486	18,167	0.66
7	1.00	64,934	0.431	28,033	0.43
8	1.05	52,533	0.407	25,334	0.48
9	1.10	118,200	0.400	76,600	0.65
10	1.15	90,567	0.372	63,066	0.70
11	1.20	59,766	0.395	72,267	1.21
	Total	448,933	0.421	306,200	0.68

Notes: Grades are net of recovery and deductions.
Operating Profit and NPV based on US$1.20/lb. EqCu
NPV assumes best case, mining shells in sequence at 20,000 ktons ore/annum.

Table 10: Optimization Results, Cumulative

Pit	Price US$/lb CuEq	>0.22% Net EqCu		Waste/OB ktons	Strip Ratio
		ktons	Grade Cu Eq%		
1	0.70	2,467	0.966	1,133	0.46
2	0.75	7,133	0.805	4,833	0.68
3	0.80	12,067	0.747	8,367	0.69
4	0.85	18,000	0.694	12,500	0.69
5	0.90	35,200	0.613	22,733	0.65
6	0.95	62,933	0.557	40,900	0.65
7	1.00	127,867	0.493	68,933	0.54
8	1.05	180,400	0.468	94,267	0.52
9	1.10	298,600	0.441	170,867	0.57
10	1.15	389,167	0.425	233,933	0.60
11	1.20	448,933	0.421	306,200	0.68

Notes: Grades are net of recovery and deductions.
Operating Profit and NPV based on US$1.20/lb. EqCu
NPV assumes best case, mining shells in sequence at 20,000 ktons ore/annum.

Actual input prices used were US$0.38 to US$0.88 per pound, which are the net prices after deduction of the treatment, refining and freight charges of US$0.32 per pound outlined in Table 2. Costs used in the optimizations were US$1.20 for waste mining and $5.25 for ore mining, process and G&A. Based on these costs, the economic mill cut off would be 0.22% net copper equivalent at US$1.20 per lb copper. Pit slopes used in the runs were 45° and 34° for rock and overburden respectively. No additional provision was made for ramps or access roads.

All values are US Dollars unless otherwise stated. The reported inventories are those resources contained within the pit boundaries above the stated cut-off values. No differentiation is made between indicated and inferred resources. No dilution has been added nor should the pits be considered mineable as logistics and access have not been considered.

RECOMMENDATIONS

The geological /resource model should be refined and more appropriate resource classification categories developed. A surveyed digital terrain model should be created. These will be used as the basis for reserve modelling.

Further analytical work is required to determine the reason for the limited, current Mo and Ag values being lower than those reported in the historic database.

Metallurgical Testing

A number of sample intervals have been selected by AMCL for further metallurgical testing which are believed to influence resource estimates and optimization results materially. Table 11 provides a selection of sample intervals that are located within the bounds of the pit optimized at US$0.90 per pound copper equivalent.

Table 11: US$0.90 Pit, Selected Sample Intervals

Area	Approximate Northing	Drill Hole ID	Sample Interval From-To (ft.)
Liard	23200	H71CH076	150-430
Liard	23400	H69CH047	220-440
West Breccia	23500	H71CH089	130-230
Liard/W.B.	23700	H71CH083	30-200
Liard	23700	H71CH072	110-390
Liard	23700	H71CH084	18-180
Liard	24000	DDHAS-12	20-160
Liard	24200	T81CH171	20-260
Liard	24500	T81CH150	40-220
Liard	24500	T81CH098	80-330
Liard	24700	T81CH179	40-250
Liard	25000	DDHAS-23	50-280
Paramount	28000	T81CH204	20-200

Table 12 (following) provides a list of intervals outside the bounds of the US$0.90 pit but within the limits of the US$1.20 pit.

The intervals are sufficiently large to allow selection of a broad range of sample grades and/or geologic compositions, depending on the program design. Alternatively, cross sections in Appendices 2 and 3 can be used in conjunction with the plan views in Appendix 1 to determine which holes and sample intervals have the greatest influence on the optimum pit boundaries. Holes with sample intervals located close to the bottoms of each pit would be good candidates for test programs as they likely played a significant role in determining the maximum depth at each price point.

Table 12: US$1.20 Pit, Selected Sample Intervals

Area	Approximate Northing	Drill Hole ID	Sample Interval From-To (ft.)
Liard	22500	H71CH085	70-480
Liard	23200	H71CH076	500-700
Liard	23400	H69CH047	440-730
Liard	23400	H69CH045	880-1480
West Breccia	23500	H71CH089	230-550
West Breccia	23800	H72CH091	240-520
Liard	23800	H71CH083	200-560
West Breccia	23900	DDHAS-18	120-380
Liard	23900	H68CH032	980-1510
Liard	24000	T80CH133	360-810
Liard/W.B.	24000	T80CH135	130-310
Liard	24200	T81CH169	160-900
Liard	24300	H69CH049	500-1000
Liard	24500	H70CH059	570-1100
Liard	24800	T81CH183	430-990
Liard	25000	DDHAS-20	230-500
Liard	25500	T81CH194	70-492
Paramount	28000	T81CH203	280-400
Paramount	28500	T80CH132	410-880
Paramount	29000	T80CH124	430-610

Composite samples should be selected from a range of lithologies comprising the deeper sections of the holes as these represent the grades that drive the development of the pit bottoms. Testing should include sufficient work to determine whether an acceptable Mo concentrate can be produced as molydenum reporting to a copper concentrate is not generally saleable and in some case may be considerd to be a penalty element.

Revised Work Program

The following modified work program (Table 13) has been proposed. Phase 1 reflects the results of the current and on-going studies. Phase 2 is a proposed drilling program. The budget forecast relates to project specific expenses only and does not include Copper Fox corporate costs for working capital, general and administrative cost or unallocated costs.

Table 13: Proposed Work Program

Phase 1		
Sample Collection		C$65,000
Assays	3 sample sets @ C$3,000	C$9,000
Metallurgical Test Work	6 composites @ C$17,000	C$102,000
Project Engineering (Scoping Study)		C$50,000
Environmental Bonding		C$150,000
Sub-total		C$376,500
Project Administration (15% of sub-total)		C$112,875
Contingency (10% of sub-total)		C$37,650
Total Phase 1		C$527,025
Phase 2		
Metallurgical Drilling	5,000 m PQ @ C$125/m	C$625,000
Exploration Drilling	1,500 m HQ @ C$105/m	C$157,500
Assays	4 samples @ C$3,000	C$12,000
Petrographic Work		C$40,000
Metallurgical Test Work	8 composites @ C$17,000	C$136,000
Project Engineering		C$50,000
DTM Surveying		C$50,000
Sub-total		C$1,070,500
Project Administration (15% of sub-total)		C$160,575
Contingency (10% of sub-total)		C$107,050
Total Phase 2		C$1,338,125

CONCLUSIONS

Preliminary metalurgical test work completed as part of the on-going preliminary assessment has been very encouraging.

A scoping level metallurgical program involving batch tests should be conducted using samples from the existing inventory to develop a preliminary flowsheet and process design. The test program would include determinations of grindability and flotation recovery of Cu, Mo, Au and Ag using the parameters adopted in the validation work. Some optimization should be carried out to maximize the recovery of copper, silver and gold to a final copper concentrate at a target grade of 30% Cu. Work should also be completed on producing a separate and saleable MoS_2 concentrate

In addition, a new drilling program should be conducted to produce fresh core samples for testing and confirmation of the flowsheet and process design. The fresh core samples should be selected to represent any variation in mineralization and bulk of the deposit. This will form the basis for a scoping level estimate of capital and operating costs for the development of the deposit and production of a marketable concentrate.

If you have any questions or concerns, please do not hesitate to contact the undersigned at any time.

Yours Sincerely,

ASSOCIATED MINING CONSULTANTS LTD.

Keith M^c^Candlish, P.Geol.,
Manager of Mineral Services

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September 20, 2004

CERTIFICATE OF QUALIFICATIONS

I, Keith M^{c}Candlish, P.Geol.;

1. Am currently employed by: Associated Mining Consultants Ltd. (AMCL)
Suite 415, 708-11th Avenue S.W.,
Calgary, Alberta, CANADA, T2R 0E4

in the capacity of: Manager of Mineral Services

2. Am a Professional Geologist (P.Geol.), member number #45717, registered with the Association of Professional Engineers, Geologists and Geophysicists of Alberta (APEGGA). A summary of my relevant experience follows;

Twenty-five years of consulting geological and engineering experience in minerals, oil sands/heavy oil, precious stones, coal and industrial minerals. In 1988 he joined Associated Mining Consultants Ltd. where he is now Manager of Mineral Services focussing on corporate finance, due diligence, technical audits, and, mining fraud investigation.

Mr. M^{c}Candlish has been actively involved on due diligence evaluations of mining projects covering a range of mineral commodities and has had extensive experience in exploration property valuations, analysis of project economics, exploration logistics, assaying and project management. Detailed due diligence evaluations have been conducted for a number of mining operations, internationally, including:

1. Greenstone Resources Las Libertad and Cerro Mahon open-pit gold projects, Honduras, Nicaragua
2. Sunshine Mining's Pirquitas silver-tin-zinc proposed development, Argentina
3. Navan Resources Chelopech copper-gold and Almagrera copper-zinc underground mines, Bulgaria, Spain
4. Gravelotte Mines Limited, South Africa
5. Nelson Gold's Jilau open-pit operation, Tajikistan
6. Avocet Resources Penjom open-pit gold mine, Malaysia
7. Sutton Resources Bulyanhulu gold mine, Tanzania
8. Randgold Resources Syama mine, Mali
9. Golden Star Resources Bogoso and Prestea operations, Ghana

Mr. M^{c}Candlish has specific experience in the evaluation of copper (± gold) porphyry type deposits having conducted exploration on and/or evaluations of operating mines in porphyry deposits in Bulgaria (Chelopech), Spain (Almagrera), Mexico (Bahuerachi) and northwestern British Columbia (Galore Creek, Kemess, Schaft Creek and Red Bird).

3: Am a "Qualified Person" for the purposes of National Instrument 43-101.

4. Am currently acting as the Project Manager for a multi-disciplinary joint-venture team between AMCL and HATCH™ completing a preliminary assessment of the Schaft Creek Deposit. I have not visited the site, however, Mr. Peter Lacroix, P.Eng., Chief Mining Consultant, an AMCL employee and participant in the study team visited the site on April 16th and 17th, 2004.

5. Have been involved in all aspects of the preparation of this update report and take full responsibility for its' conclusions.

6. Am not aware of any material fact or material change with respect to the subject matter of the technical report which is not reflected in the update report, the omission to disclose which makes the technical report misleading.

7. The preliminary assessment referred to in this report uses as its starting point, a NI 43-101 compliant technical report entitled:

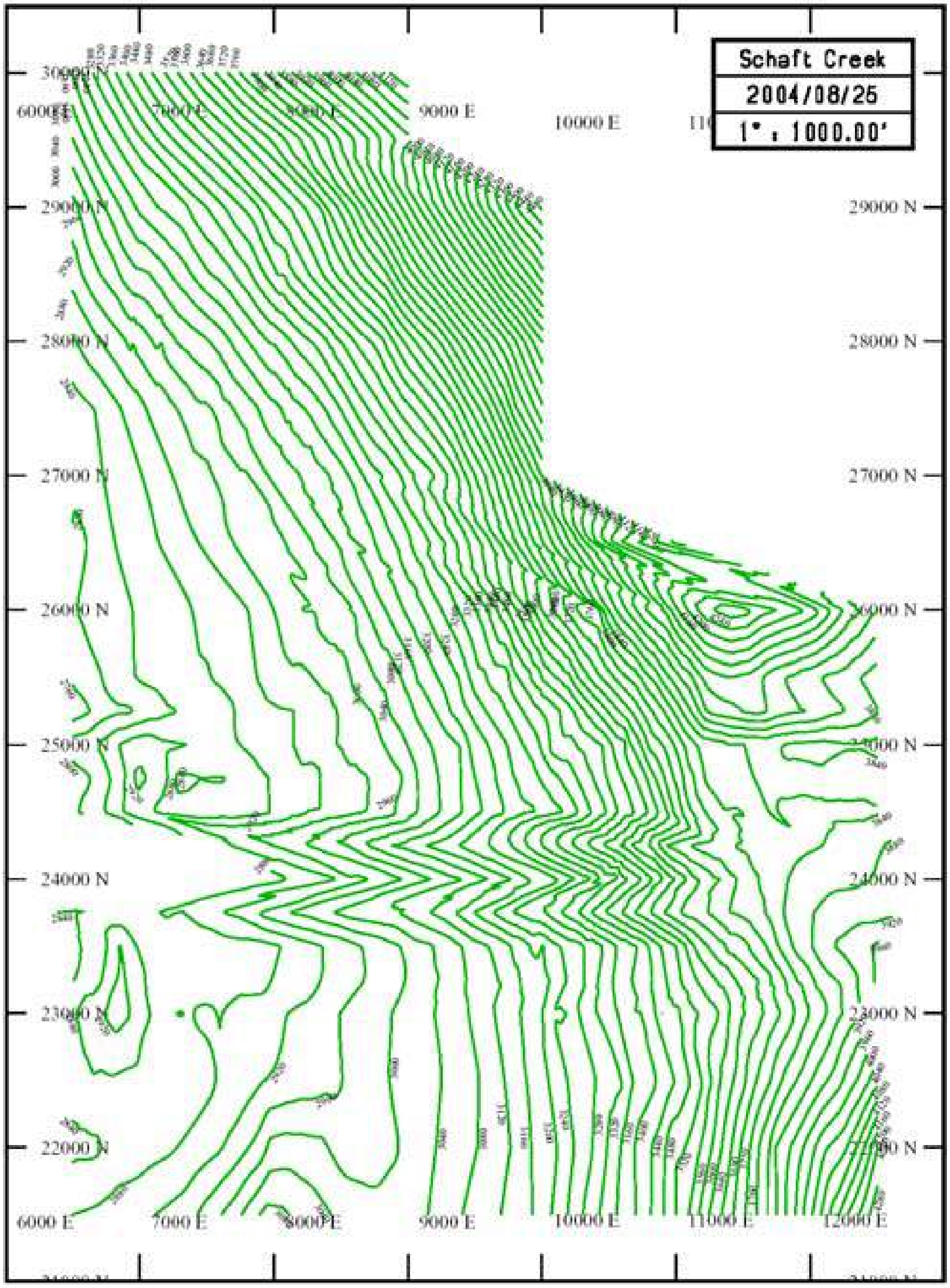
   Summary Report, Status and Resource Estimate, Schaft Creek Property, Northwestern British Columbia. Prepared by Gary Giroux, P.Eng., Giroux Consultants Ltd. and Erik Ostensoe, P.Geo. and Dated June 30, 2003.

8. Am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

Dated this 20th Day of September, 2004 at Calgary, Alberta, CANADA

Keith McCandlish, P.Geol.
Manager of Mineral Services
Associated Mining Consultants Ltd.

APPENDIX 1
PLAN VIEWS



Initial Topography from Overburden Solid

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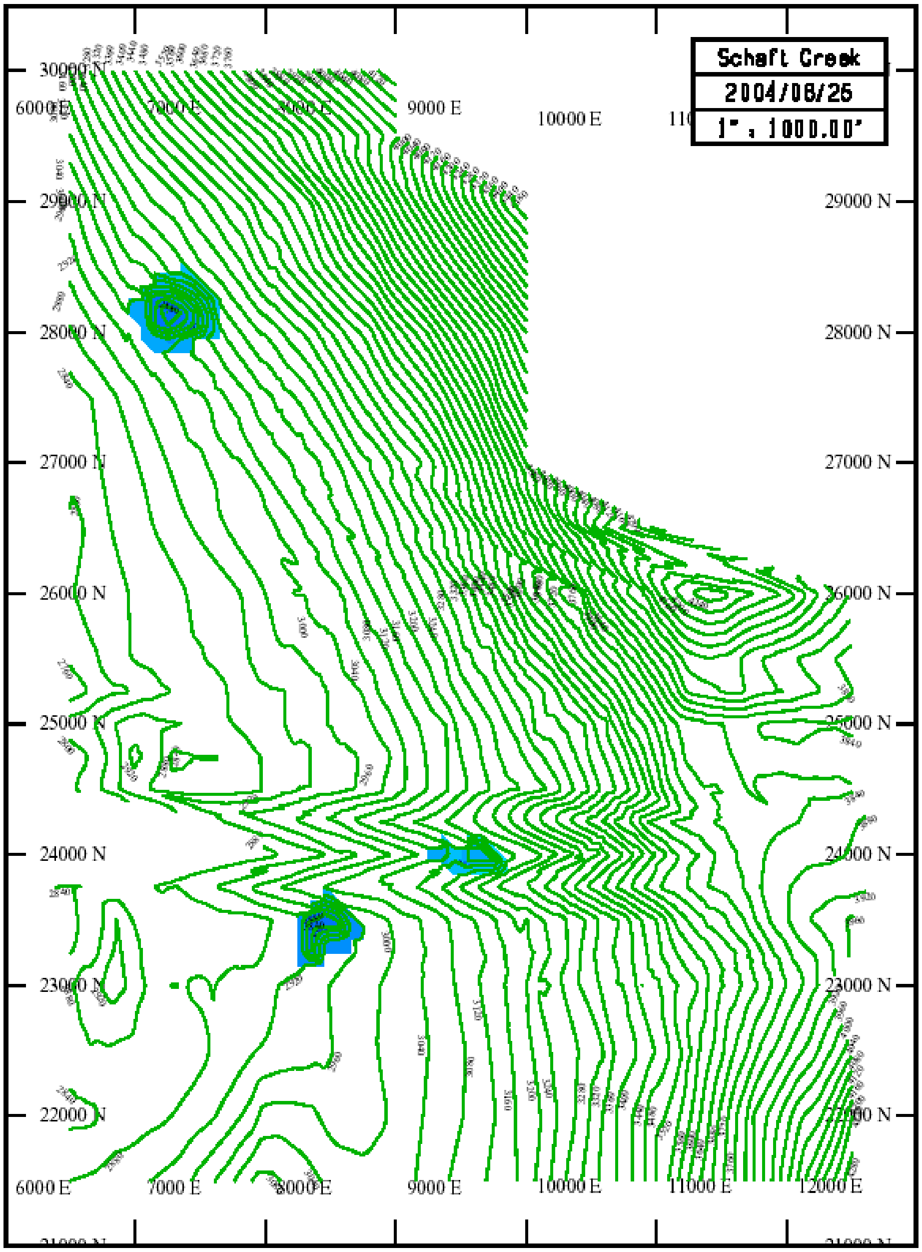
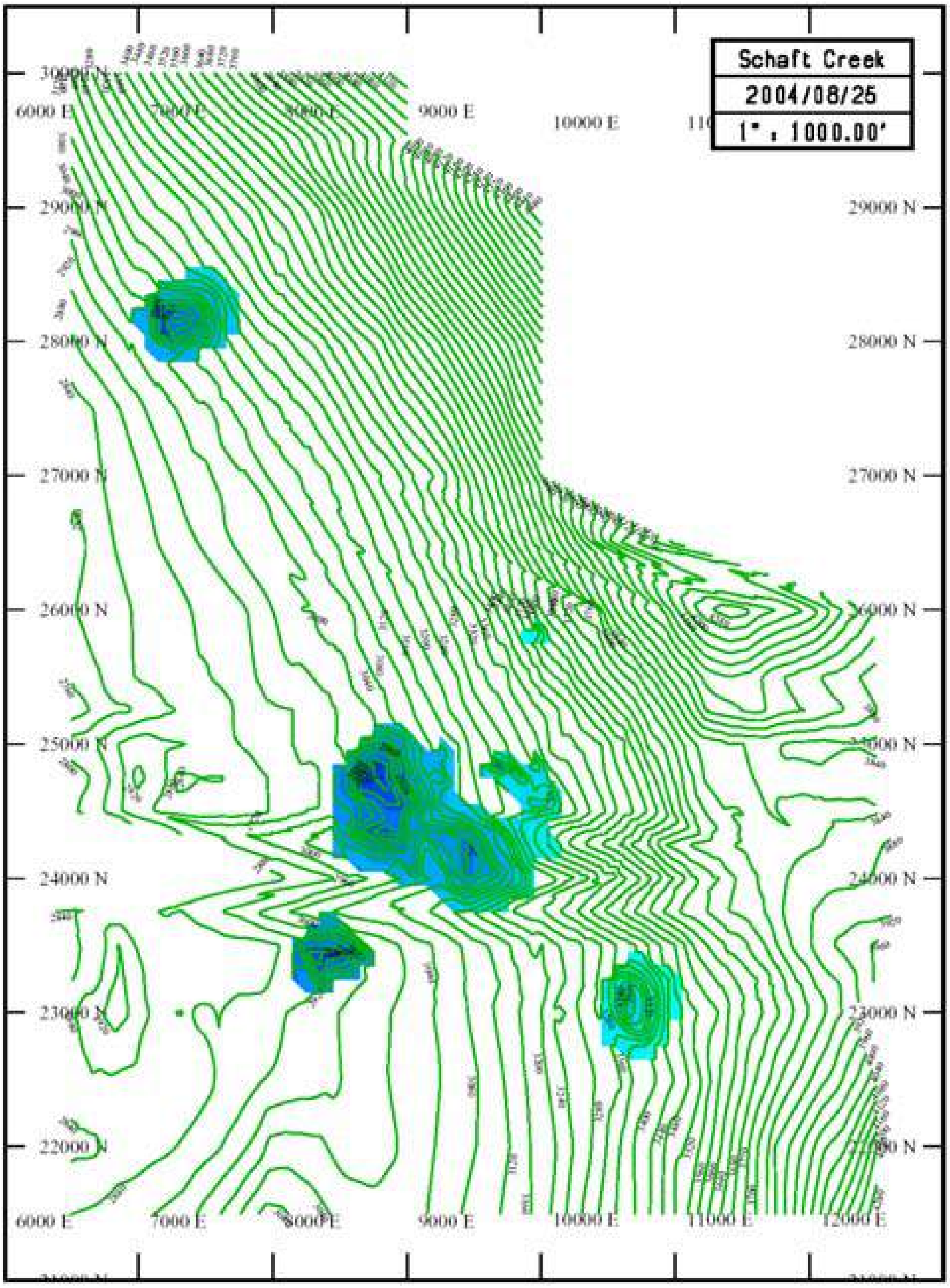
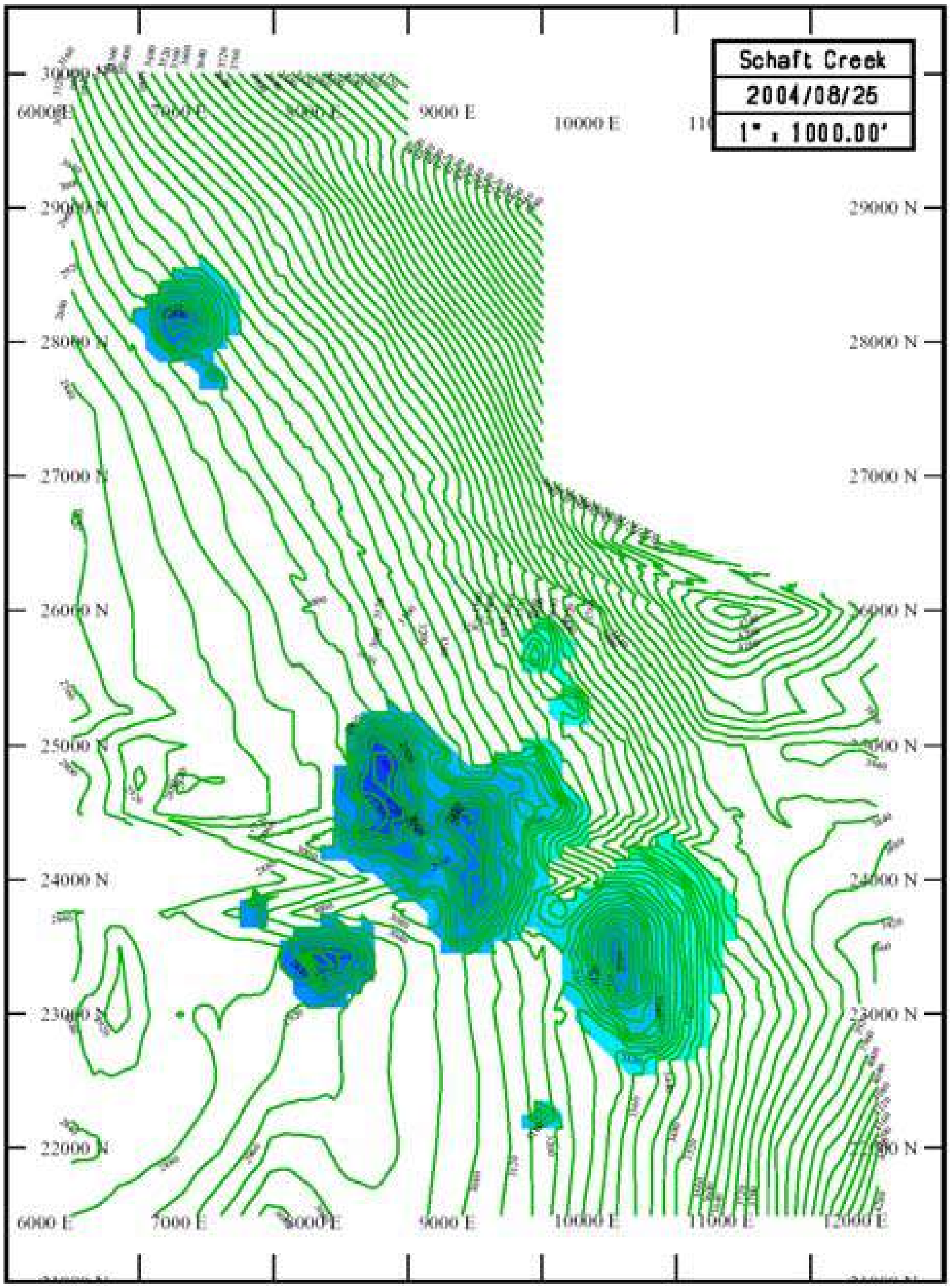


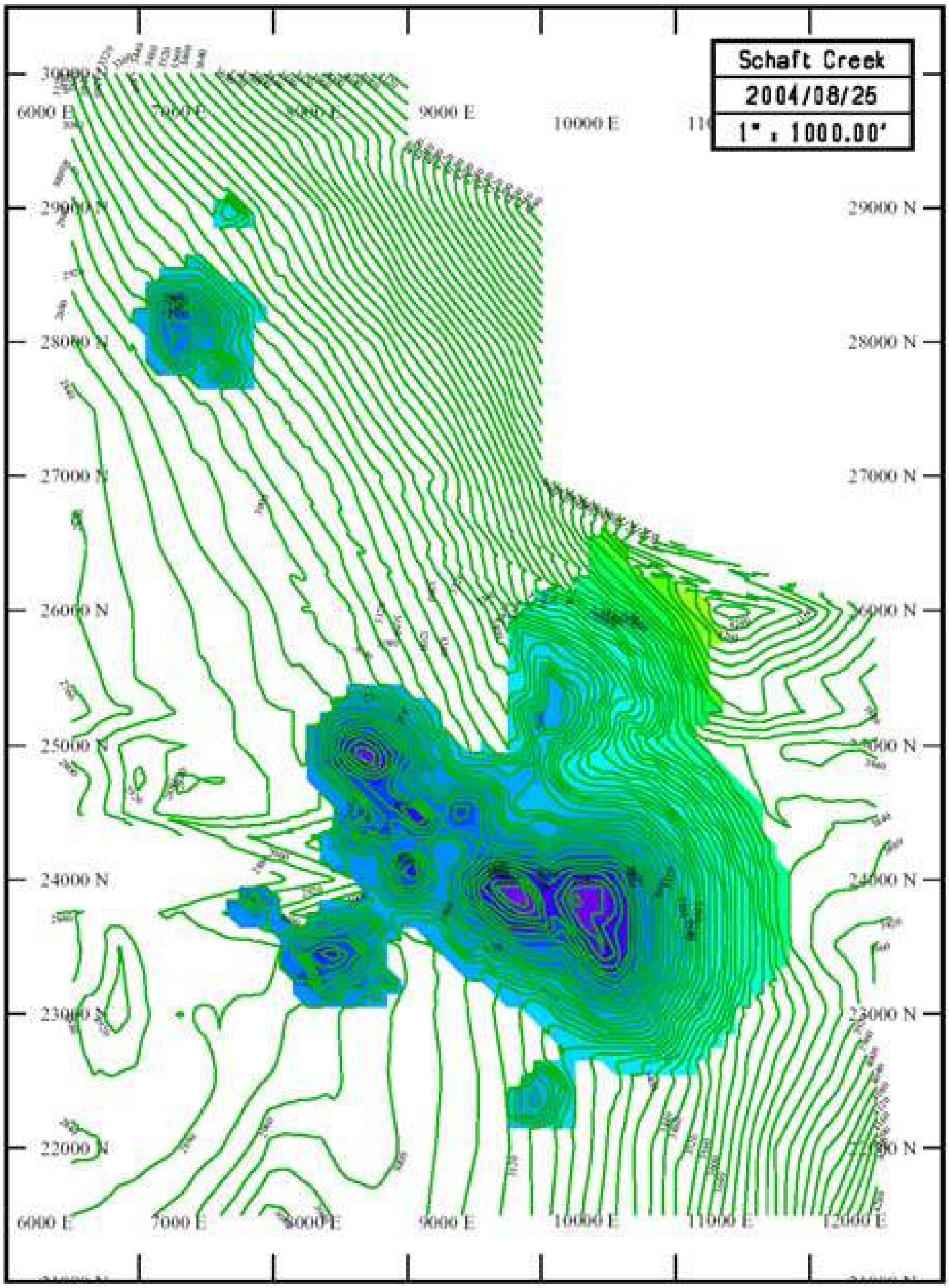
US$0.70/lb Cu Equivalent Optimized Pit Contours (Shaded)



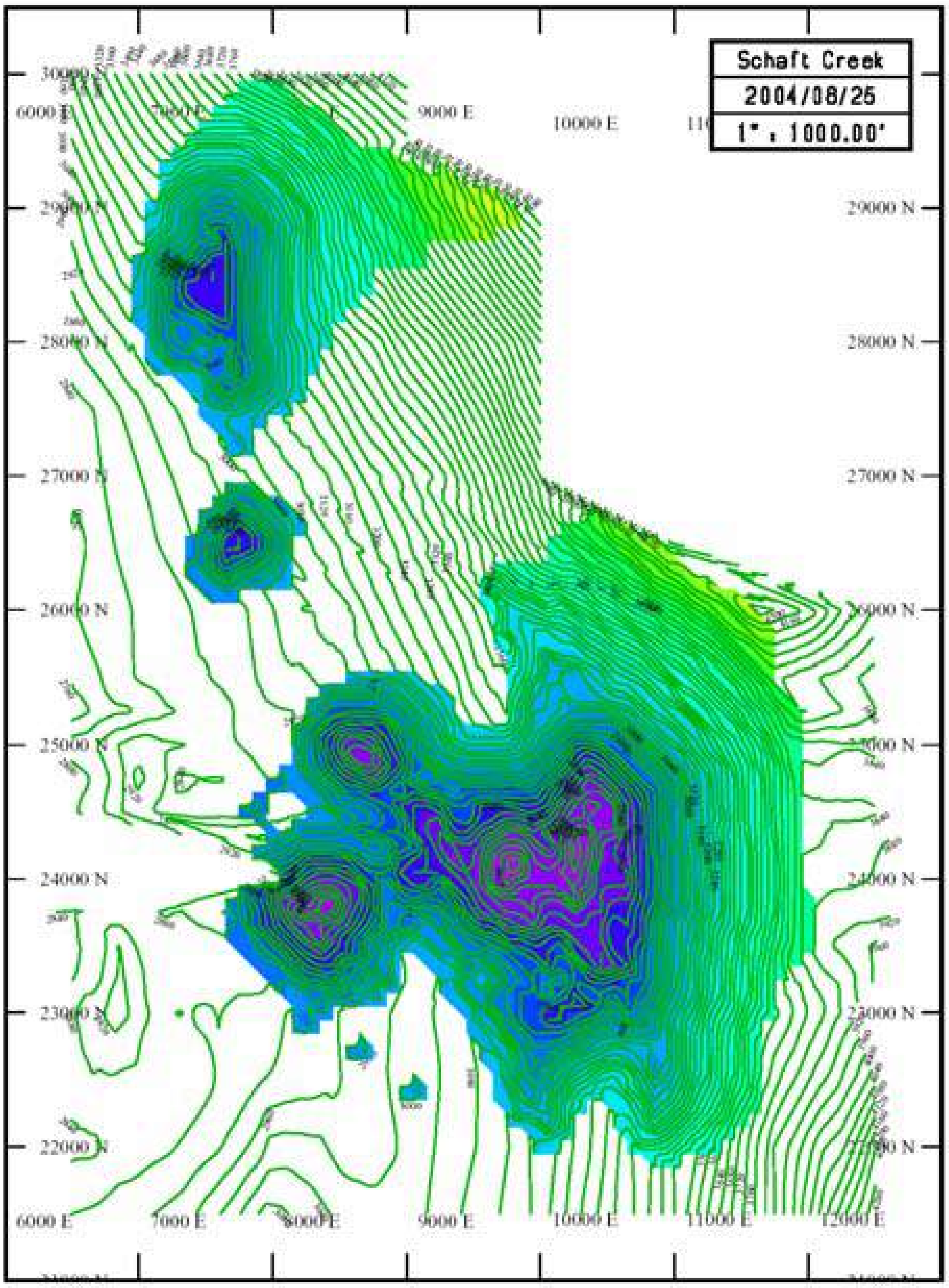
US$0.80/lb Cu Equivalent Optimized Pit Contours (Shaded)



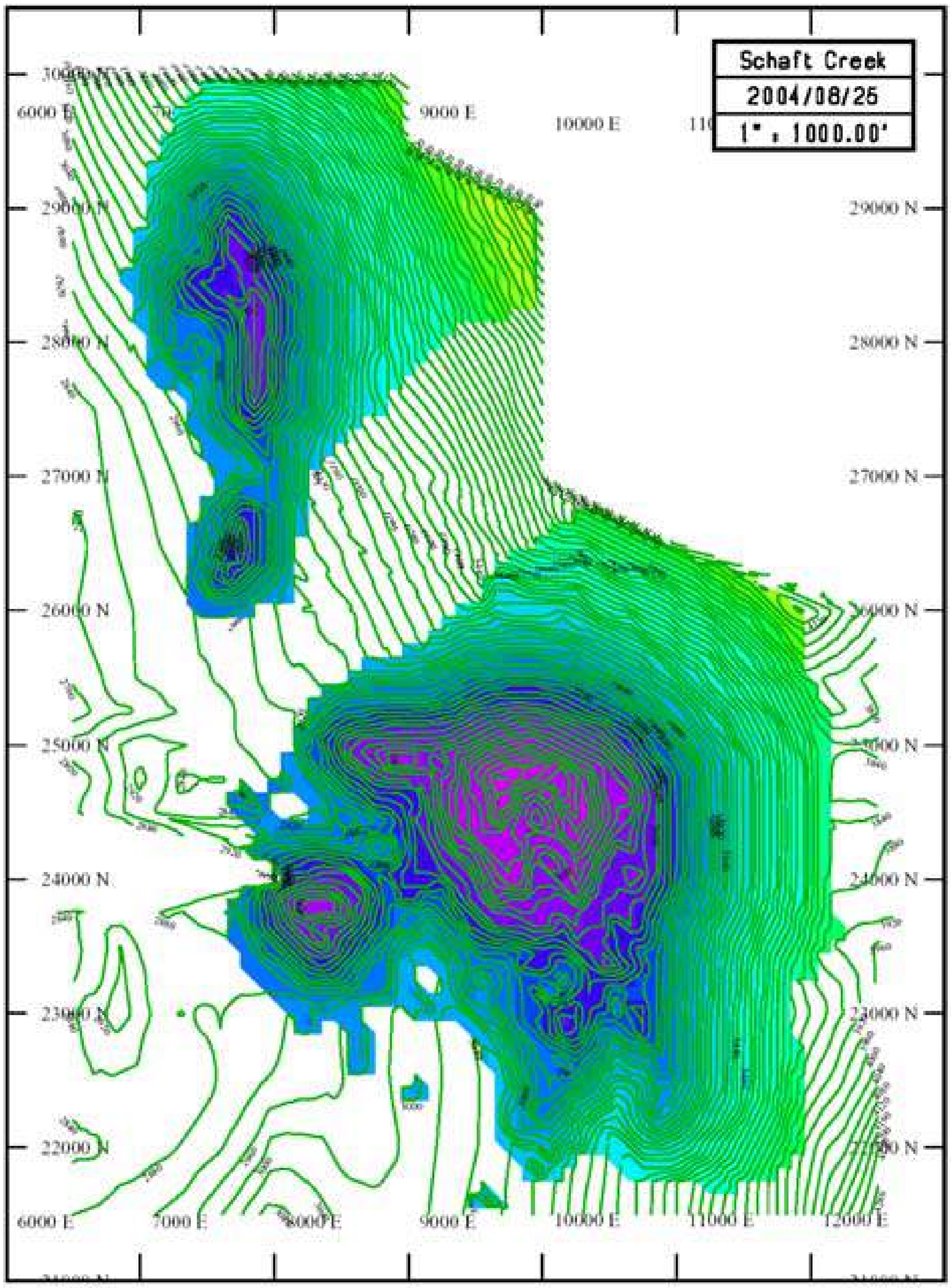
US$0.90/lb Cu Equivalent Optimized Pit Contours (Shaded)



US$1.00/lb Cu Equivalent Optimized Pit Contours (Shaded)

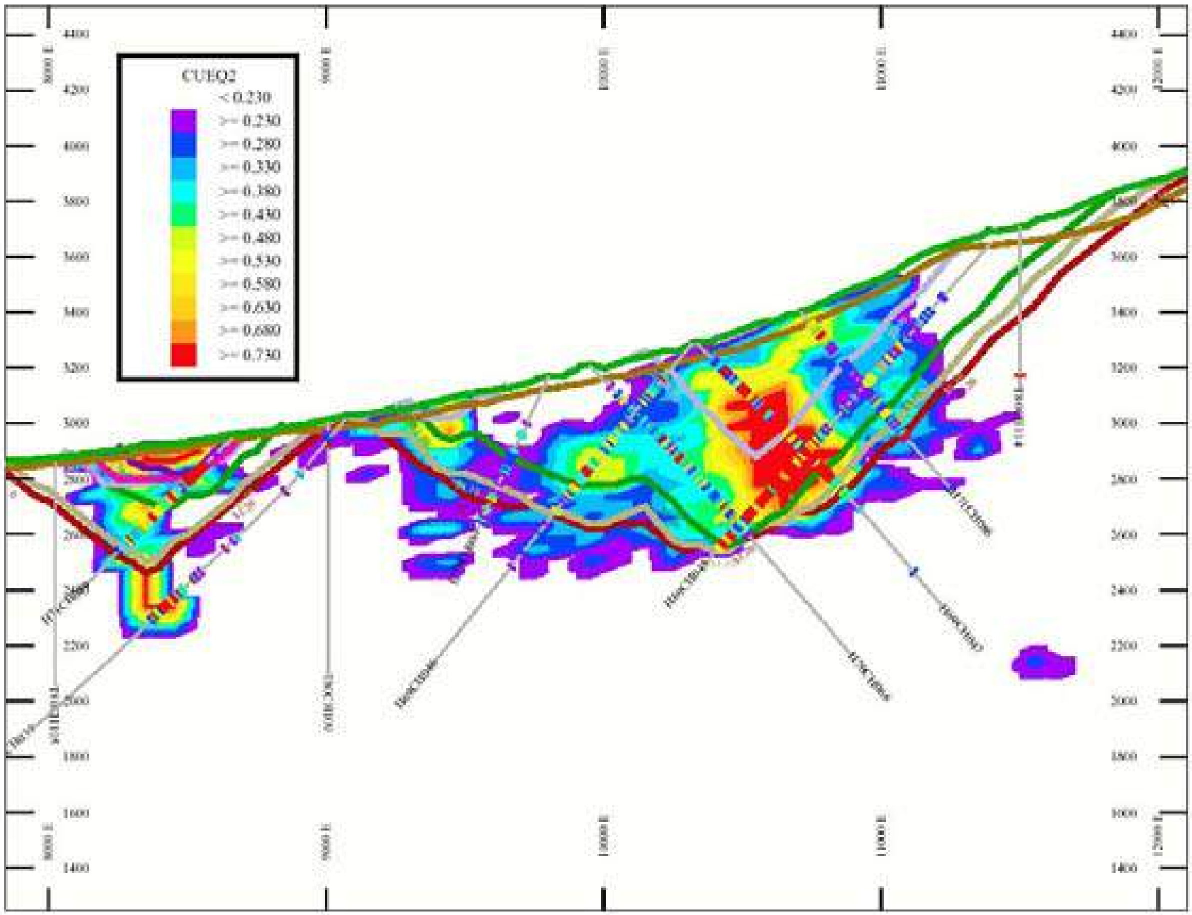


US$1.10/lb Cu Equivalent Optimized Pit Contours (Shaded)

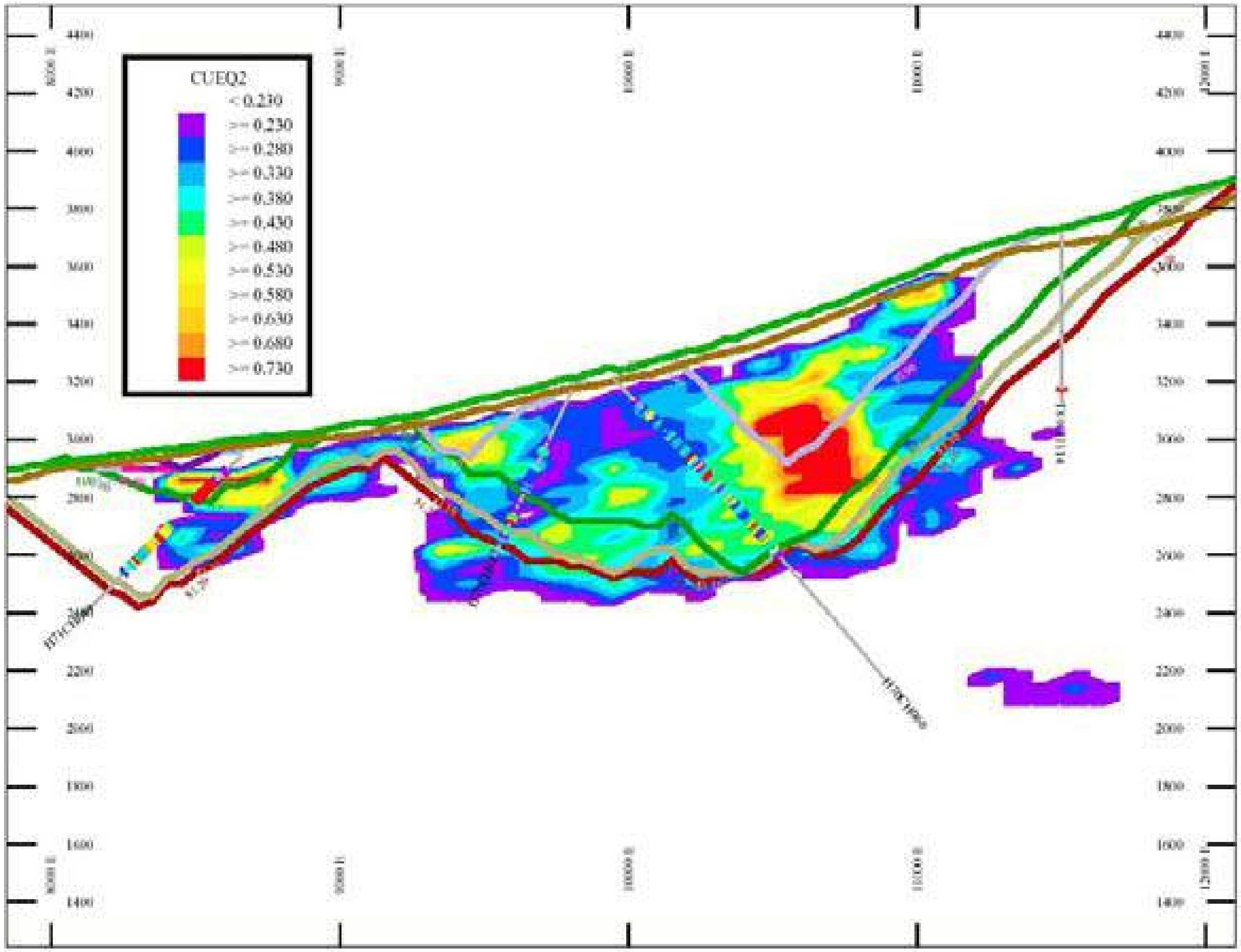


US$1.20/lb Cu Equivalent Optimized Pit Contours (Shaded)

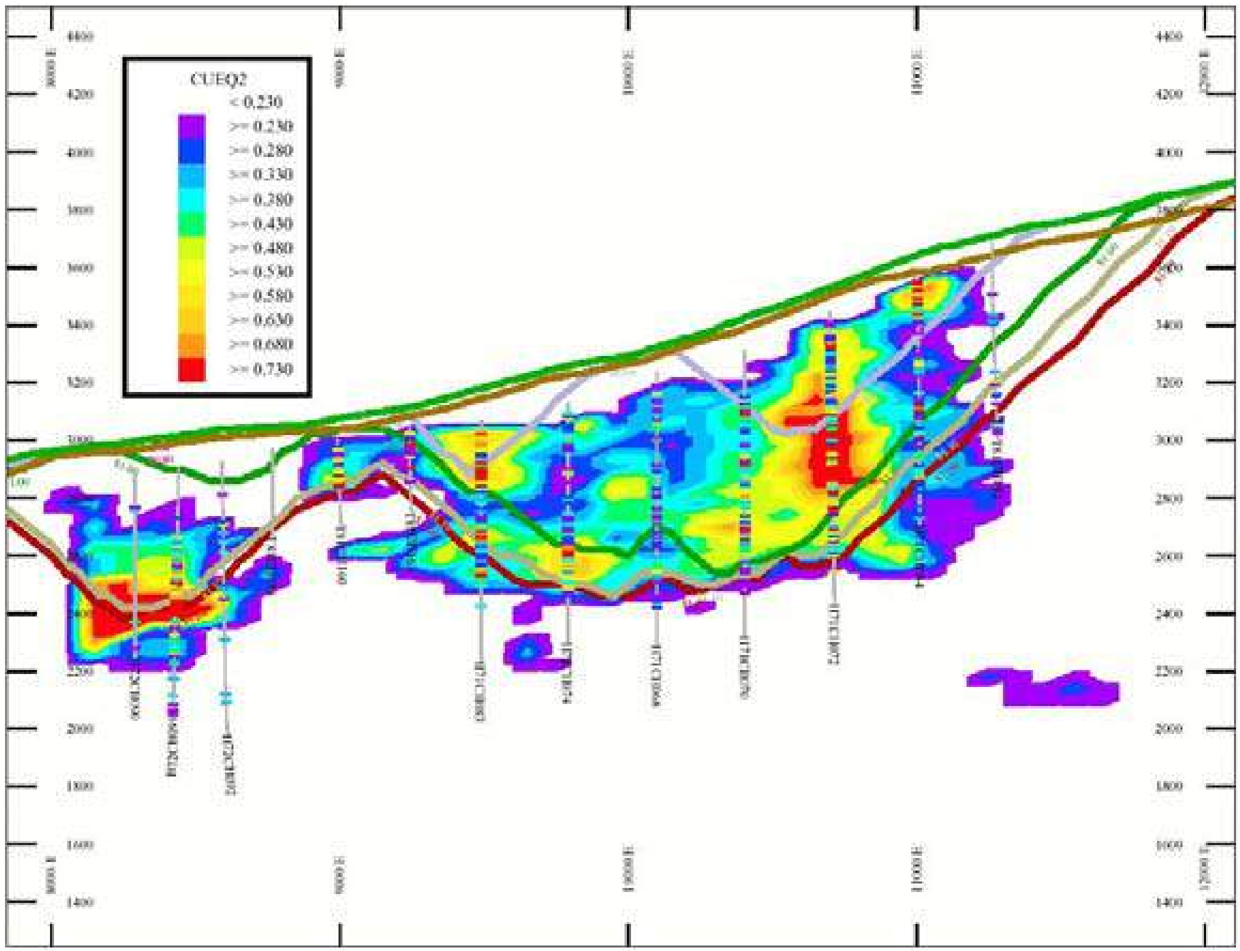
APPENDIX 2
CROSS SECTIONS



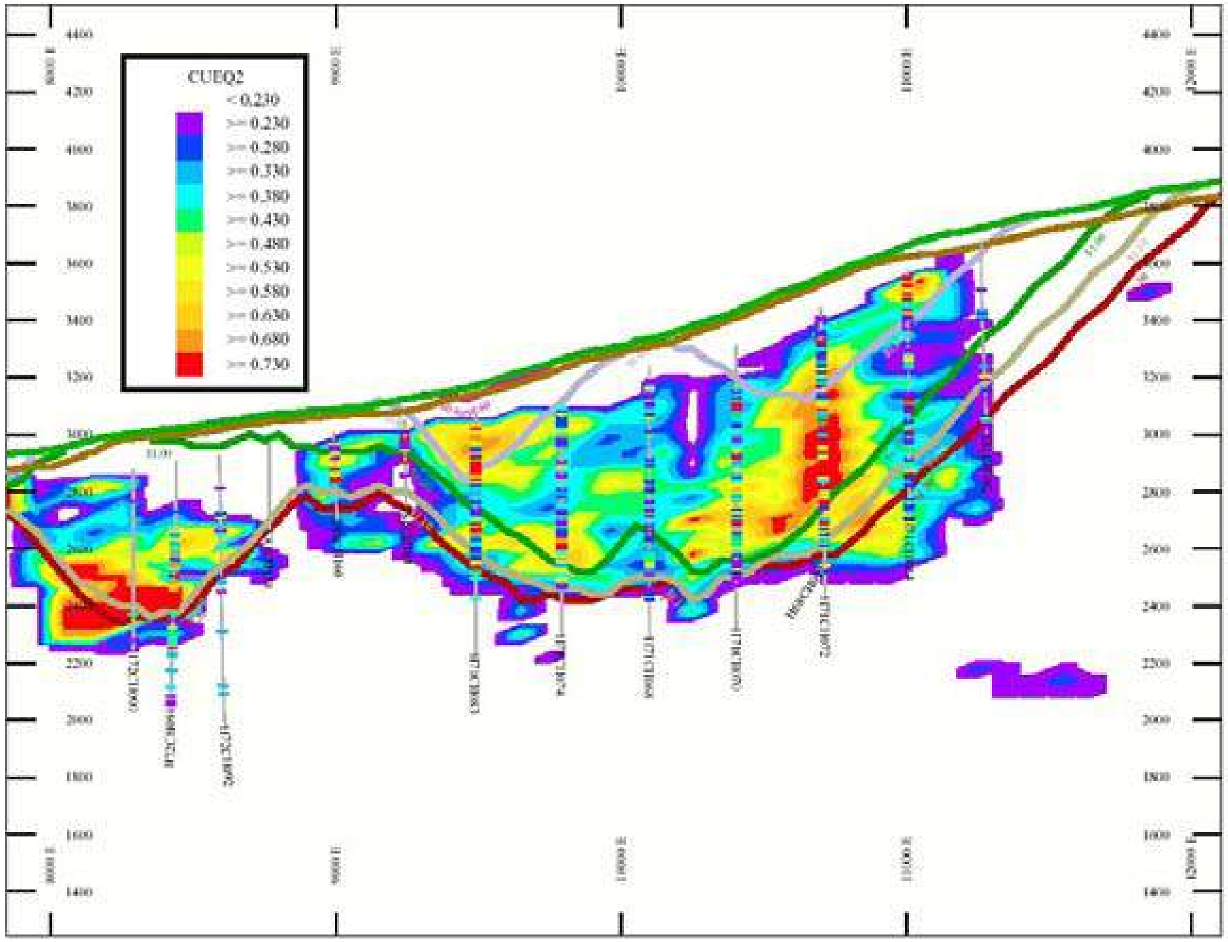
Cross Section at 23500N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



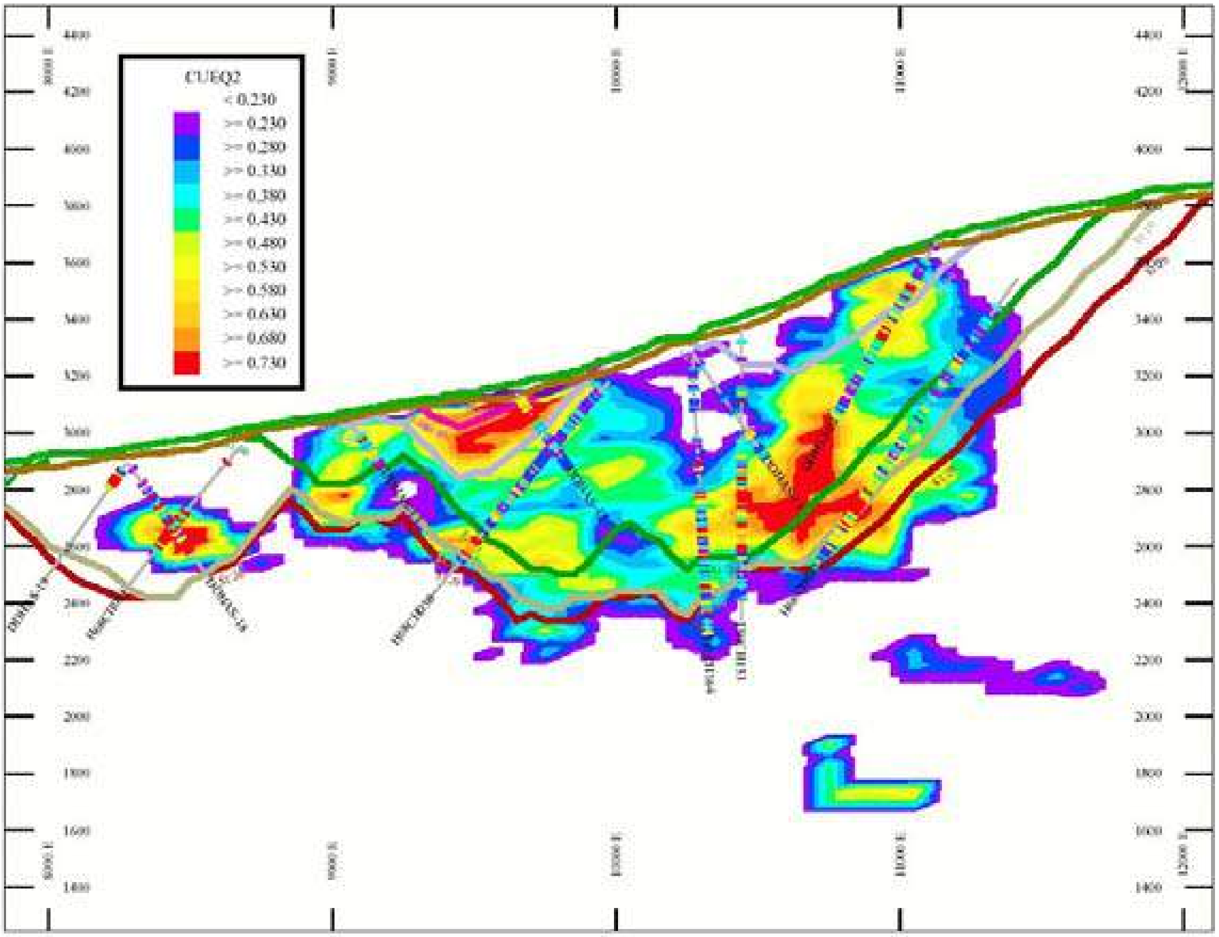
Cross Section at 23600N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



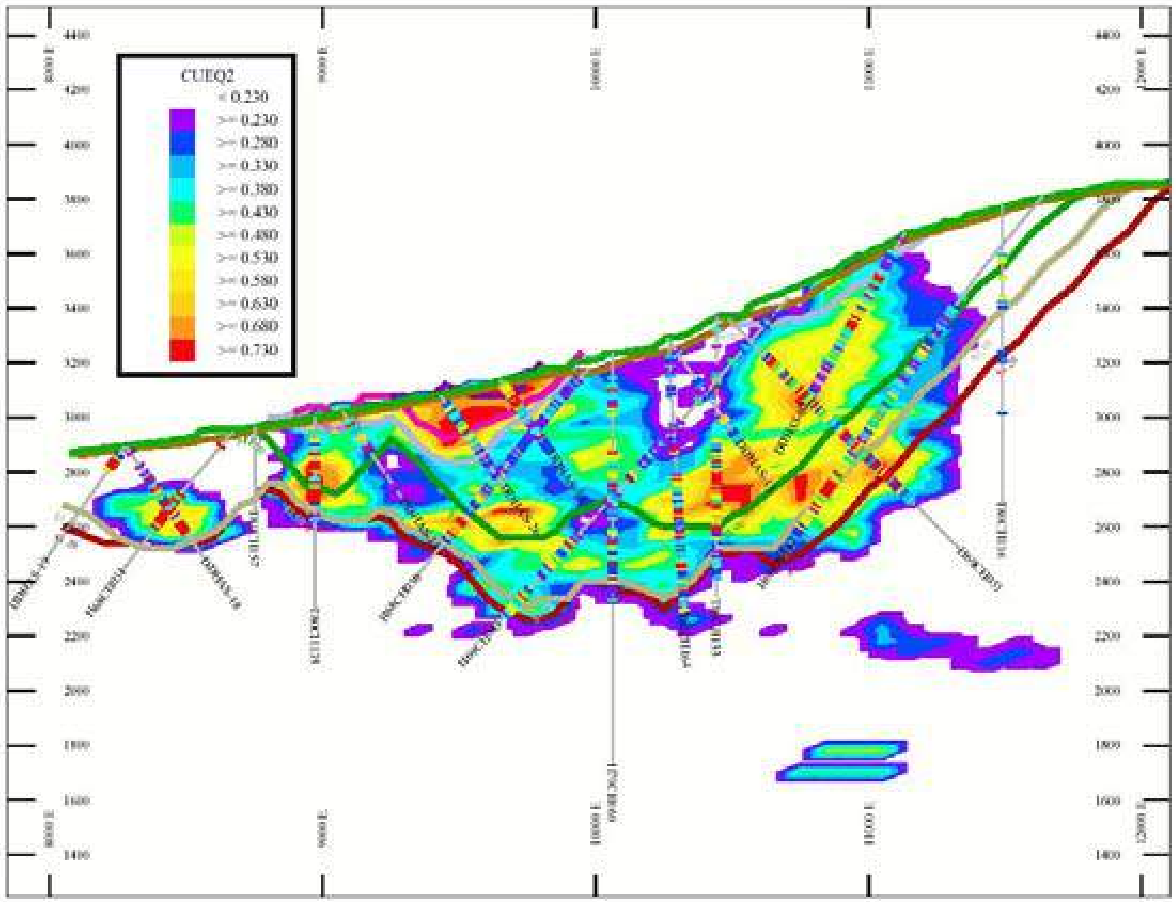
Cross Section at 23700N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



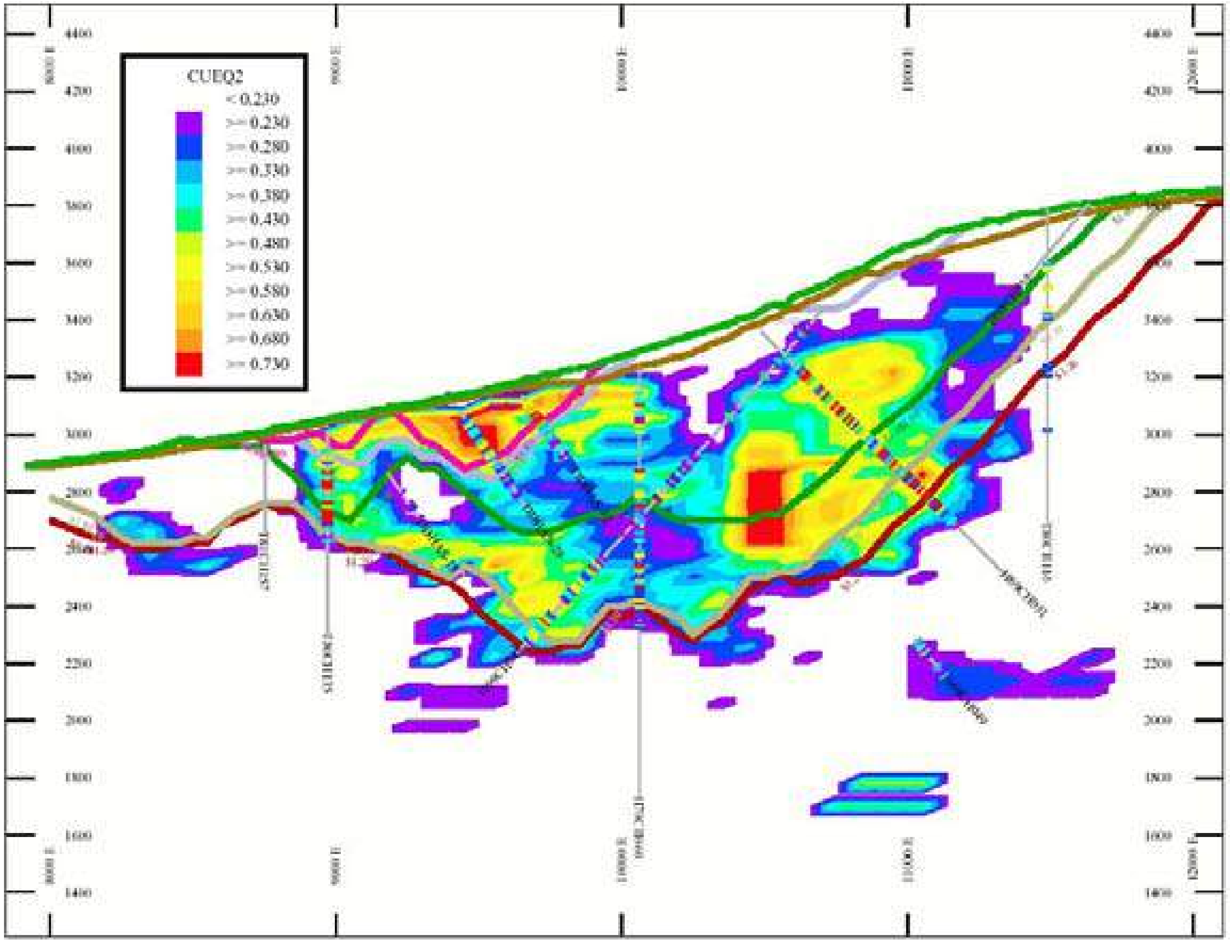
Cross Section at 23800N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



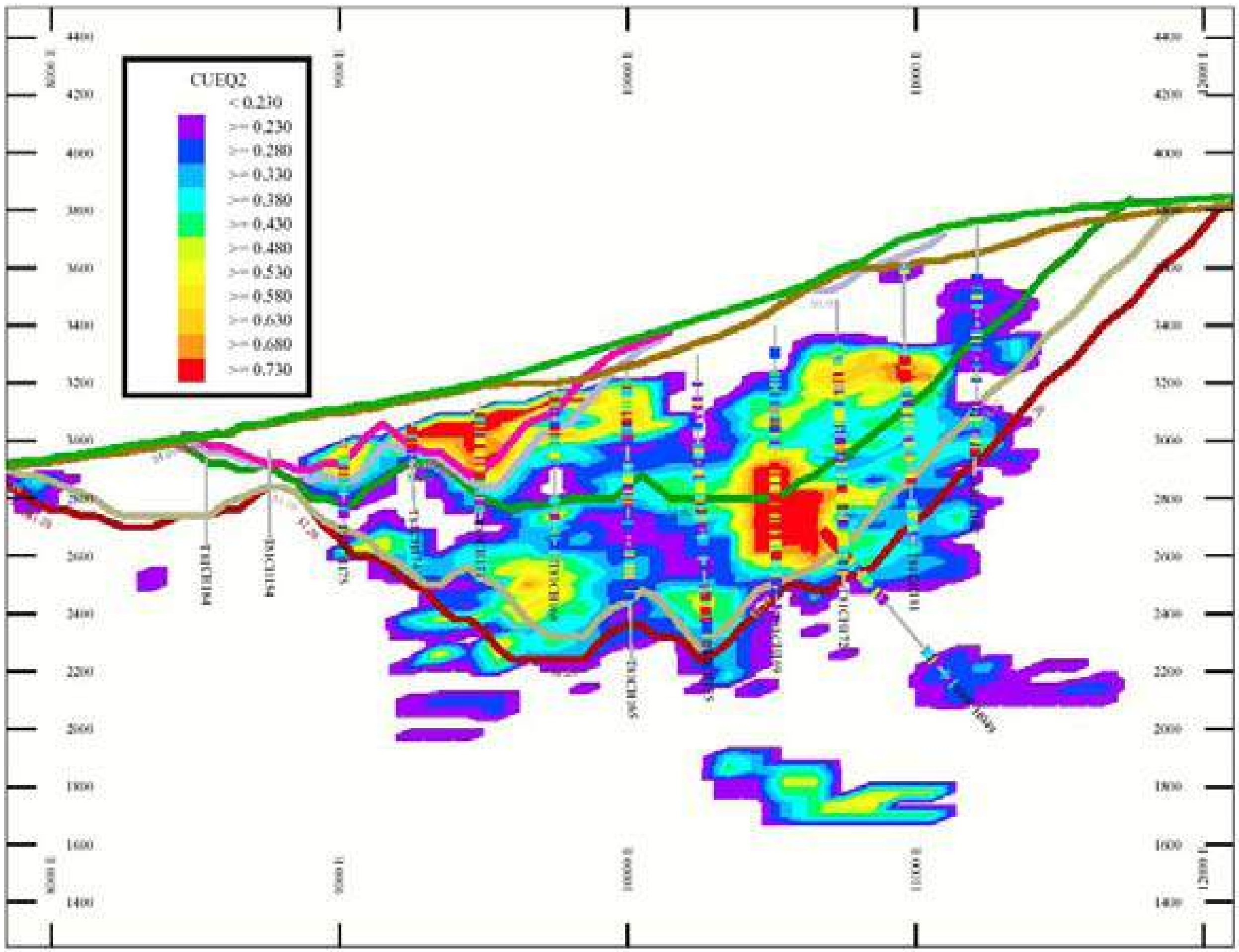
Cross Section at 23900N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Cross Section at 24000N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Cross Section at 24100N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Cross Section at 24200N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

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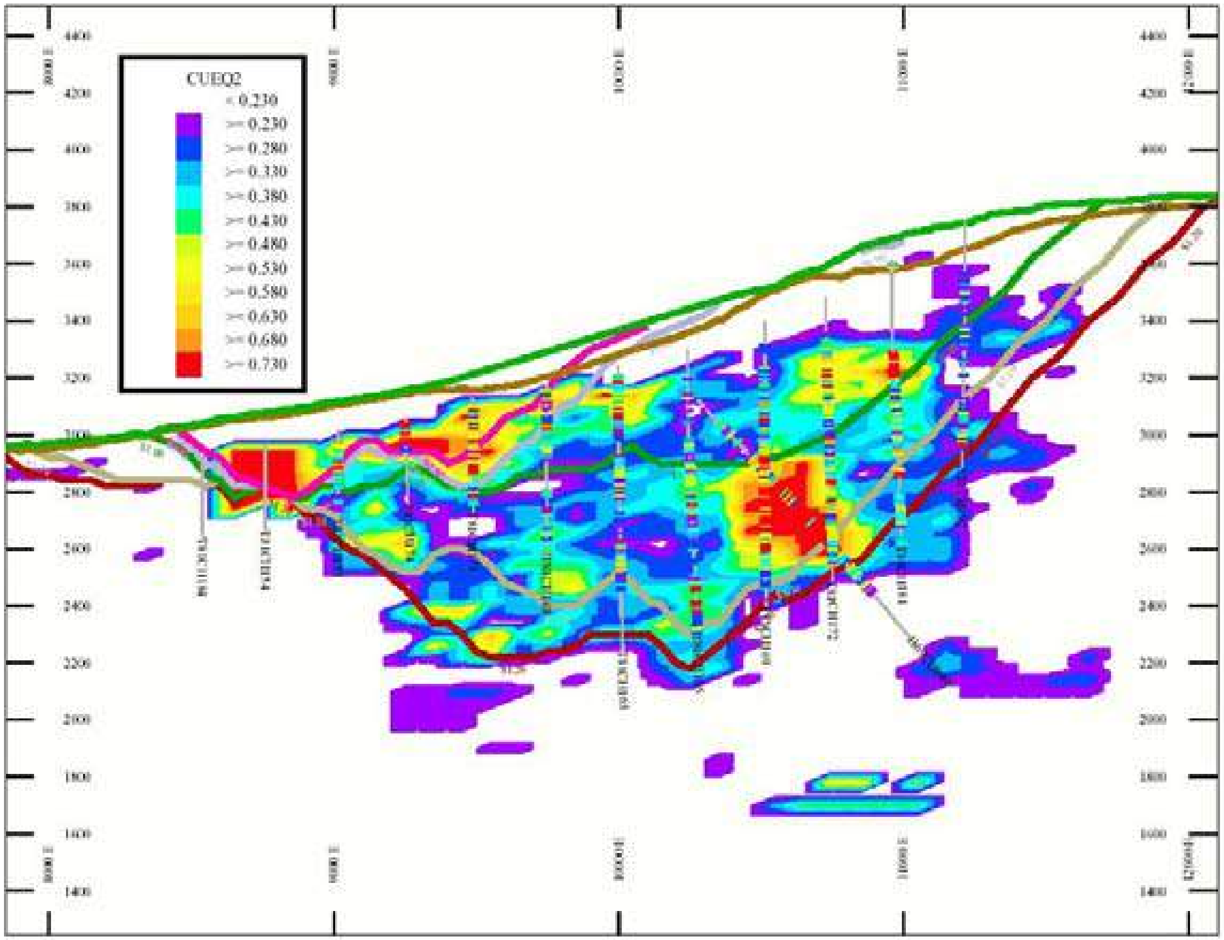
September 20, 2004



Cross Section at 24300N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

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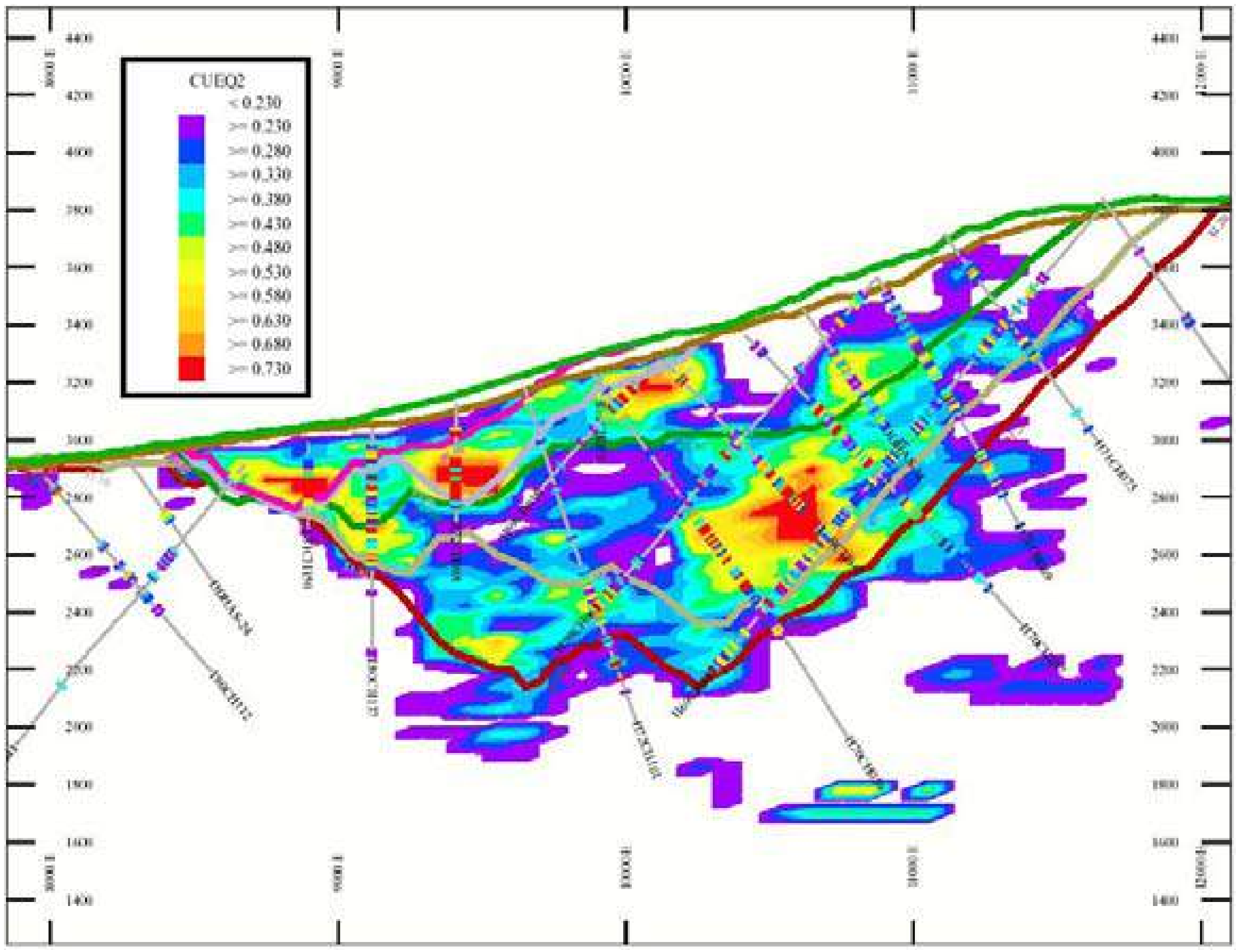
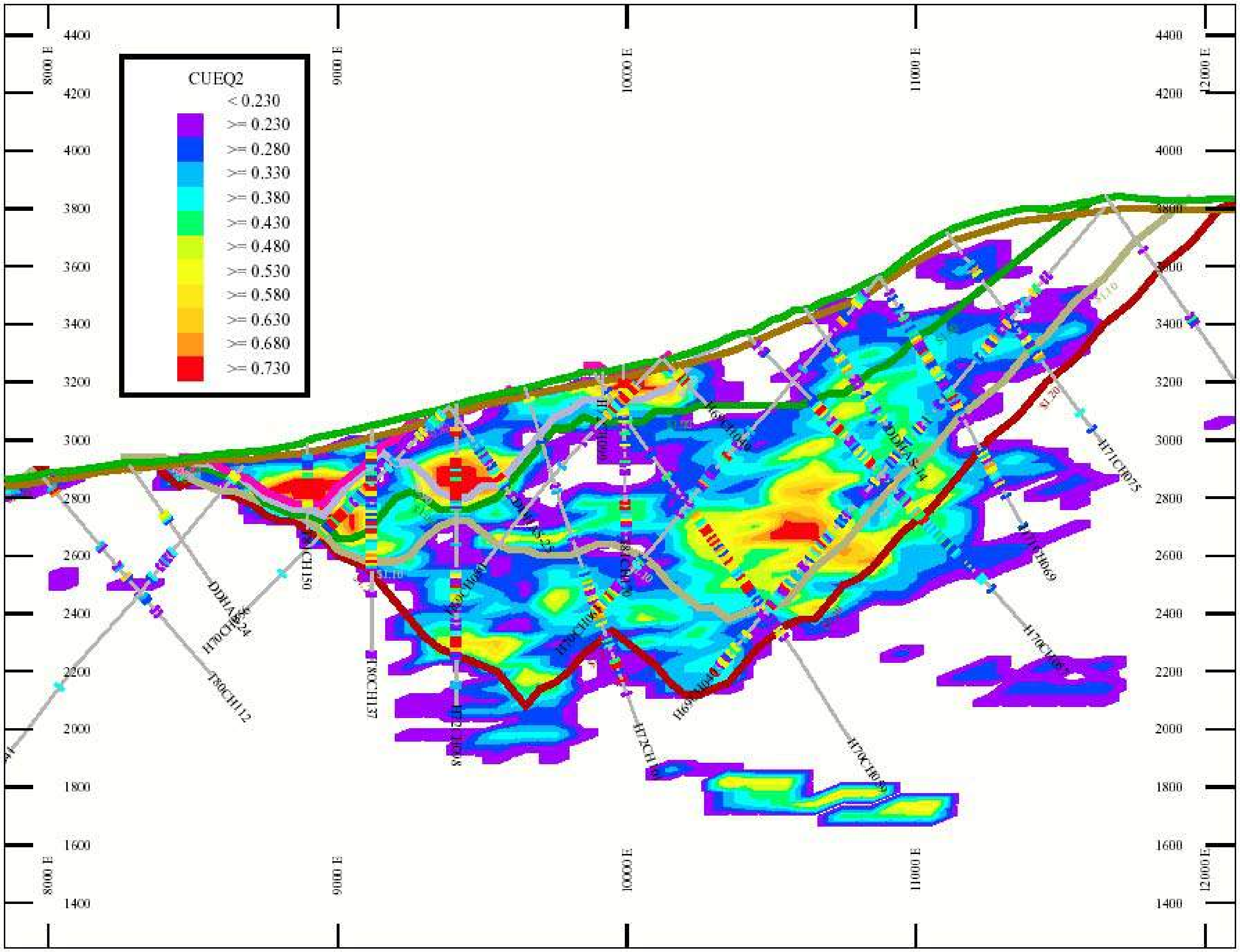
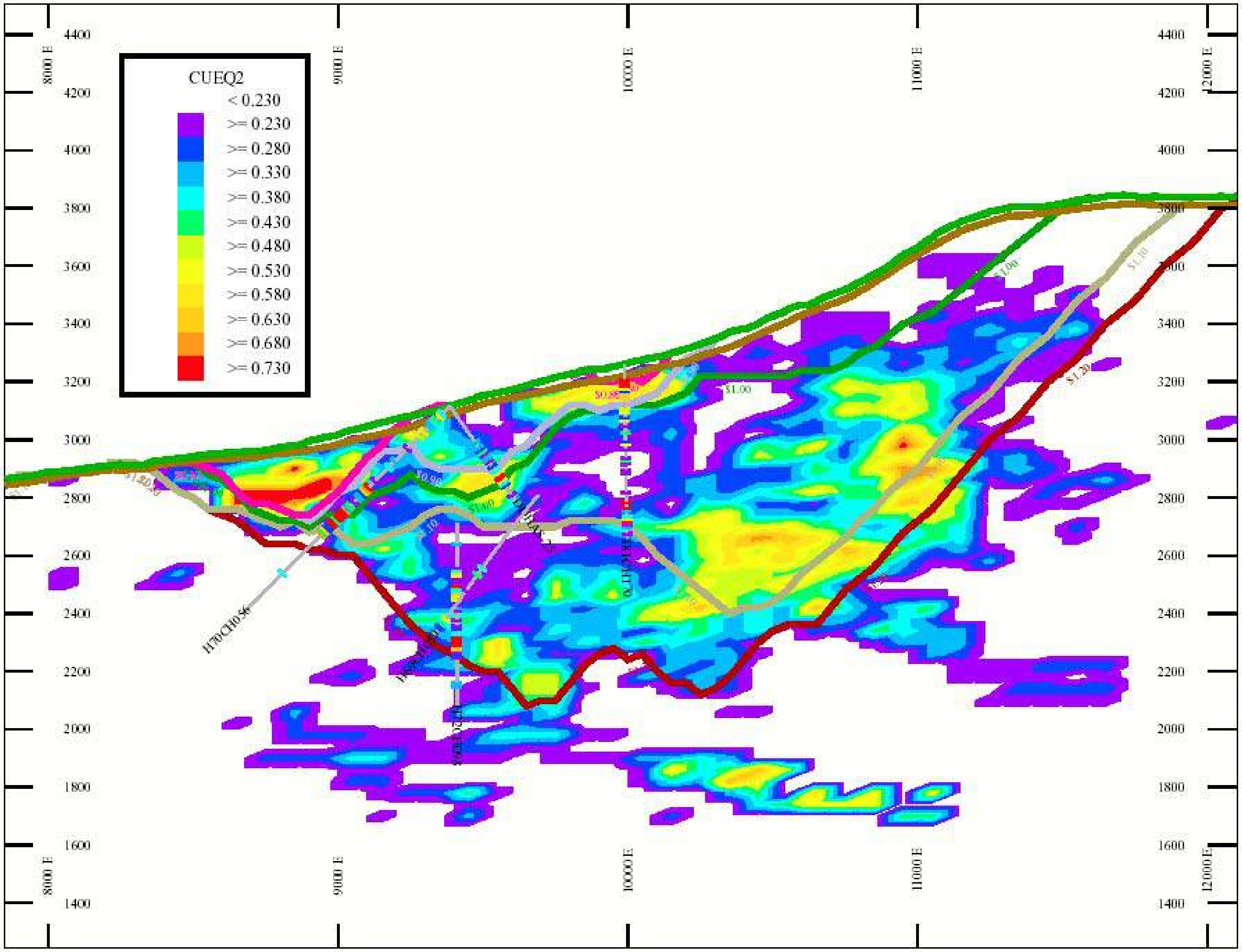


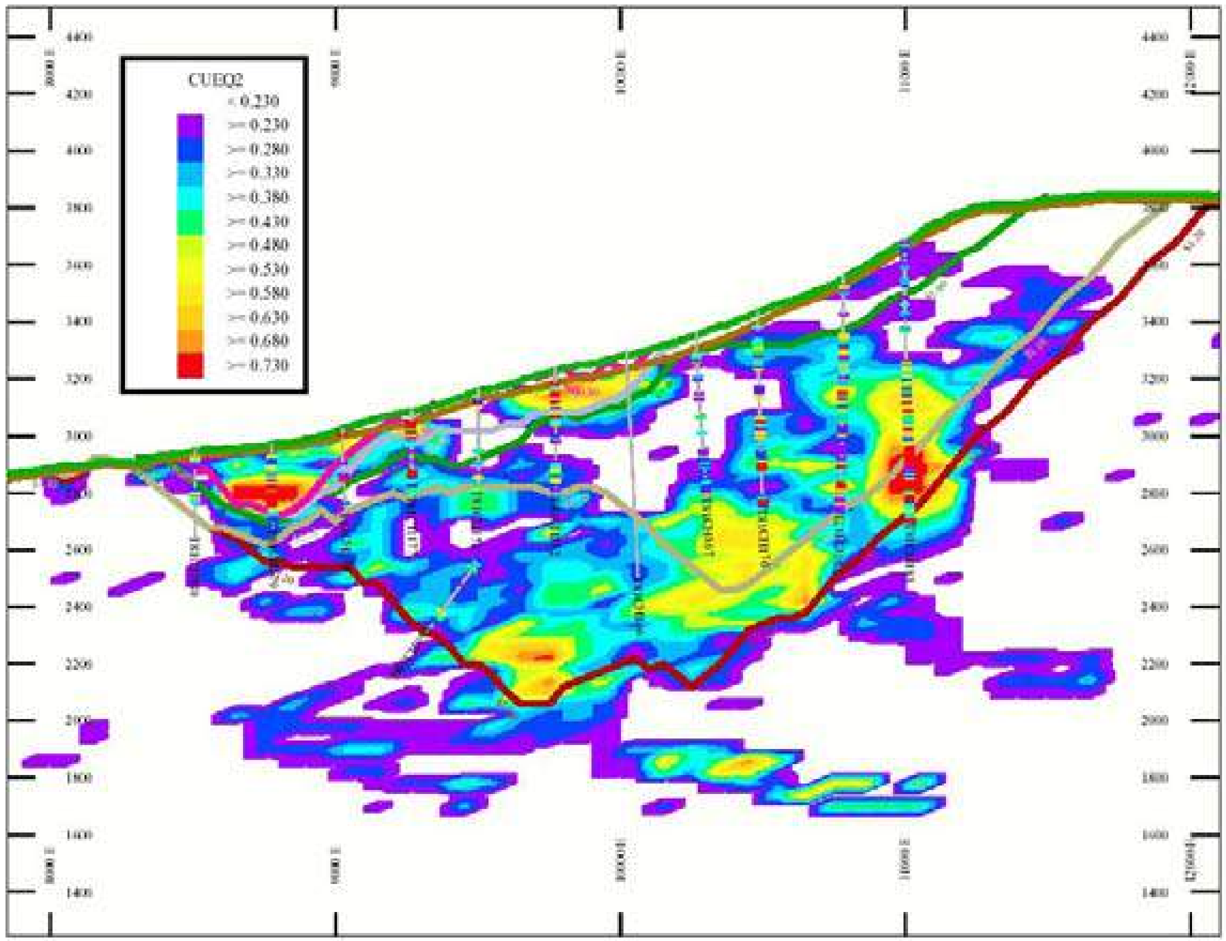
Cross Section at 24400N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



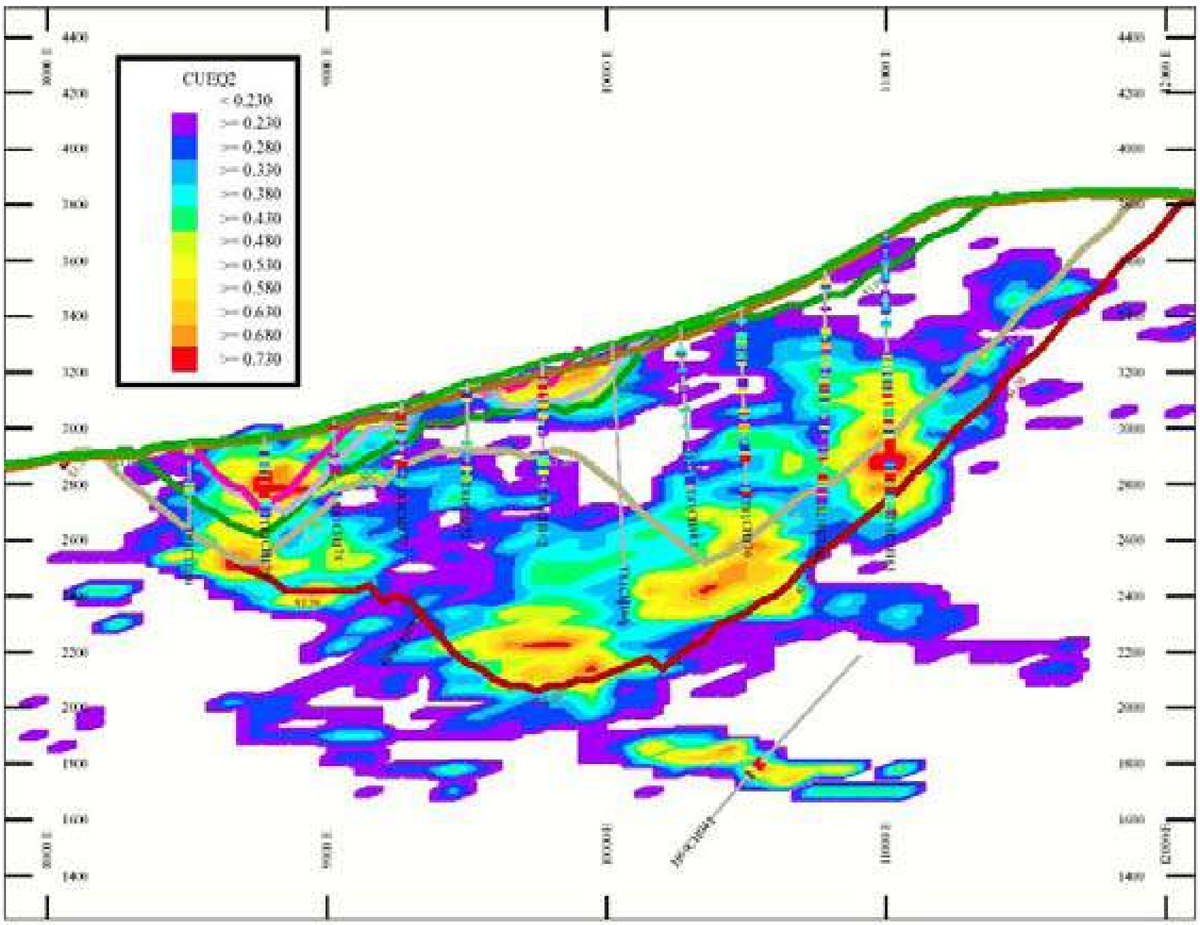
Cross Section at 24500N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



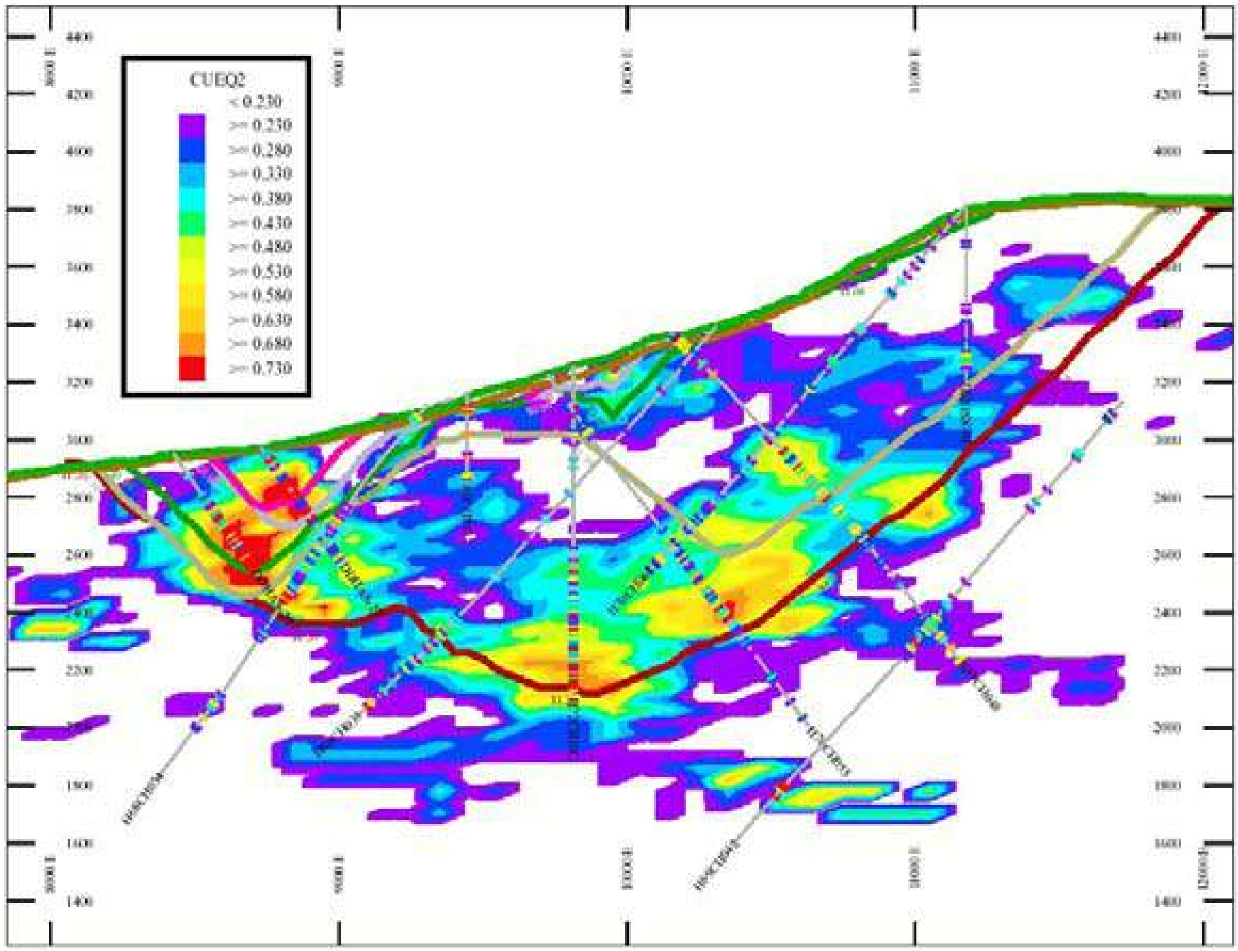
Cross Section at 24600N Showing Optimized Pit Outline and Grade Model with Net Copper Equivalent



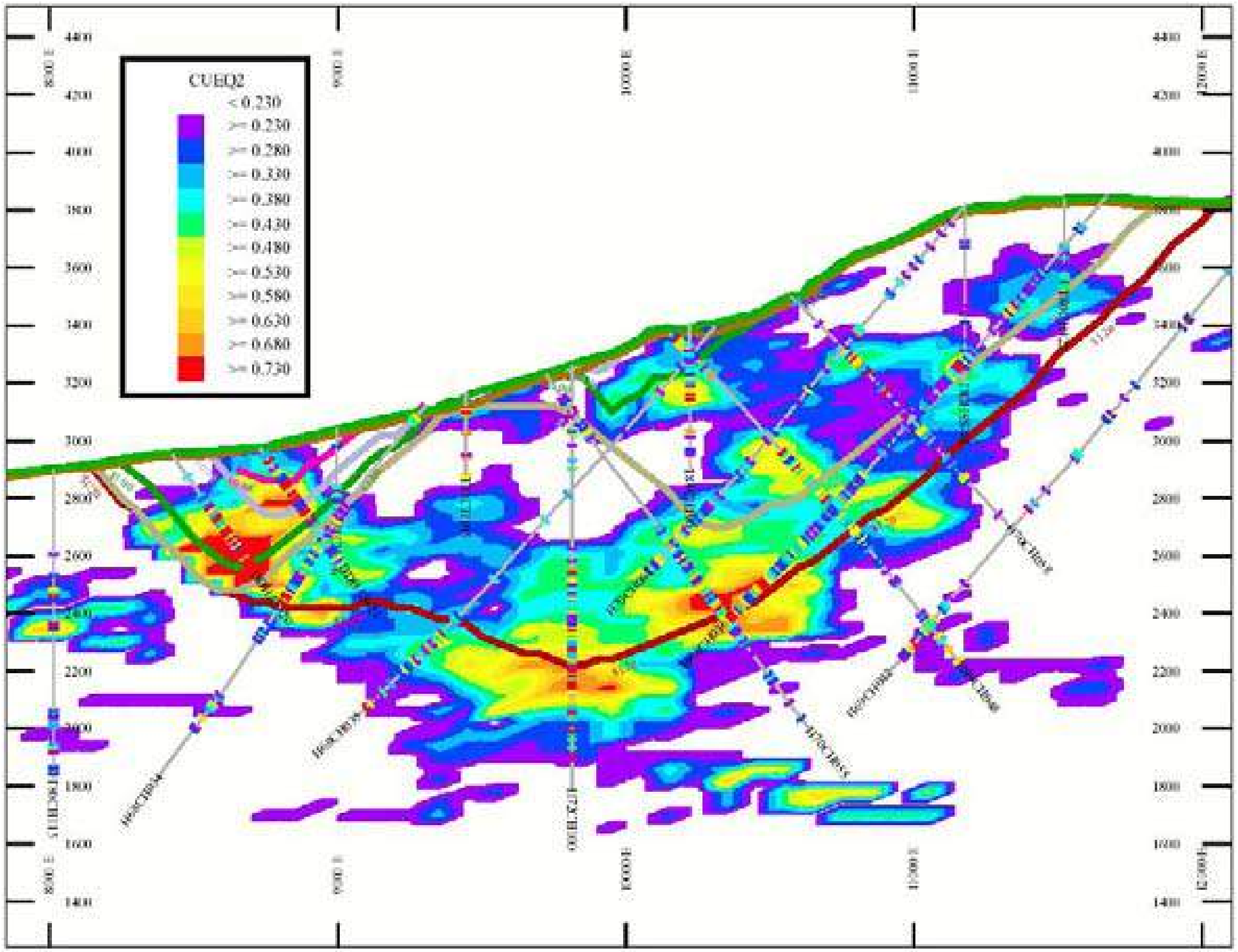
Cross Section at 24700N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



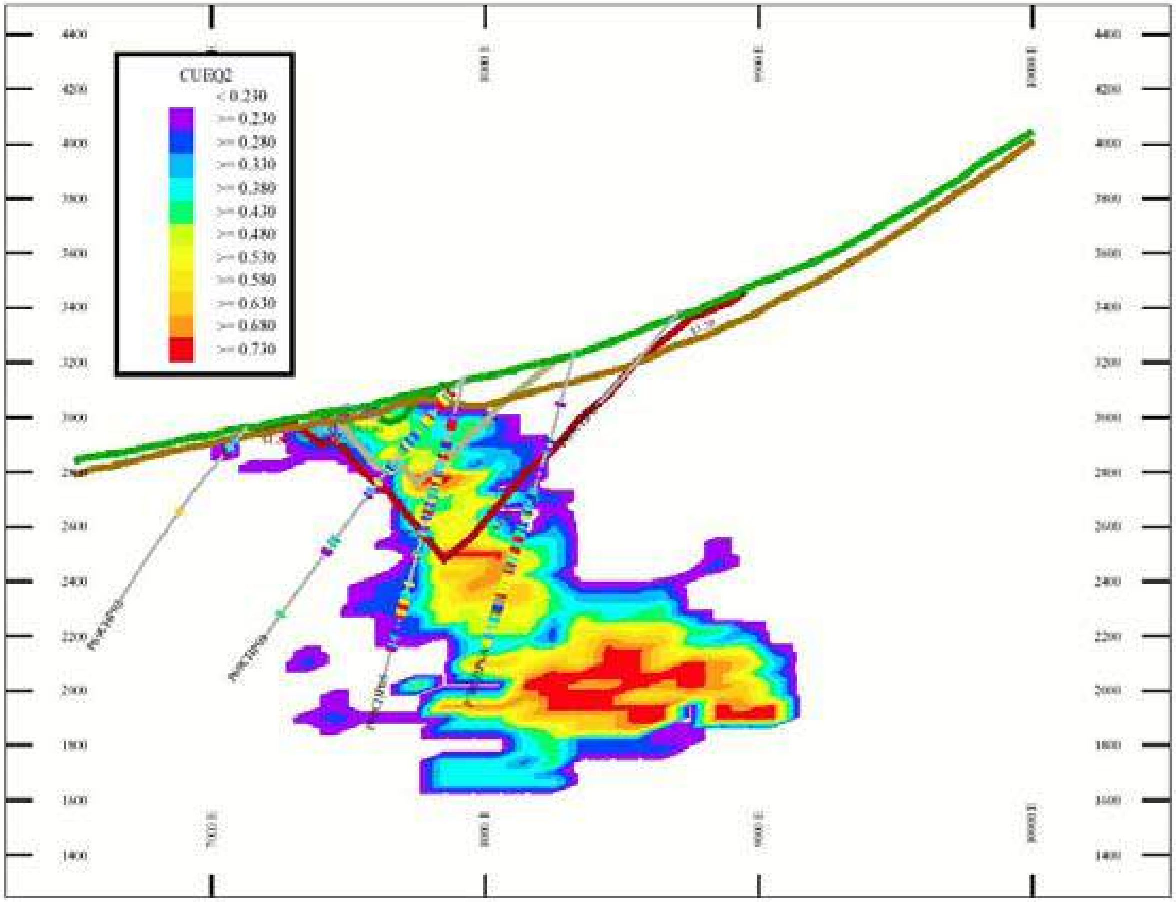
Cross Section at 24800N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Cross Section at 24900N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Cross Section at 25000N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Cross Section at 27700N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

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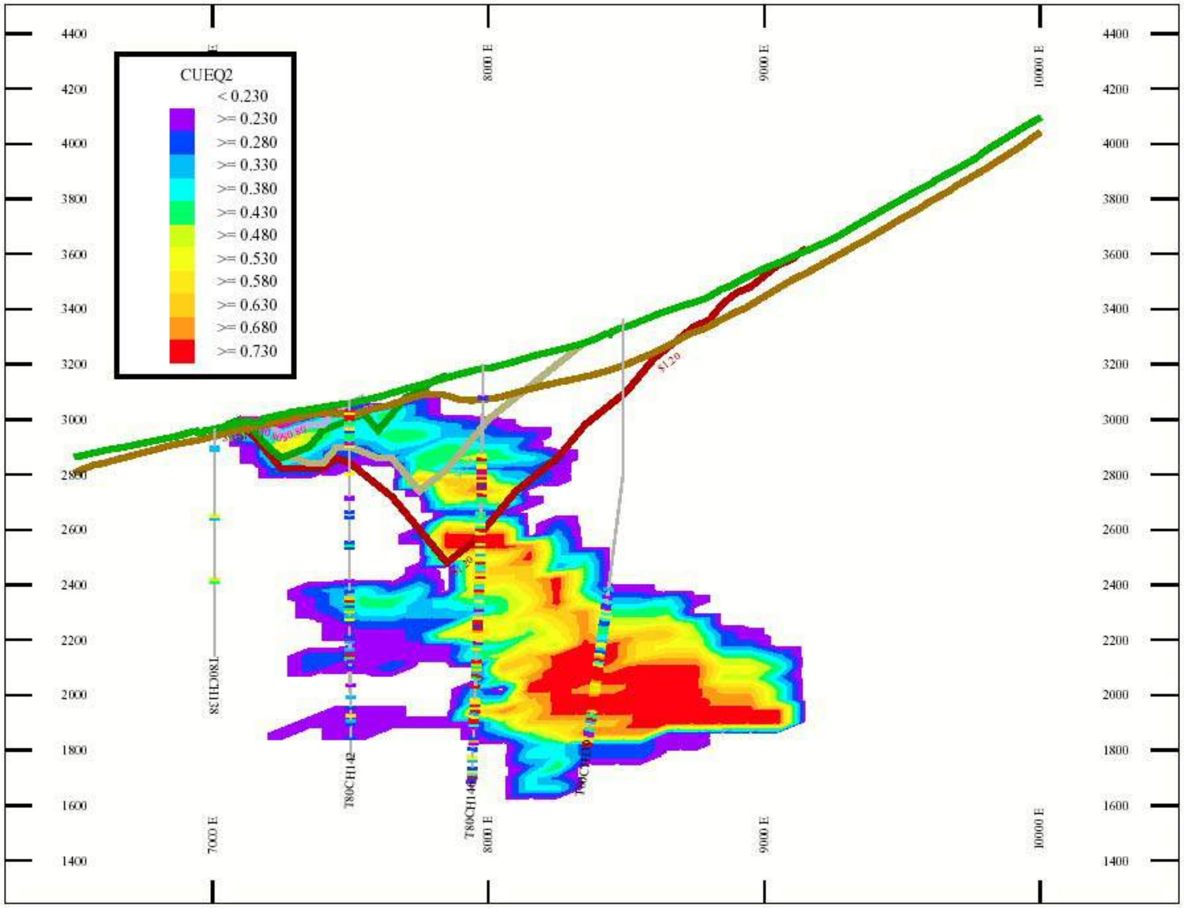
September 20, 2004



Cross Section at 27800N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

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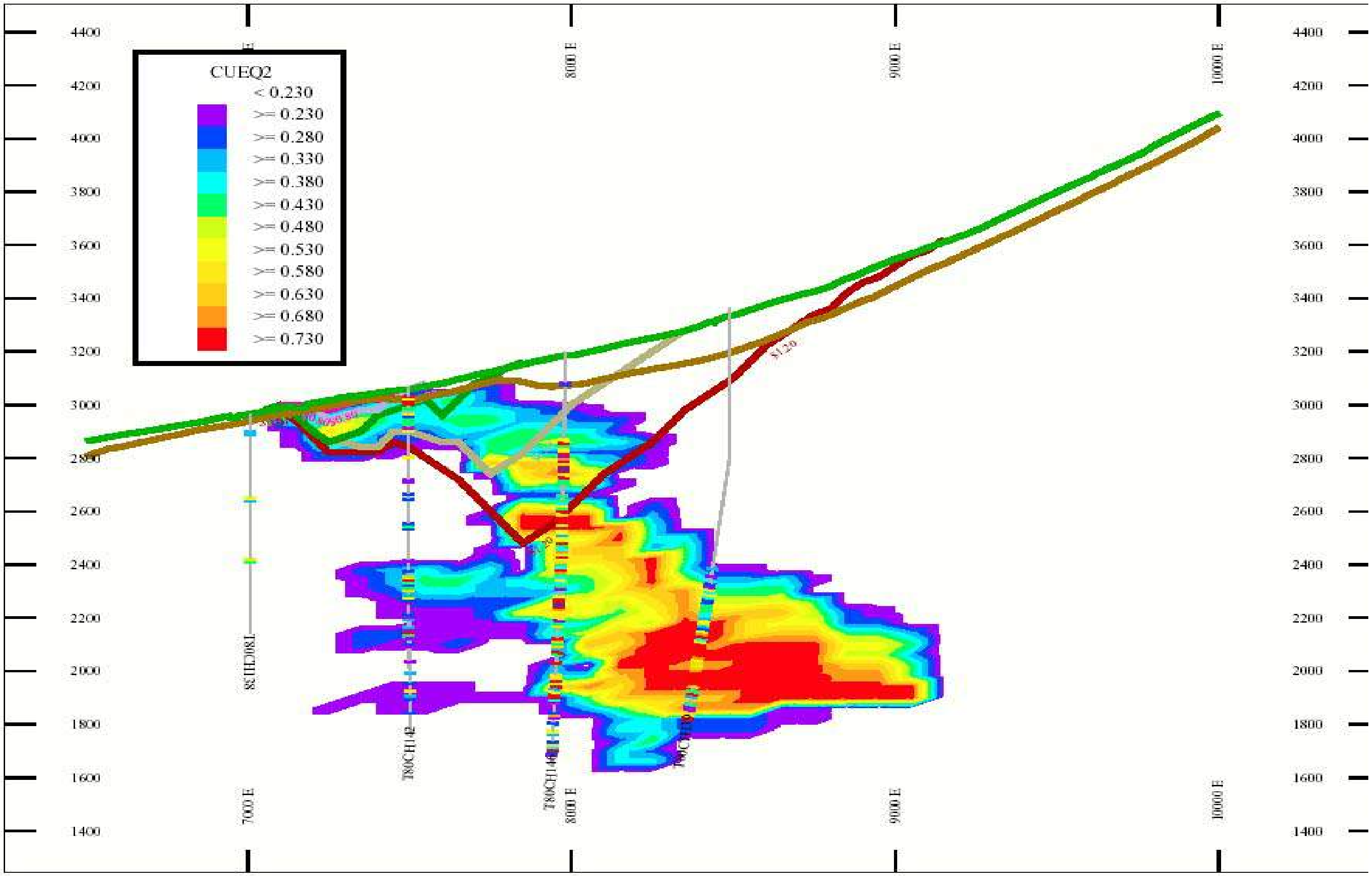
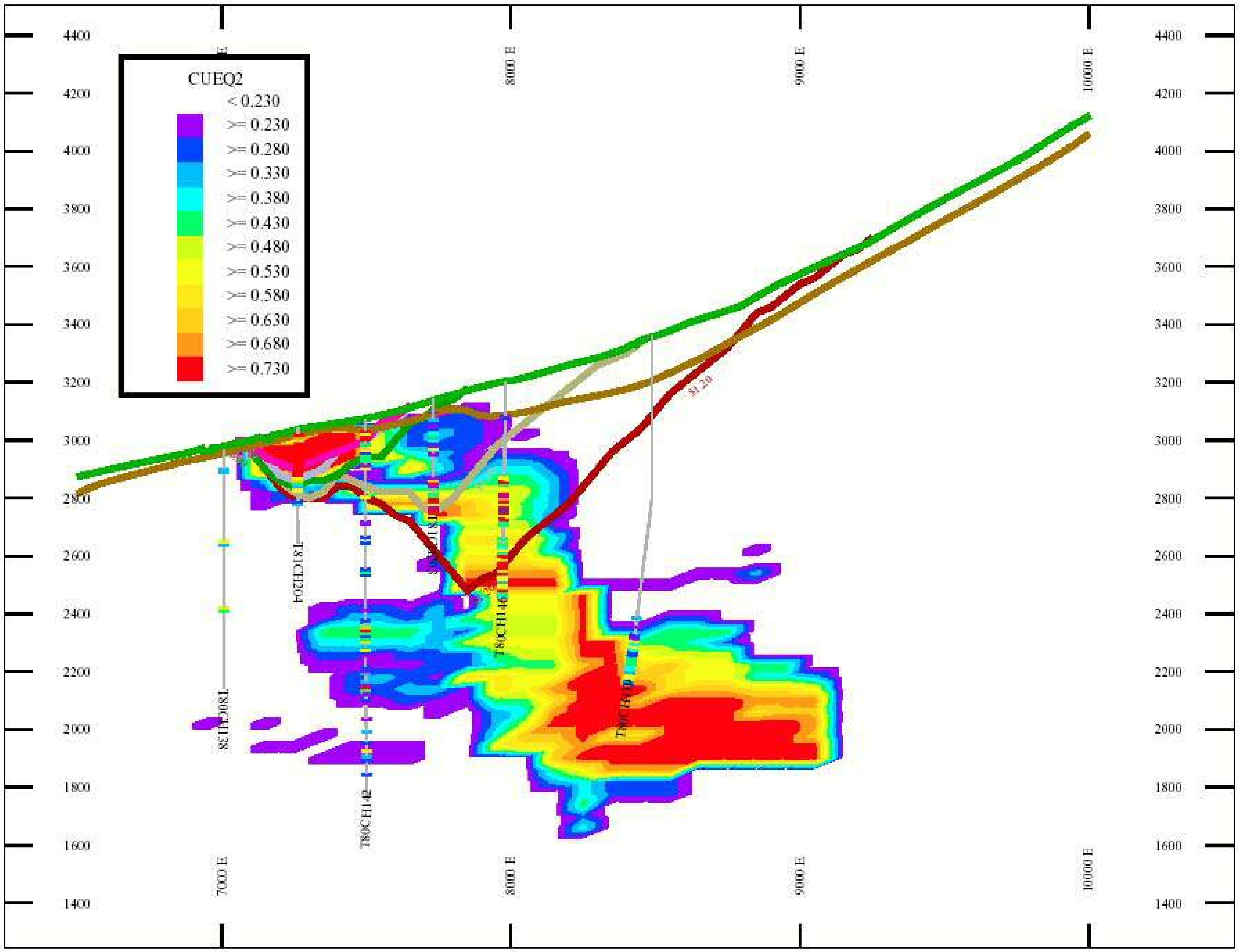
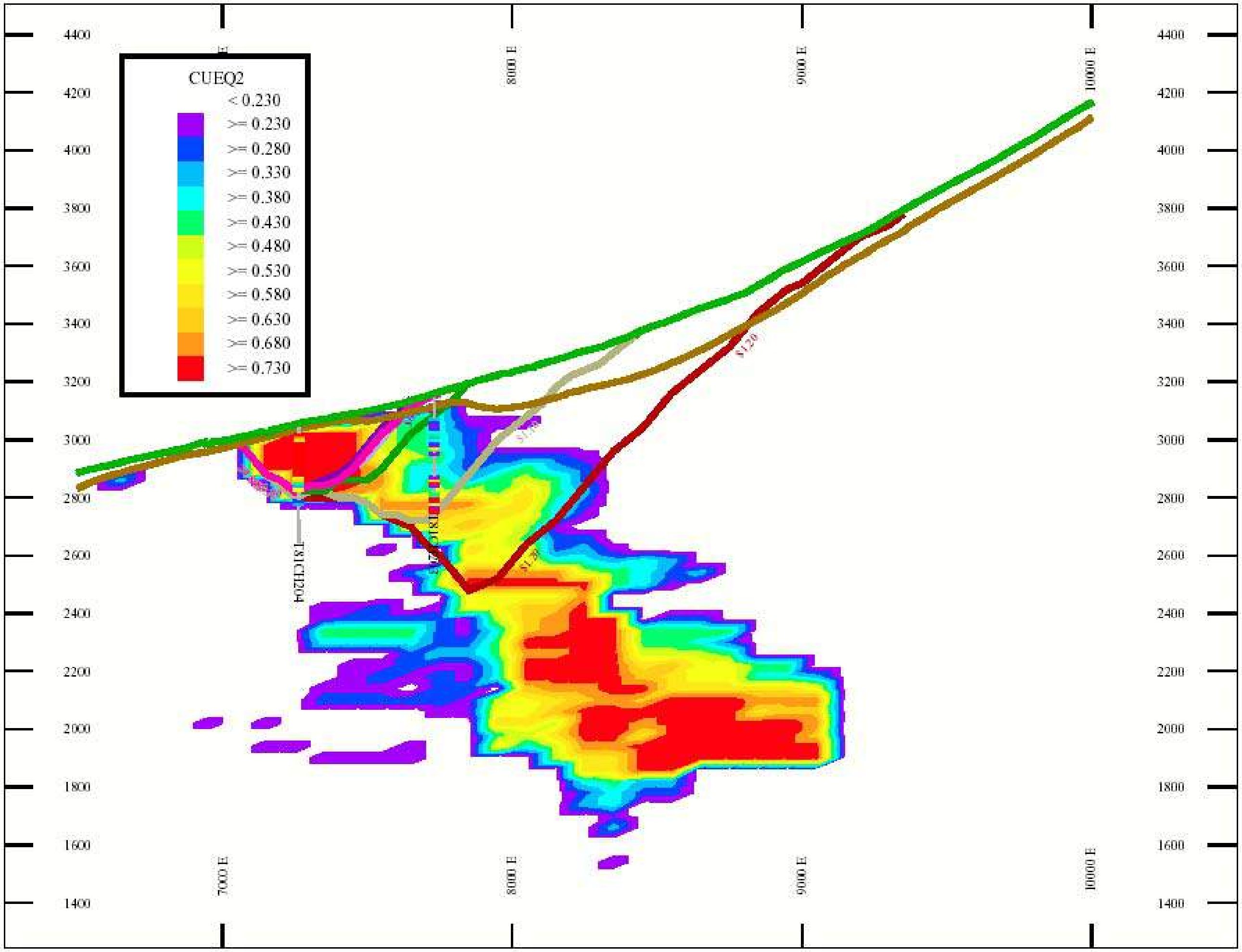


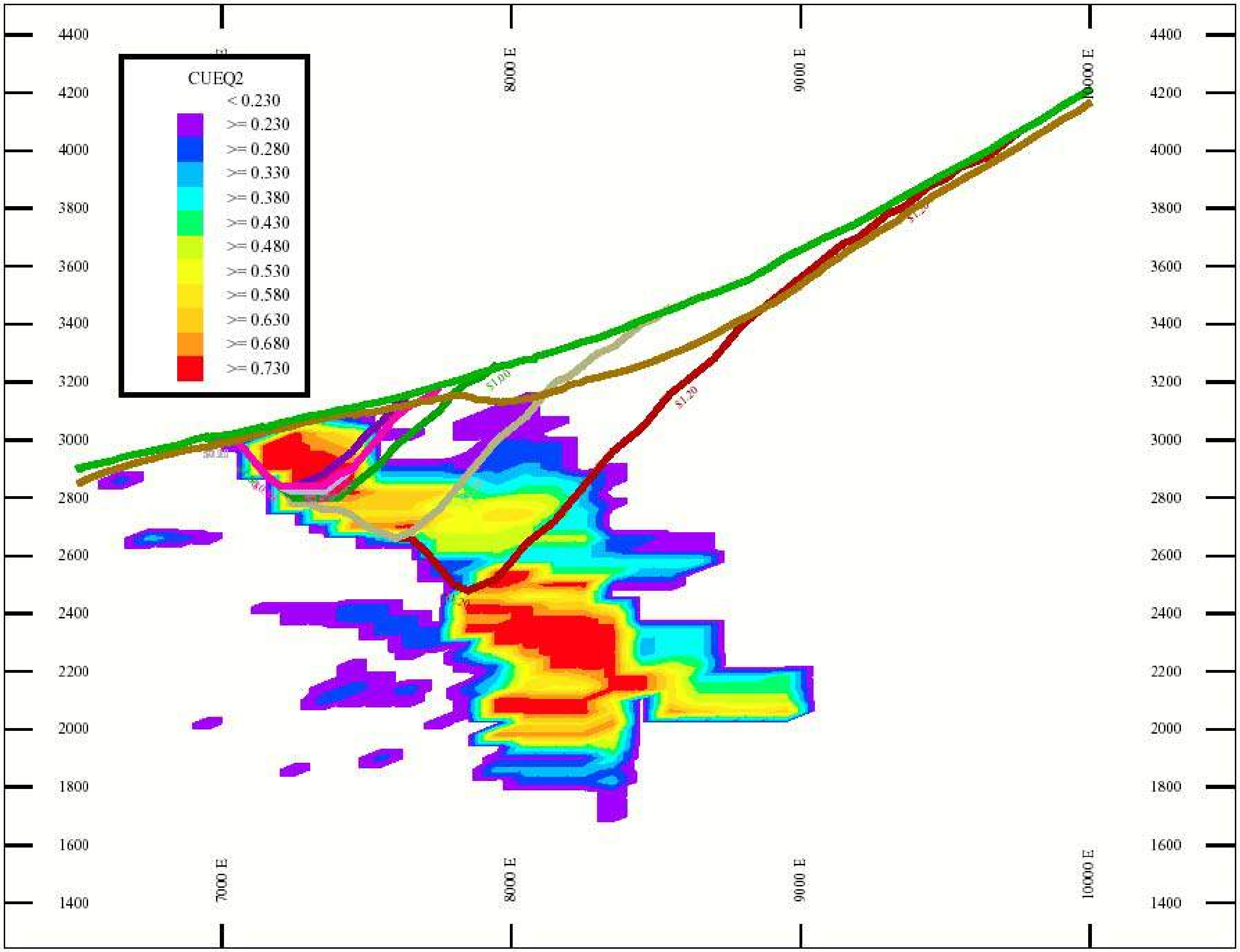
Cross Section at 27900N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Cross Section at 28000N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

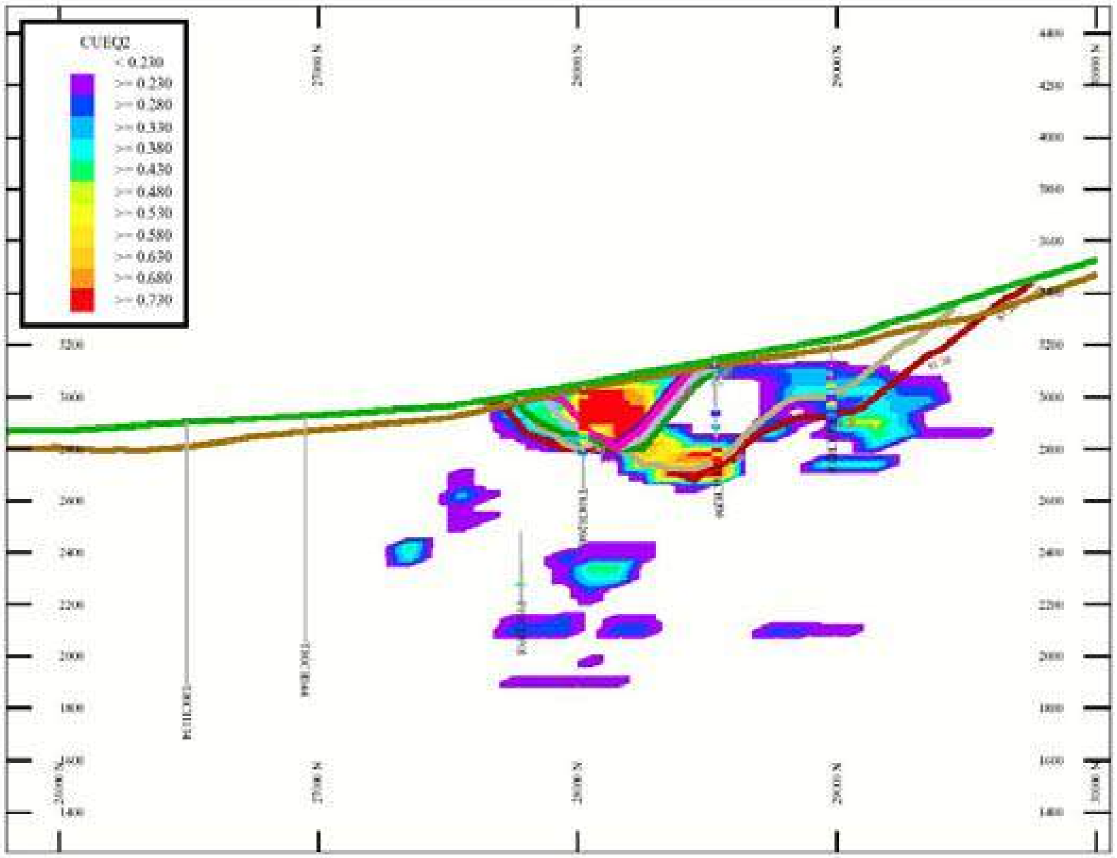


Cross Section at 28100N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

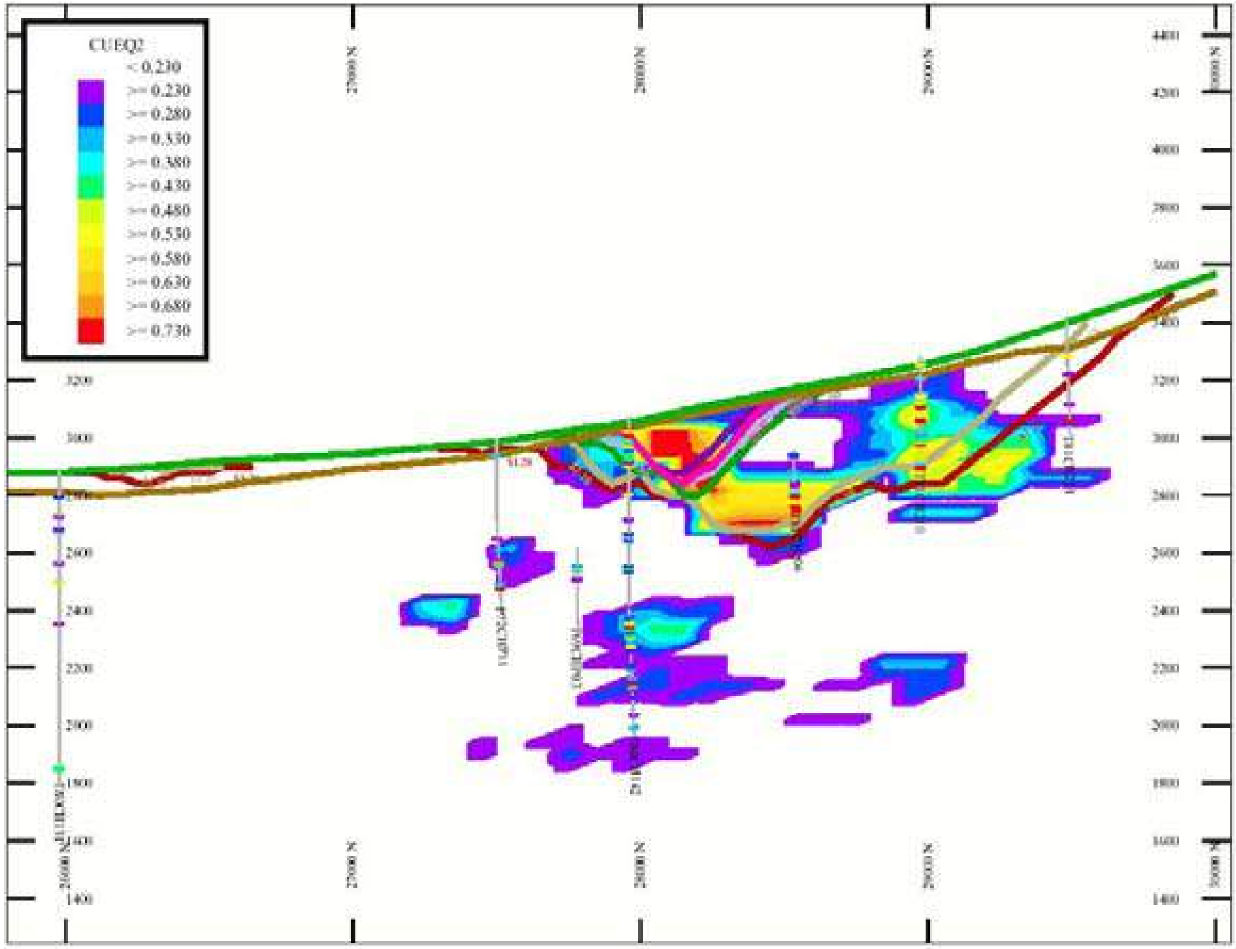


Cross Section at 28200N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

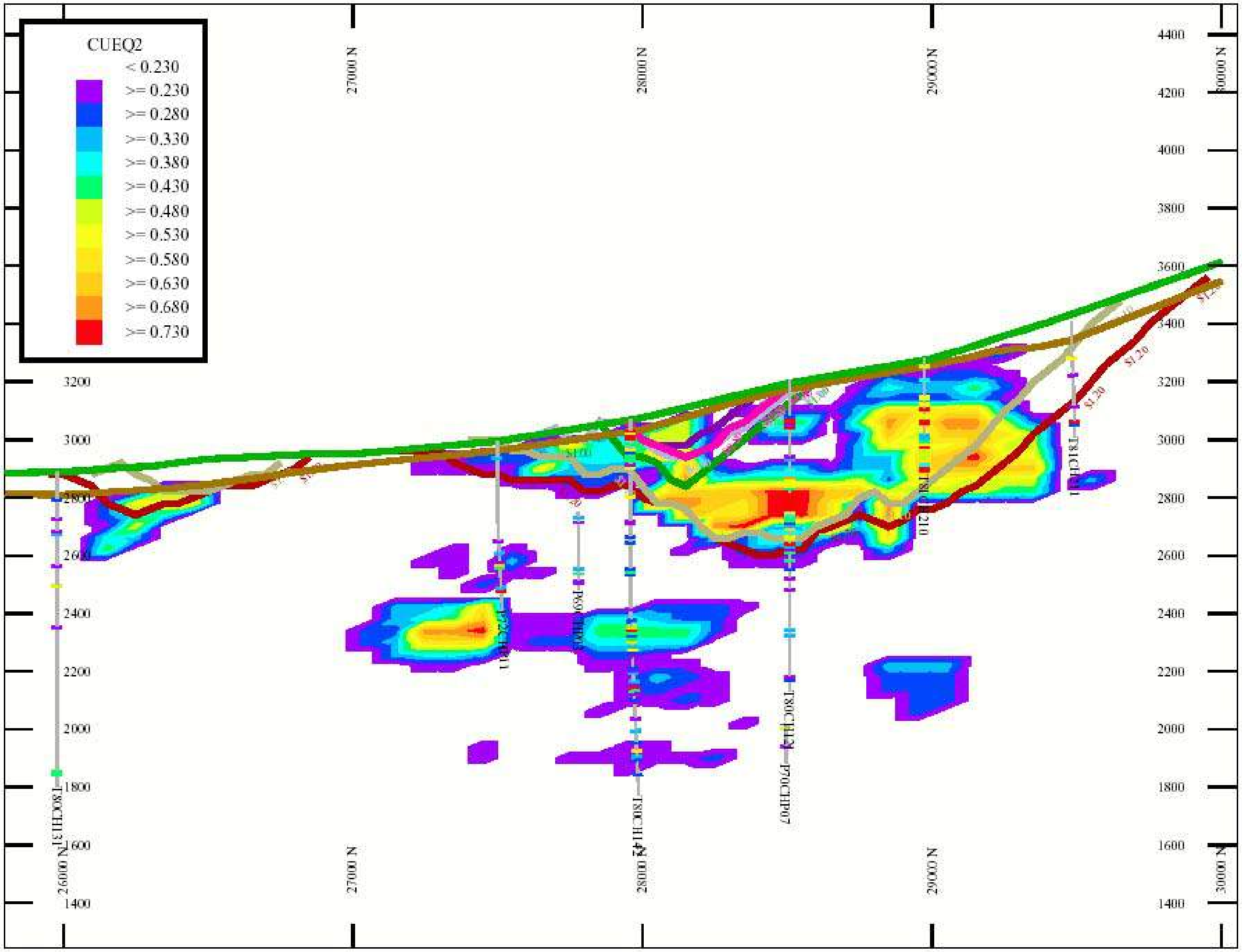
APPENDIX 3
LONG SECTIONS



Long Section at 7300E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Long Section at 7400E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



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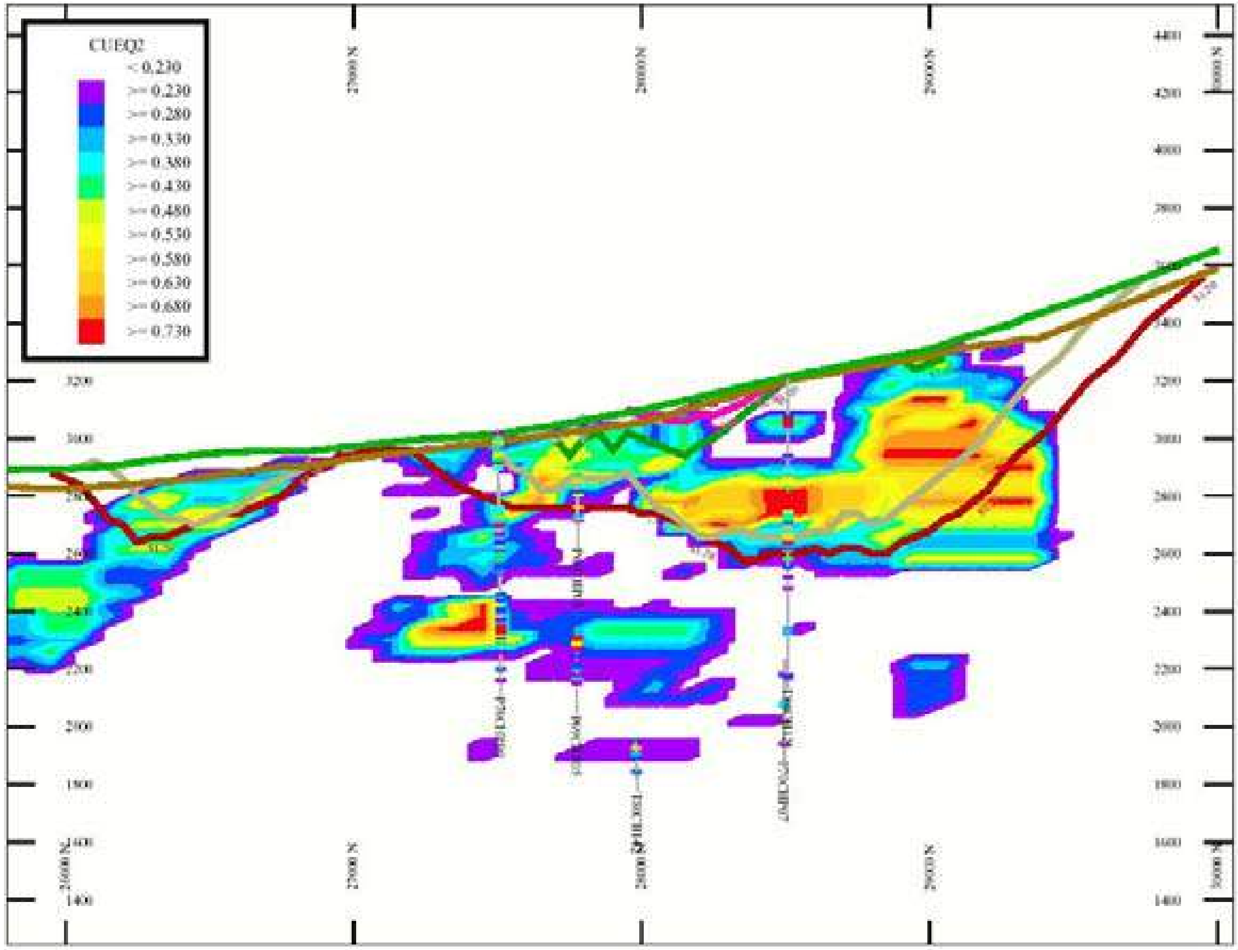

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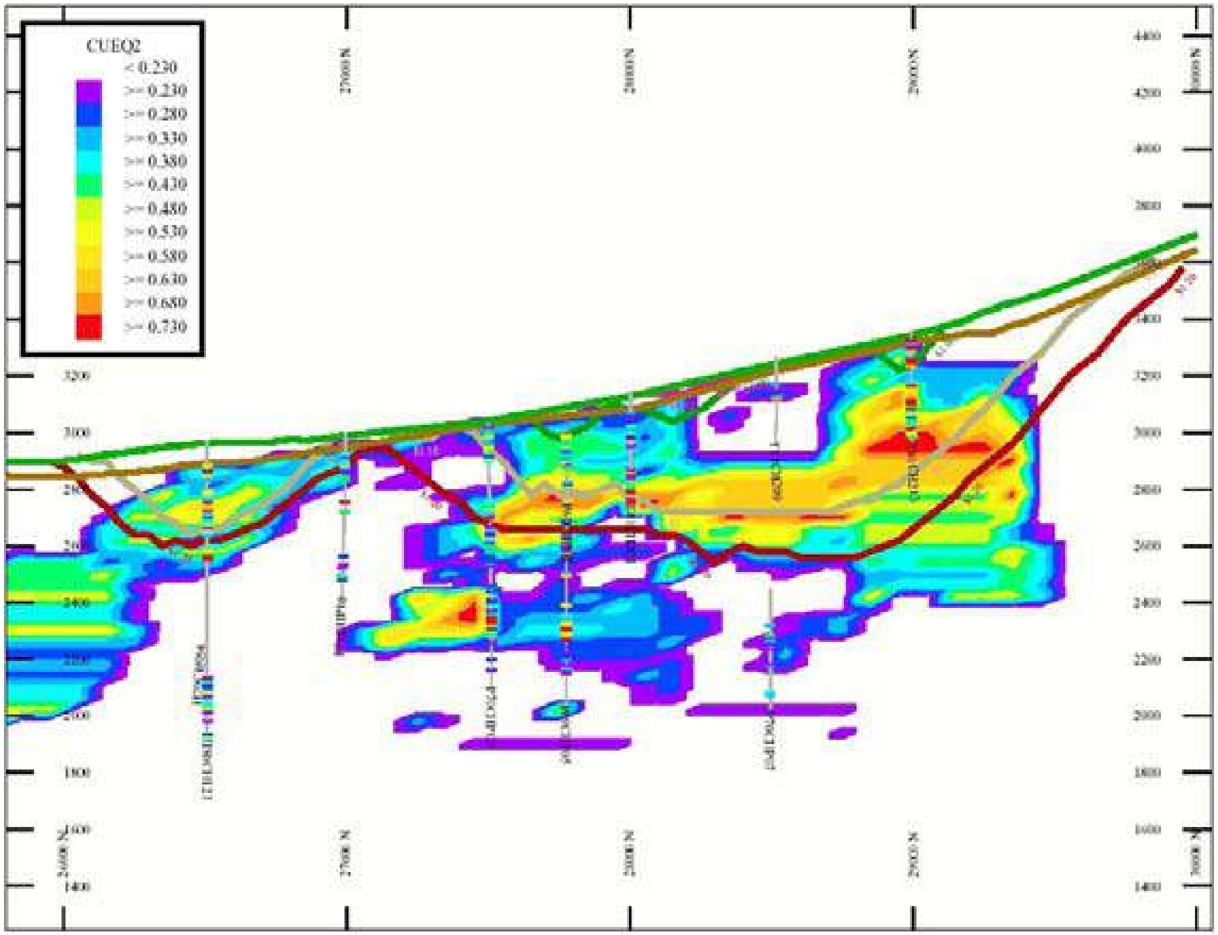
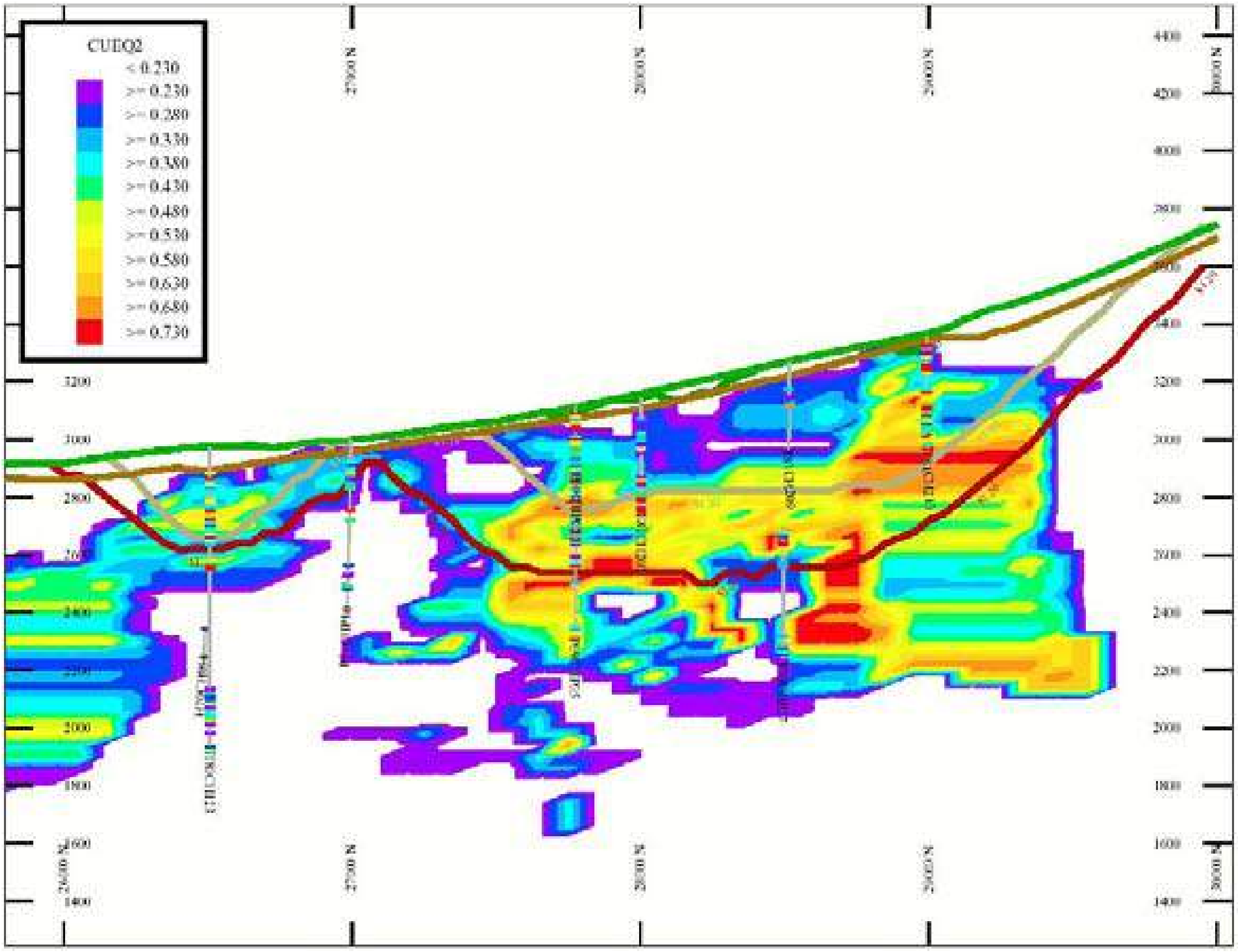
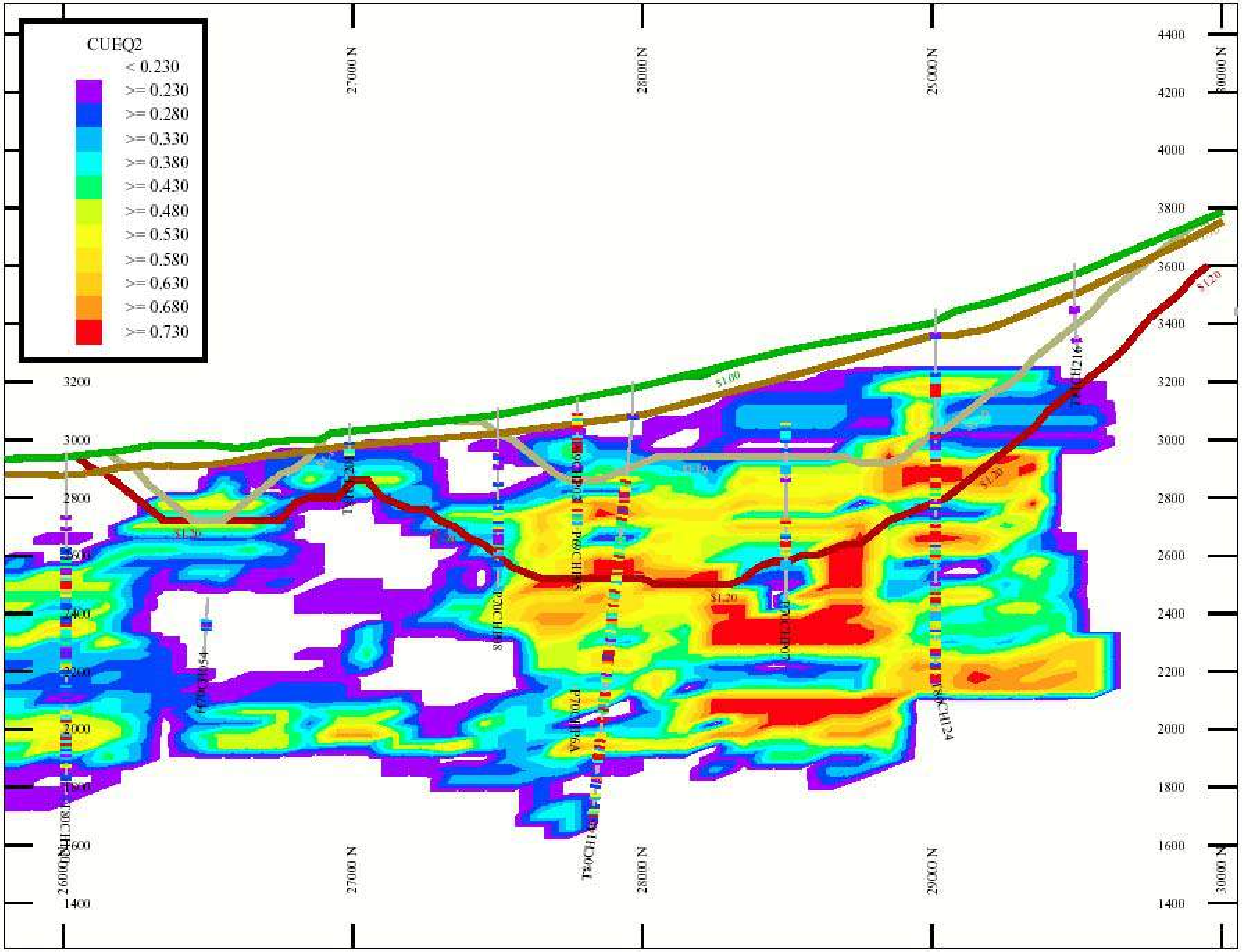


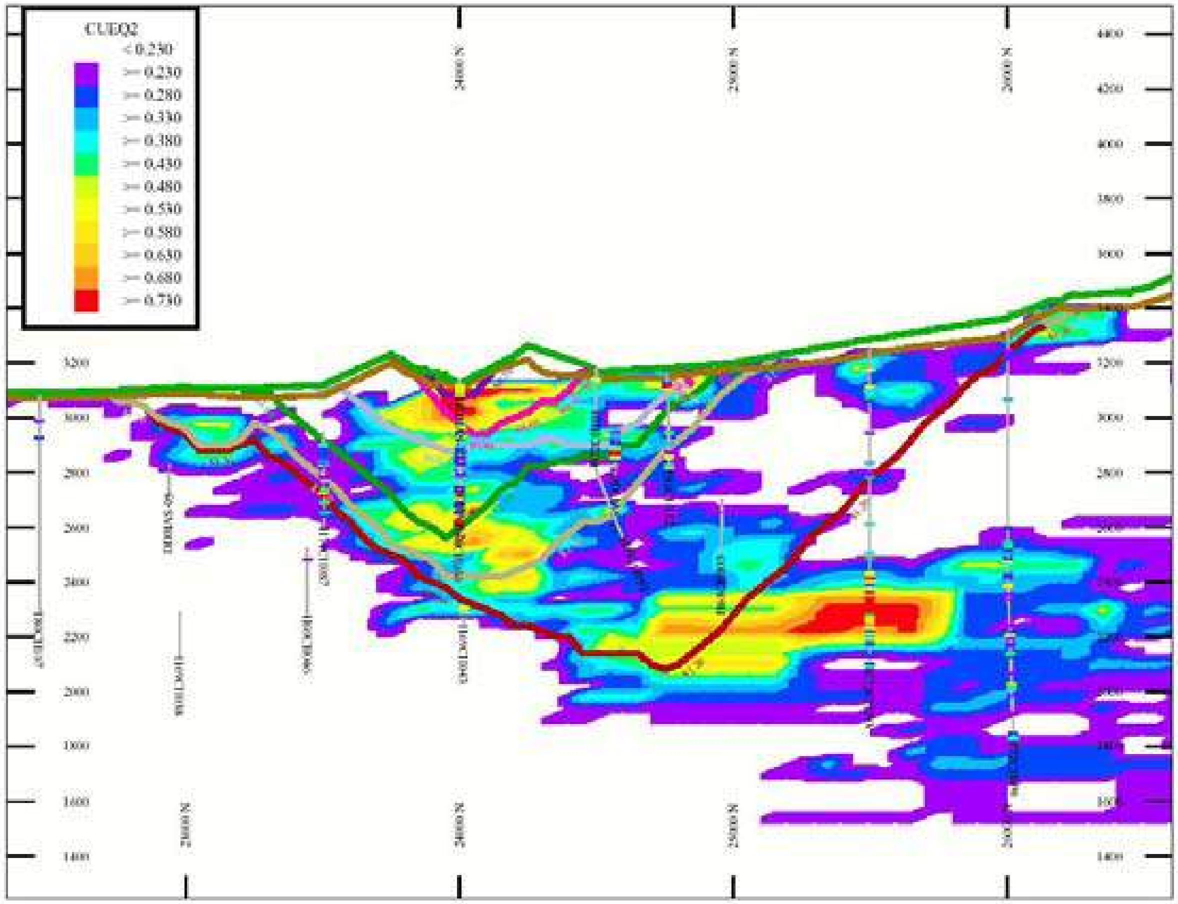
Long Section at 7700E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Long Section at 7800E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Long Section at 7900E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Long Section at 9600E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

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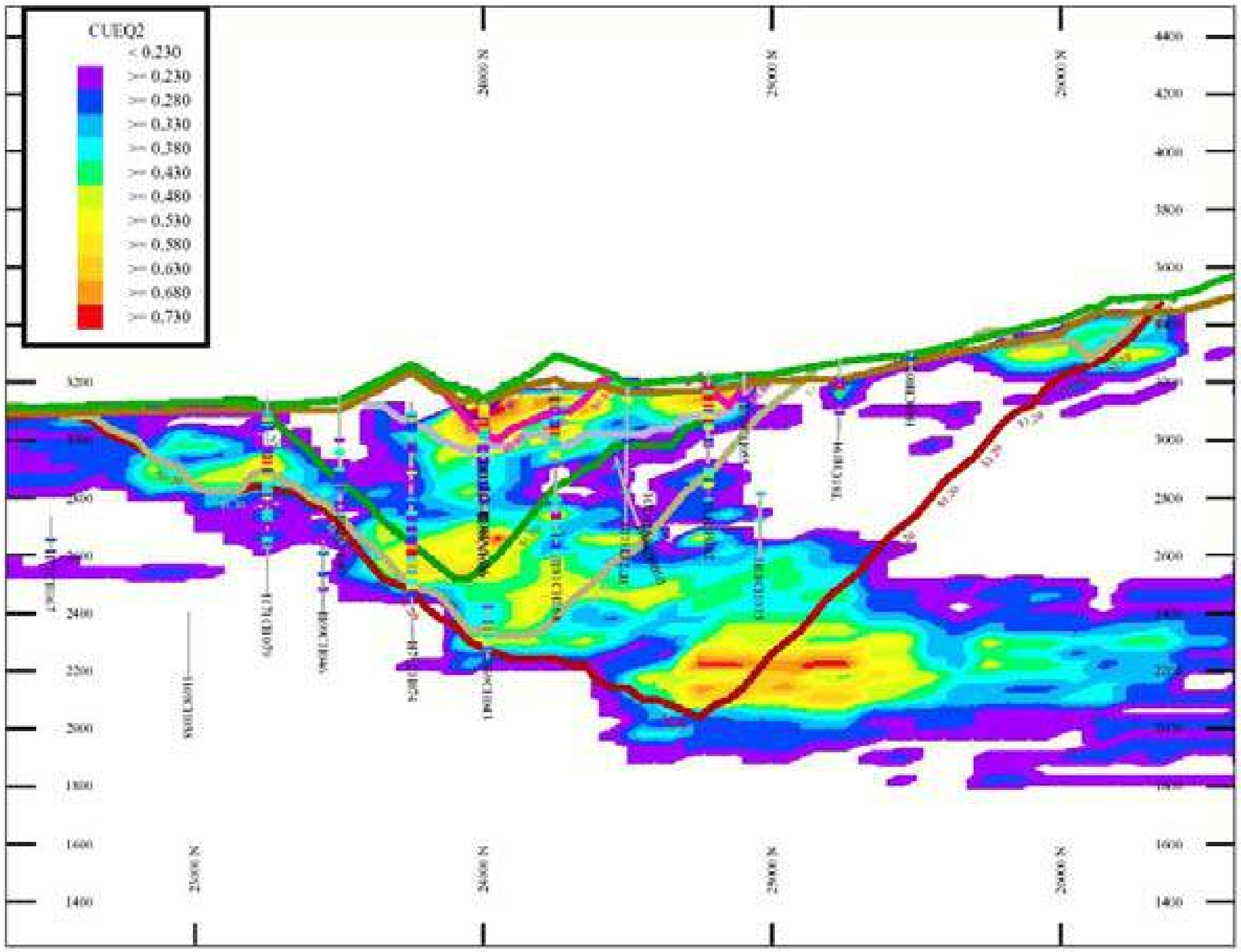
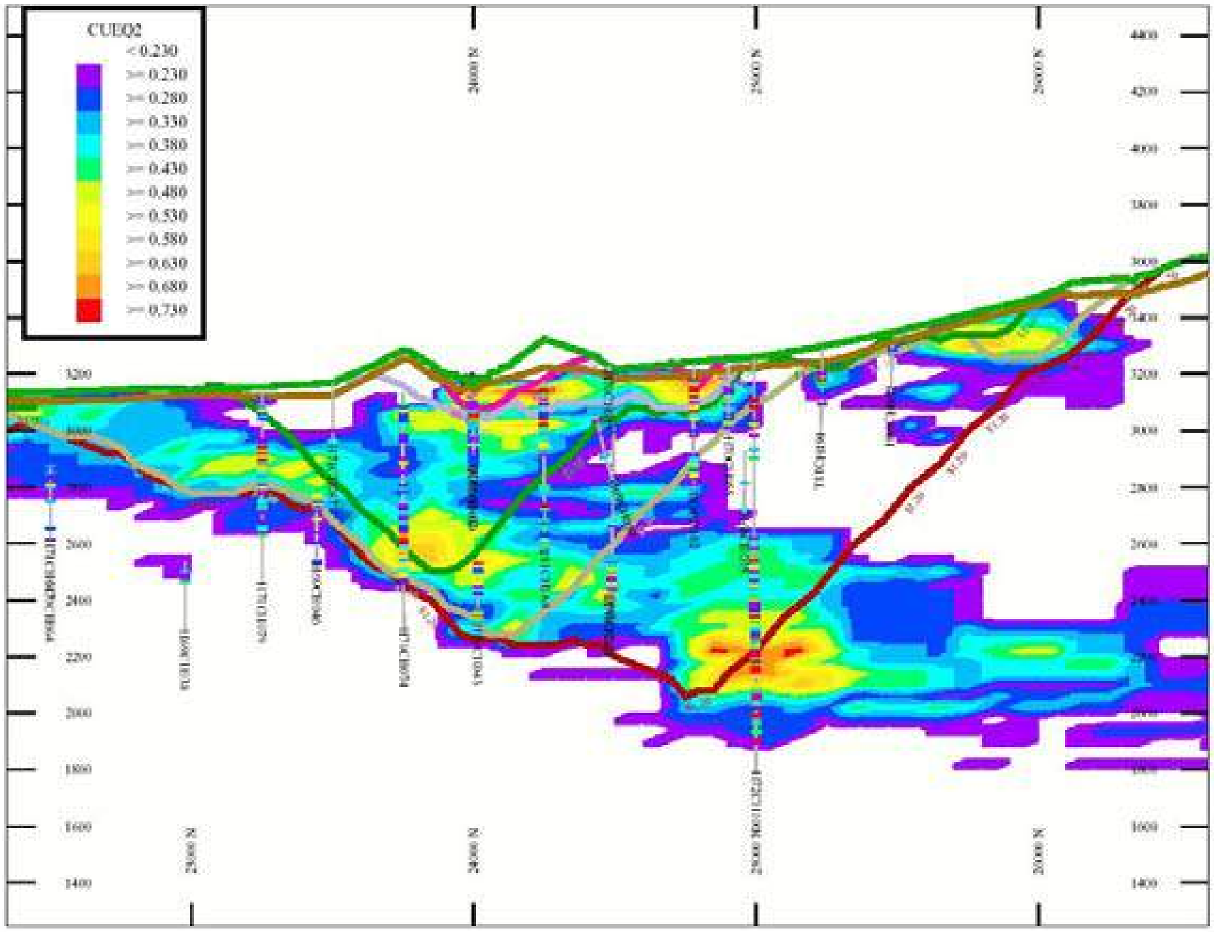
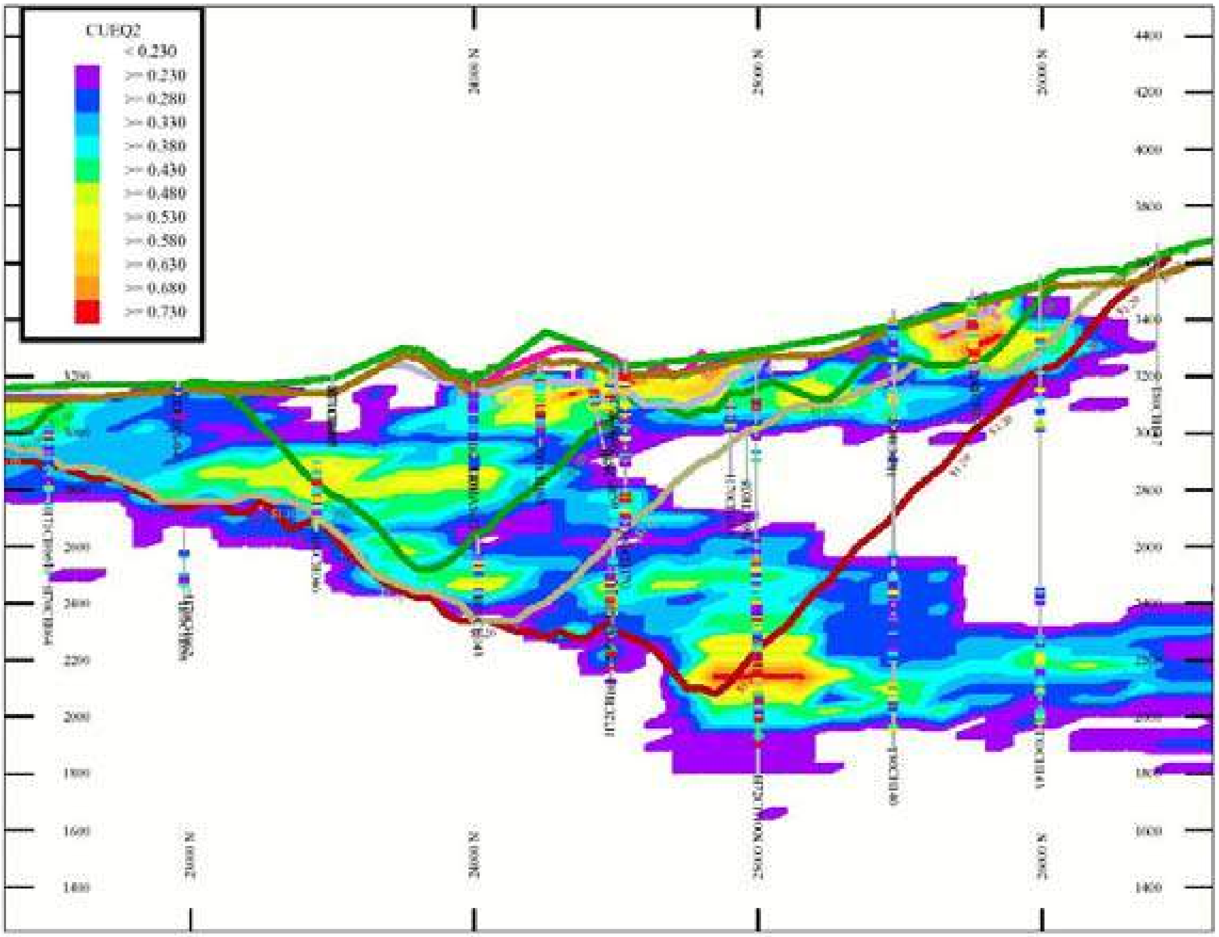
September 20, 2004



Long Section at 9700E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Long Section at 9800E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



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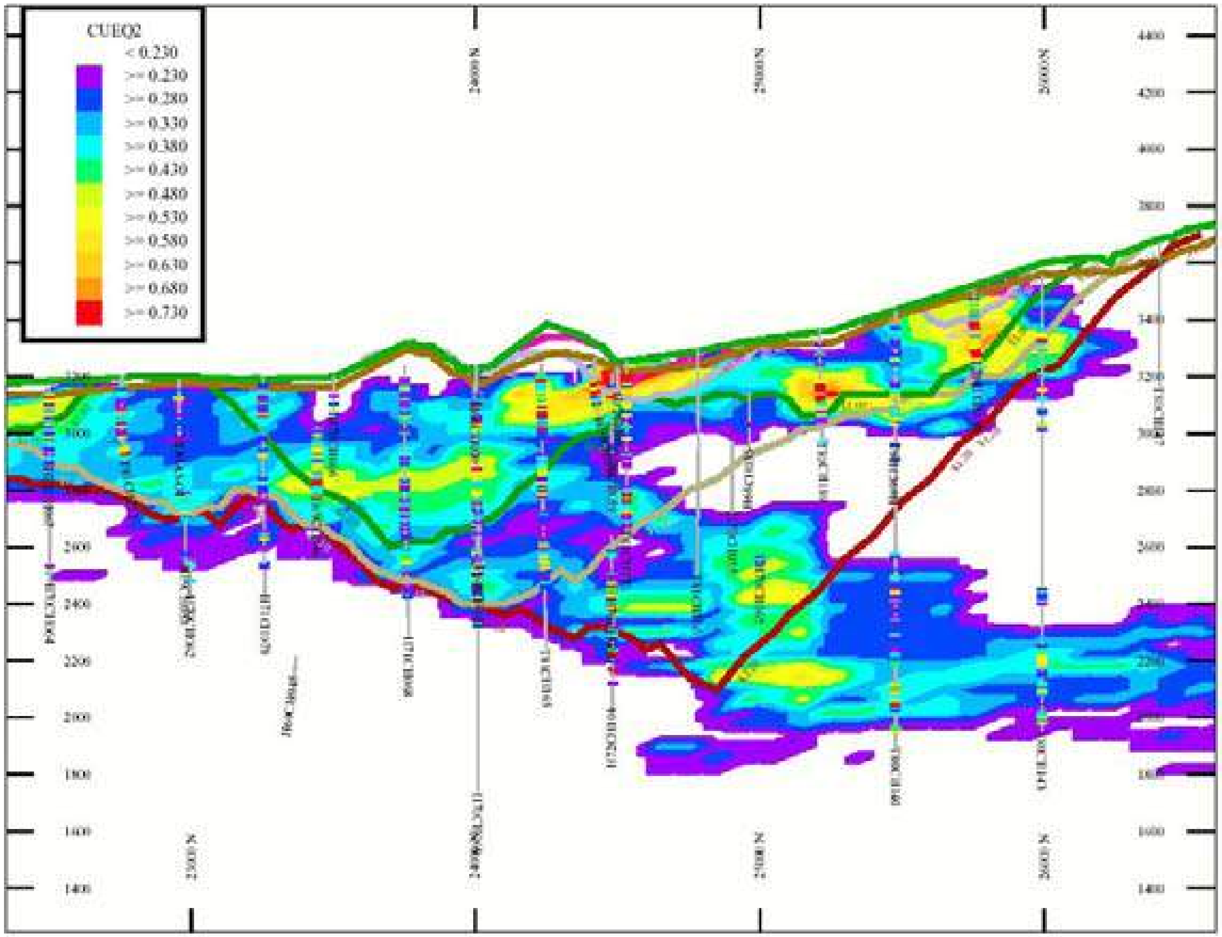

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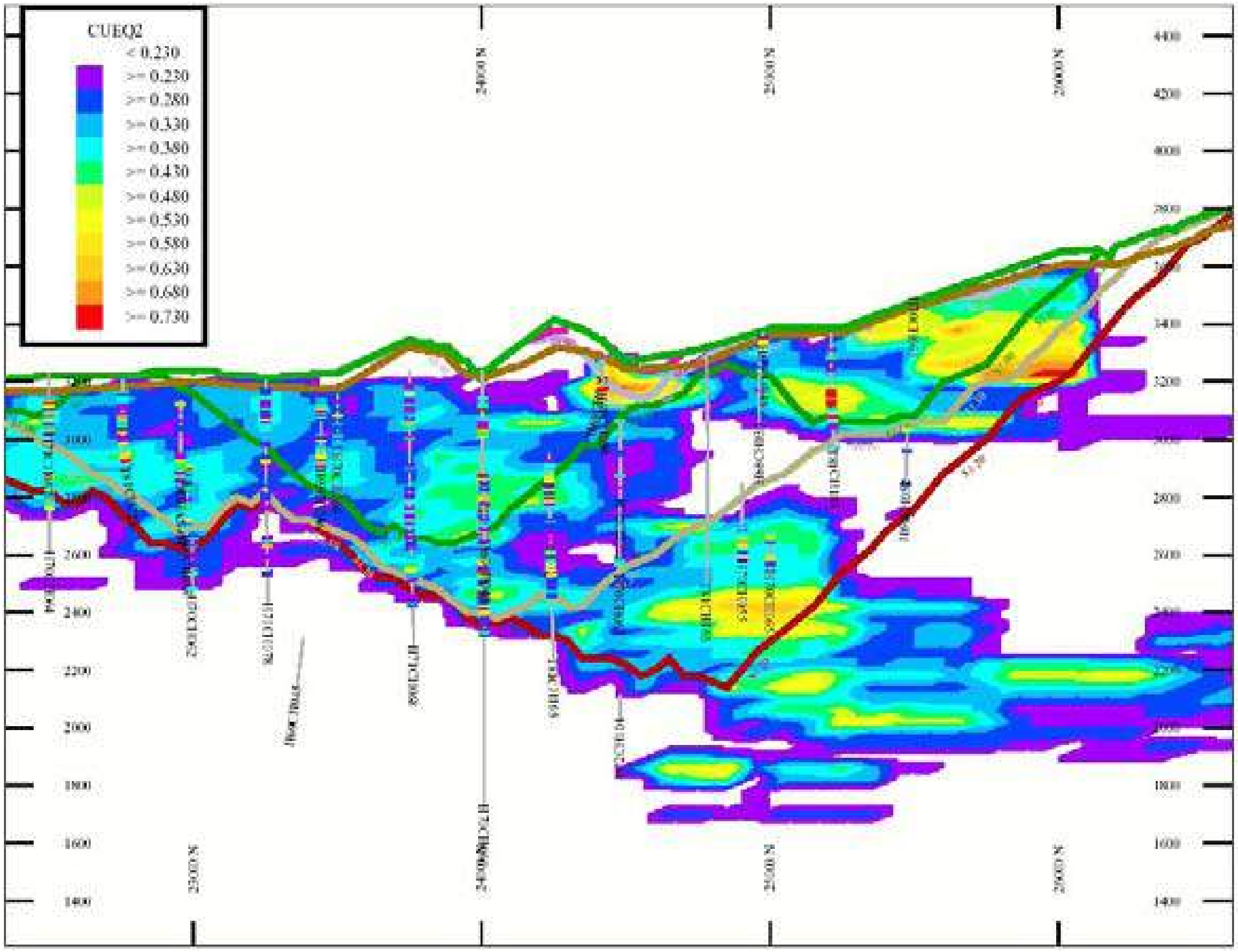

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