Ministry of Energy & Mines

Energy & Minerals Division **ASSESSMENT REPORT** Geological Survey Branch **TITLE PAGE AND SUMMARY**

TITLE OF REPORT: **TOTAL COST: $ 148,785.79**

AUTHOR(S) Keith McCandlish

SIGNATURE(S)

NOTICE OF WORK PERMIT NUMBER/DATE: N/A **YEAR OF WORK**: 2004

STATEMENT OF WORK - EVENT NUMBER: 3218847 DATE: Oct 21 2004

PROPERTY NAME: SCHAFT CREEK

CLAIM NAME(S) (on which work was done): BB 1-10, BB12, BB21-30, BB 80 Fr, Dave 1FR-9FR, Dave 12 FR-14FR, EMU 1FR-8Fr, p 1-3, P 5-13, P15-53, P 57FR-63FR, Pit 1-80, Rum 1FR, Rum 7, Rum 9, Sue 2, Sue 4, Von 1-45, Bird 1-4, Bud1-6, bud 13-18, Bud 25-34, Gav 1-9, Gav 16, Gav 17, Gav 21-23, Gav56, Gav 58, Gav 60, Gav 62, Gav 66, Gav 67FR, Gav 58FR, ID 1FR-7FR, ID 9FR-11FR, Nov 1-8, Sno 1-16, Hill, Schaft, Side hill and Swamp

COMMODITIES SOUGHT: COPPER, GOLD, MOLYBDENUM, SILVER, RHENIUM
MINERAL INVENTORY MINFILE NUMBER(S), IF KNOWN: 104G 015
MINING DIVISION: LIARD **NTS:** 104G/035, 104G/036, 104G/046
LATITUDE 57° 40= North **LONGITUDE** 131° 57= West

OWNER(S): 1) TeckCominco Ltd. 2) Liard Copper Mines Ltd.

3) 955528 Alberta Ltd. is earning its option to acquire TeckCominco's share in the project.

MAILING ADDRESS: Suite 700, 200 Burrard Street; Vancouver, B.C. V6C-3L9

OPERATOR(S): 1) Guillermo Salazar 2) 955528 Alberta Ltd.
MAILING ADDRESS: Suite 650, 340 – 12th Avenue SW, Calgary, Ab T2R-1L5

PROPERTY GEOLOGY KEYWORDS: Developed Prospect. Porphyry Copper-Molybdenum-Gold deposit. Intermontane Belt of northern B.C. Stikine Terrane. Biotite alteration has been dated at 182 Ma +/- 5 Ma (CIMM, 1975). The reports appendixed to this report describe 43-101 compliant metallurgical tests on old core, check re-assaying of old core definition of pit optimization parameters using the resources defined in the Gyroux-Ostensoe report (already on file with the BC Government) for the purpose of defining the areas from which a bulk test will be collected.

REFERENCES TO PREVIOUS ASSESSMENT WORK AND ASSESSMENT REPORT NUMBERS

588, 589, 900, 4587, 4842, 5210, 5302, 6939, 7669, 8832 & 9920

TYPE OF WORK IN THIS REPORT: preliminary metallurgical re-assessment of drilled off resources & pit optimization plans.

PROJECT COSTS APPORTIONED (incl. support):
GEOLOGICAL (scale, area)
Ground, mapping
Photo interpretation
GEOPHYSICAL (line-kilometres)
Ground
Magnetic
Electromagnetic
Induced Polarization
Radiometric
Seismic
Other
Airborne
GEOCHEMICAL
(number of samples analysed for Y)
Soil
Silt
Rock
Other
DRILLING
(total metres; number of holes, size)
Core
Non-core
RELATED TECHNICAL
Sampling/assaying
Petrographic
Mineralographic
Metallurgical: $81,902.81
PROSPECTING (scale, area)
PREPARATORY/PHYSICAL
Line/grid (kilometres)
Topographic/Photogrammetric
(scale, area)
Legal surveys (scale, area)
Road, local access
(kilometres)/trail
Trench (metres)
Underground dev. (metres)

Other: $66,782.98

TOTAL COST: $148,785.79

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Introduction

This report is used as a cover letter to include the 955528 Alberta Ltd.'s expenditures and the summary reports that constitute the work done on the Schaft Creek property up to October 30th, 2004. It complies with the requirements set in Event No. 3218847 dated October 21, 2004.

Summary & Conclusions

The complete reports of the qualifying work carried out by 955528 Alberta Ltd. on the Schaft Creek Property, northern British Columbia, are enclosed with this report. The reports are:

Appendix I: Metallurgical Validation tests on Selected Drill Hole Samples – Schaft Creek Project, B.C. – Process Research Associates Ltd. August 24, 2004.

Appendix II: Valuation Opinion of 955528 Alberta Ltd's Option to acquire the Teck Cominco Limited interest in the Schaft Creek Mineral Deposit, British Columbia, Canada – author: Keith McCandlish, P. Geol, Associated Mining Consultants Ltd. July 20th, 2004.

Appendix III: Preliminary Assessment of the Schaft Creek Deposit, British Columbia; Project Status Report No. 1; by Keith McCandlish, P. Geol., Associated Mining Consultants Ltd., September 20th, 2004.

Each report represents a complete phase of the program carried out during 2004. Appendix III analyses the work done and its results. It recommends a follow up program totalling (Can)$1,338,125, which is endorsed by 955528 Alberta Ltd. The Statement of Expenditures, however, summarizes all expenditures for the period.

Statement of Expenditures

Schaft Creek Project
Statement of Recordable Expenses for Assessment Record Purposes
by 955528 Alberta Ltd. at the Schaft Creek Project
For the year starting from January 1, 2004 to October 15, 2004
Statement prepared by Guillermo Salazar, P. Geol. Q.P.

DATE	DESCRIPTION	Can $
Mobilization/Demobilization		
	Airfares	
12-Jan-2004	Transportation	265.45
1-Feb-2004	Departure tax	10.00
24-Feb-2004	Transportation	265.72
30-Mar-2004	Transportation Vancouver	725.55
30-Mar-2004	Transportation Vancouver	725.55
31-Mar-2004	Transportation Refund	-442.00
31-Mar-2004	Transportation Refund	-341.42
12-Apr-2004	Airport Improvement fee	5.00
12-Apr-2004	Departure tax	15.00
12-Apr-2004	Departure tax	15.00
12-Apr-2004	Charge for changing reservation	26.75
12-Apr-2004	Trans. Van-Smithers	452.70
20-Apr-2004	Transportation	703.08
21-Apr-2004	Transportation Tax	10.00
8-May-2004	Transportation Tax	10.00
8-May-2004	Transportation	539.50
8-Jun-2004	Flight to Vancouver	815.78
	TOTALS	**3,801.66**
	Truck Rental & Transportation	
12-Apr-2004	Gas	27.01
12-Apr-2004	Gas	31.41
18-Apr-2004	Gas	32.00
19-Apr-2004	Gas	25.14
20-Apr-2004	Gas	16.82
20-Apr-2004	Rental Field	1,068.63
4-May-2004	Transportation	312.17
6-May-2004	Parking	4.00
7-May-2004	Parking	6.37
8-May-2004	Parking	33.17
8-May-2004	Gas	33.30
5-Jun-2004	Rental	80.86
	TOTALS	**1,670.88**
	Hotel, Meals & Sundry Office Supplies	

	Meals	
2-Jan-2004	White Spot	$21.36
1-Feb-2004	Toast Café	$10.19
1-Apr-2004	Blackstone Grill	$32.37
5-Apr-2004	Candia Taverna	$102.18
12-Apr-2004	Hudson Bay Lodge	$11.43
12-Apr-2004	Petro Canada	$12.60
12-Apr-2004	Coffee Shop	$19.26
12-Apr-2004	Hudson Bay Lodge	$21.29
12-Apr-2004	Bulkley Valley Wholesale	$143.29
18-Apr-2004	Gordo's Café	$37.24
18-Apr-2004	SuperValu	$55.83
18-Apr-2004	Bear Paw Resort	$1,120.81
19-Apr-2004	Bear Paw Resort	$24.61
19-Apr-2004	Bell 2 Lodge	$67.59
20-Apr-2004	Smithers Town Pantry	$7.25
20-Apr-2004	Hudson Bay Lodge	$23.05
6-May-2004	Feenies	$86.40
7-May-2004	White Spot	$76.01
	TOTALS	**$1,872.76**

	Hotels	
12-Apr-2004	Hudson Bay Lodge	$95.65
18-Apr-2004	Bear Paw Resort	$1,737.00
19-Apr-2004	Bear Paw Resort	$178.70
20-Apr-2004	Hudson Bay Lodge	$256.83
20-Apr-2004	Hudson Bay Lodge	$256.83
	TOTALS	**$2,525.01**

	Field & Office Supplies	
17-Jan-2004	Photocopying	$86.11
7-Apr-2004	core boxes	$618.30
9-Apr-2004	Photographic equipment	$117.67
11-Apr-2004	Field supplies	$82.17
11-Apr-2004	Office	$92.74
12-Apr-2004	Telephone	$9.11
12-Apr-2004	Field supplies	$8.55
12-Apr-2004	Photocopying	$8.59
12-Apr-2004	Field supplies	$820.31
14-Apr-2004	Field supplies	$36.56
20-Apr-2004	Telephone	$6.43
20-Apr-2004	Telephone	$6.43
20-Apr-2004	Courier - boxes of core to YVR	$1,157.57
24-Apr-2004	Dry cleaning	$23.09
3-May-2004	Field supplies	$15.39
5-May-2004	Supplies	$24.02
10-Jun-2004	Maps	$261.62
10-Jun-2004	Computer software	$321.00
22-Jun-2004	Maps	$370.22
20-Jul-2004	Computer Program	$267.45

21-Jul-2004	Set-up of network	$1,016.50
23-Jul-2004	Set-up of network	$251.45
18-Nov-2003	Dues	$10.70
	TOTALS	**$5,611.98**
	Personnel	
	Name &/or purpose	
5-Apr-2004	Associated Mining Consultants - Scoping Study	$42,800.00
18-Apr-2004	Contractors – Field Crews Work done on property	$3,000.00
31-May-2004	Erik A. Ostensoe - Core handling	$412.50
28-Feb-2004	GSS & Assoc - Geological Consulting Feb 23..28: 5 days	$2,889.00
28-Feb-2004	GSS & Assoc - Geological Consulting Feb 5..15: 11 days	$5,296.50
31-Mar-2004	GSS & Assoc - Geological Consulting Mar 22..23: 2 days	$963.00
31-Mar-2004	GSS & Assoc - Geological Consulting Mar 1..5: 5 days	$2,407.50
30-Apr-2004	GSS & Assoc - Geological Consulting april 8..26: 19 days	$9,148.50
31-May-2004	GSS & Assoc - Geological Consulting May 10..21: 12 days	$5,778.00
30-Jun-2004	GSS & Assoc - Geological Consulting June 14..16: 3 days	$1,444.50
28-Apr-2004	Highland Helicopters Ltd. - Shaft Creek-Flow Through	$18,351.57
30-Apr-2004	Highland Helicopters Ltd. - Shaft Creek-Flow Through	$124.01
11-May-2004	Highland Helicopters Ltd. - Shaft Creek-Flow Through	$164.08
5-Apr-2004	Process Research Associates Ltd. - Shaft Creek Samples	$8,000.00
27-Apr-2004	Process Research Associates Ltd. - Shaft Creek	$560.00
29-Jul-2004	Process Research Associates Ltd. - Samples Met	$7,769.54
Sep 30 2004	G. Salazar & Associates Ltd - Sep 5-11 & 13-16; 12.2 days	$5,874.30
2004 Oct 15	G. Salazar & Associates Ltd - Oct 1-10; 10 days	$4,815.00
	TOTALS	**$119,798.00**
	SUB TOTAL	**$135,280.29**
	Miscellaneous Charges	**$13,425.50**
	GRAND TOTAL	**$148,705.79**

Certificate of Author

I, Keith McCandlish, 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. McCandlish 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. McCandlish 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. Am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

Dated this 15 Day of January, 2005 at Calgary, Alberta, CANADA

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

Appendix I: Metallurgical Validation tests on Selected Drill Hole Samples - Schaft Creek Project, B.C. - Process Research Associates Ltd. August 24, 2004.

METALLURGICAL VALIDATION TESTS ON SELECTED DRILL HOLE CORE SAMPLES SCHAFT CREEK PROJECT, B.C.

Prepared for: **955528 ALBERTA LTD.**
23 Brabourne Mews S.W.
Calgary, AB
T2W 2V9

Attention: **Mr. Guillermo Salazar**

Prepared by: **PROCESS RESEARCH ASSOCIATES LTD.**
9145 Shaughnessy Street
Vancouver, B.C.
V6P 6R9
Canada

PRA Project No.: **0402903**

Report by:
Gie Tan, Ph.D.
Senior Metallurgist

Reviewed by:
André de Ruijter, M.Sc.
Senior Metallurgist

Date: August 25, 2004

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1.0 SUMMARY

Metallurgical test work was undertaken on selected drill core samples from the historical drill core inventory of the porphyry Schaft Creek Project in northwestern BC. The test work objectives were:

- to assess the suitability of the drill core samples for process development purposes
- to confirm and extend the analytical data base for 16 selected drill core samples
- to assist in planning the pre-feasibility study work required for treating the different mineral zones using the existing drill core inventory.

Drill core samples of specific intervals were selected for test work by the client, Mr. G. Salazar. The selection was based on several criteria that included:

- age and location of the drill holes
- availability of historical test data in the immediate vicinity of the selected drill holes
- drill hole depth
- visual observation of the degree of oxidation
- lithology and grade of the 10 foot interval drill hole cores
- general strategy developed by the client for actual exploitation.

The head assay results obtained in this study showed acceptable correspondence with the historical data for the main elements of interest, namely Cu, Mo, Au and Ag, to the extent that a reliable assignment of the higher and lower grade samples could be made. In addition, the total sulphur content was compared to the sulphide sulphur grade and used as an indicator of the degree of oxidation of the samples. Only sample H61 showed a significant difference of more than ~20% and was selected for test work as representing possibly the most oxidized material.

A comparison of the assay results obtained is summarized in the table below.

Sample Assay Validation Results

HOLE #	Current Test Results				Historical Database			
Starting Level, ft	**Cu, %**	**Mo, %**	**Au, g/t**	**Ag, g/t**	**Cu, %**	**Mo, %**	**Au, g/t**	**Ag, g/t**
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
AVERAGE	0.36	0.017	0.24	1.42	0.37	0.034	0.24	2.31

The assays obtained during this study were confirmed by independent checks as well as metallurgical test work on selected samples. While the correlation of Cu and Au assays with historic data is excellent, the current Mo and Ag assay values are significantly lower. This could possibly be as a result of weathering, mineralogical variations in the ore body, or systematic differences in sample preparation and assaying techniques.

The results of baseline flotation tests were encouraging considering the low head grades although the mass recovery was relatively high. Recoveries for Au varied from 70 to 89%, for Ag from 90 to 99%, and for Cu from 91 to 96%. The recovery for Mo varied from 34 to 85% and depended on the head grade of the sample. These results are shown in the table below.

Baseline Rougher-Scavenger Flotation Results

Hole #	P_{80}	Mass	Ro.-Sc. Product Grade				Recoveries			
	μm	%	Cu %	Mo %	Au g/t	Ag g/t	Cu %	Mo %	Au %	Ag %
H61	105	24.9	1.24	0.030	0.45	-	91.1	53.0	74.7	-
T186	73	23.2	1.57	0.070	1.05	8.4	93.5	68.0	88.8	89.5
T182	72	25.1	1.34	0.167	0.68	4.3	93.5	84.9	88.3	99.3
T176	74	20.5	2.85	0.024	1.80	5.1	95.8	37.8	92.1	99.2
T172	76	19.5	1.93	0.066	0.65	11.4	95.1	61.6	88.8	93.3
H61	76	28.2	1.10	0.027	0.36	1.8	92.3	51.3	70.3	98.6

The assay grades of the tailings hovered near the detection limits, namely 0.02 to 0.06g/t Au, <0.1 to 0.3g/t Ag, 0.02 to 0.04% Cu, and <0.001 to 0.005% Mo. Based on these tests, it seems likely that a unified flow sheet may apply to the various ore types in the deposit.

A preliminary single cleaner flotation test was conducted on sample T186. This recovered 64% Au, 48% Ag, 72% Cu and 49% Mo into a cleaner concentrate product with a grade of 23g/t Au, 240g/t Ag, 30% Cu and 1.2% Mo. These results could probably be improved upon by optimization of the flotation process. Similarly, the recovery of molybdenum would be addressed once the process parameters have been optimized.

Because of the modest sulphide contents of the samples, acid generating problems are not anticipated. Acid-Base Accounting (ABA) results showed that the Net Neutralization Potentials (NNP) by the modified Sobek method averaged ~62kg/t of $CaCO_3$ equivalents in the samples. The acid generating potential (AP) is low and the (NNP/AP) ratio is greater than 6 indicating that the potential for acid generation is minimal. Interactively Coupled Plasma (ICP) scans further identified the absence of potentially hazardous elements. The reduction of total sulphur in the tailings after flotation to below 0.03% further reduces concerns regarding acid generation. An alkaline flotation procedure would further increase this excess neutralization capacity.

The existing core samples tested yielded positive and encouraging results. Despite their age and unmonitored storage conditions, the samples responded well to generic flotation procedures. The integrity of the assay database was substantially confirmed and enhanced with new assay data.

It is concluded that the remaining historical core samples can be used to prepare composites for further testing and development of a process flowsheet Additional new samples generated during the coming exploration program should also be tested to complete the metallurgical evaluation of the deposit.

2.0 INTRODUCTION

Process Research Associates Ltd. (PRA) was requested by Mr. G. Salazar to validate the historical drill core samples from the Schaft Creek project in northwestern B.C. These drill cores were produced by Hecla Mining Company and Teck Corporation during the mid-1970s and early 1980s respectively. The main objectives were to confirm the head assays of selected samples against the existing information database and to test selected samples for the recovery gold, silver, copper and molybdenum by bulk flotation. Once validated, the historical drill core samples could be then be used in a new test program to characterize the project. Newly drilled samples generated by additional drilling would be used to confirm the results obtained.

The client provided detailed instructions for the sampling of the drill cores and the test work program. This defined the scope, objectives and logistics of conducting the validation test program.

All the tests were conducted using standard procedures that are summarized in Section 3. The results are presented and discussed in Section 4, and conclusions are drawn in Section 5. Detailed test results are provided in the Appendix I to III.

3.0 PROCEDURES

3.1 SAMPLE PREPARATION AND CHARACTERIZATION

Core samples from the historical collection of drill holes taken for the Schaft Project arrived at PRA on April 23, 2004. The samples were in the form of 10-foot interval splits contained in drill core boxes. Based on the documented lithology, grade, age, the location of the original holes and the core intervals, a suite of 16 samples was selected by the client as being representative samples for purposes of validation of the drill cores.

Each selected core interval was located and put into a labeled plastic bag, the sample was logged, then crushed to 10 mesh Tyler (P_{90} ~1.7mm), blended and split by riffling into appropriate test charges. Representative sub-samples were pulverized for head assay and ABA analyses prior to blending two core intervals each of 5 selected drill holes for comparative flotation tests. The 5 selected blends represented drill holes H61, T182, T186, T172 and T176 respectively.

The client selected a few specimens for mineralogical examination. This work was performed elsewhere and the results are not part of this report. After completion of the test work, an additional 91 core intervals were selected by the client and placed into labeled sample bags.

3.2 GRINDING AND SCREENING

Grinding was performed for a specific timed period in a stainless steel laboratory rod mill using 2kg samples at a 65% by weight solids content. Test grinds were conducted on the 5 drill hole sample blends to produce the desired fineness for flotation testing.

A particle size analysis was performed for each sample using a Rotap™ equipped with 20 cm (8") diameter test sieves, stacked in ascending mesh sizes. The sample was initially wet screened at 37 microns (400 Tyler™ mesh). The +37 micron fraction was then dry screened through the stacked sieves. Each sieved fraction was collected, weighed and the individual and cumulative percent retained then calculated.

3.3 ANALYTICAL DETERMINATIONS

The Au and Ag analyses were done by standard fire assay procedures with an Atomic Absorption (AA) finish. The samples were also submitted for multi-element ICP scans at different sensitivity levels, as well as for quantitative sulphur and metal species determinations.

Special metals of interest were determined by ICP or AA spectroscopy, following complete dissolution in multiple acids. Total sulphur was measured using a Leco furnace and sulphide sulphur assays were based on a wet chemical gravimetric procedure.

3.4 FLOTATION TESTS

Standard batch kinetic bulk rougher and scavenger flotation techniques applied to recover the sulfides by kinetic bulk rougher and scavenger flotation techniques. Lime and collectors Aero 3302 and sodium ethyl xanthate (SEX) were added to the grinding step.

The baseline tests were conducted using a Denver D12 laboratory flotation machine with 2 kg sample batches in a 4L cell to yield a typical pulp density of ~35% solids by weight. The solids were pulped in Vancouver municipal water at an ambient temperature of ~18°C. The impeller speed was set at the required rate according to cell size and the airflow was controlled manually to maintain the froth level. One larger scale bulk rougher cleaning flotation test using a 4kg charge in a larger cell was also conducted, using similar conditions.

3.5 ACID BASE ACCOUNTING

The acid potential (AP) of samples was calculated from the sulphide sulphur analyses and the NNP was determined using the modified Sobek method. The results are in Appendix VI.

4.0 RESULTS AND DISCUSSION

The samples were delivered in about 165 core boxes. These were labeled by drill hole number with 10 feet intervals (see Appendix I for details). For the validation work, 16 intervals were selected as identified in Table 4.1 with their corresponding sample numbers:

Table 4.1 – Selected Validation Samples

Sample ID	Hole No.	Interval, ft	Lithology	Received, kg
M587918	H44	750-760	ANDS	6.43
M587919	H61	770-780	ANDS	6.62
M587920	H61	780-790	ANDS	5.20
M587921	T140	860-870	ANDS	5.83
M587922	T182	770-780	ANPF	7.80
M587923	T182	800-810	ANPF	5.33
M587924	T183	540-550	ANLP	6.36
M587925	T186	650-660	BRVL	7.95
M587926	T186	660-670	BRVL	8.68
M587927	T188	300-310	ANPF	7.67
M587928	T188	320-330	ANPF	7.88
M587929	T172	870-880	ANCG	7.20
M587930	T172	880-890	ANCG	6.63
M587931	T176	530-540	PPFQ	7.45
M587932	T176	490-500	PPFQ	6.72
M587933	T125	350-360	ANDS	11.2

Some of the archived core box samples included some overburden intervals but most of the split cores were dry, competent, brownish gray rock. The client selected a few specimens for mineralogical examination elsewhere. Test work was conducted on the 5 selected blends shown in the table.

4.1 SAMPLE CHARACTERISTICS

Head assay results of the primary elements of interest were determined for the selected samples as mentioned in Section 3.2. The results are in Table 4.2 and additional details are provided in Appendix II.

Table 4.2 – Main Element Head Assays

Sample ID	Au, g/t	Ag, g/t	Cu, %	Mo, %	S=, %	S^T, %
M587918	0.10	<0.1	0.24	0.048	0.23	0.28
M587919	0.18	1.4	0.46	0.010	0.93	1.19
M587920	0.08	0.5	0.10	0.007	0.17	3.16
M587921	0.43	2.0	0.31	0.004	0.11	0.16
M587922	0.23	1.5	0.46	0.051	0.55	0.61
M587923	0.08	0.4	0.15	0.010	0.37	0.42
M587924	0.41	3.1	0.46	0.064	0.31	0.35
M587925	0.30	1.9	0.36	0.017	0.36	0.43
M587926	0.33	2.2	0.39	0.015	0.32	0.38
M587927	0.41	1.6	0.40	0.014	0.46	0.52
M587928	0.33	0.5	0.26	0.005	0.73	0.79
M587929	0.15	2.1	0.31	0.004	0.19	0.23
M587930	0.13	2.5	0.42	0.023	0.30	0.35
M587931	0.61	2.0	0.88	0.004	0.90	0.95
M587932	0.16	0.2	0.24	0.004	0.21	0.24
M587933	0.10	0.7	0.35	0.003	0.72	0.77

All the sulphide sulphur to total sulphur ratios are greater than 0.70 except for sample M587920 from drill hole H61. This was originally considered to be as a result of extensive oxidation due to the weathering of the older samples. However, it is more likely the result of mineralogical variations in the core samples, and/or differences in analytical techniques.

Separate ICP scans at lower detection limits confirmed the general reliability of the head assays and the absence of major penalty element concentrations.

The client also submitted samples for specific elements assayed at lower detection limits which could have potential pay and penalty implications. Since this analytical work was done at a different laboratory, the results are not reported here.

4.2 ASSAY VALIDATION

The head assay results obtained were compared to values from the historical as well as the results from another independent laboratory. The detailed sample assays are provided in the Summary and in Appendix I while the average values are listed in Table 4.3.

Table 4.3 – Head Assay Averaged over 16 Selected Samples

Source of Data	Au, g/t	Ag, g/t	Cu, %	Mo, %	Fe, %	Mn, %	S^T, %
Historical Data	0.24	2.31	0.37	0.034	n.a.	n.a.	n.a.
Current Test	0.24	1.42	0.36	0.017	3.55	0.045	0.677
Other Laboratory	0.23	1.64	0.38	0.017	3.14	0.058	0.578

The averaged values listed in the table above show excellent correlation for Au and Cu. The historical data for Ag and Mo is different from that of this current study and the “Other Laboratory” while the results from this current study and that from the “Other Laboratory” are reasonably close. Similarly, the assays for Fe, Mn and S^T from this present study and from the “Other Laboratory” are also reasonably close.

The historically higher Ag and particularly the Mo results may reflect real mineralogical variations of the core splits, and/or systematic differences in analytical methods with lower detection limits in modern techniques.

The 5 blends of drill hole intervals selected for testing served as a further check on the head assays. Material balances on the flotation tests, and the assay results from the "Other Laboratory" are used to further confirm the individual head assays results. The pertinent results are summarized in Table 4.4, this time comparing the averaged assays from this current investigation as well as the calculated flotation test material balances with the results from the "Other Laboratory".

Table 4.4 – Blend Assay Comparisons

Hole #	Source of Data	Au g/t	Ag g/t	Cu %	Mo %	Fe %	S^T %
H61	Calc. Flotation Head	0.14	0.50	0.29	0.015	3.88	2.11
	Ave.-Current Test	0.13	0.95	0.28	0.009	3.50	2.18
	Other Laboratory	0.10	0.48	0.30	0.007	3.21	2.18
T186	Calc. Flotation Head	0.31	4.17	0.40	0.020	4.68	0.41
	Ave.-Current Test	0.32	4.10	0.38	0.016	4.13	0.41
	Other Laboratory	0.25	2.19	0.39	0.013	3.52	0.32
T182	Calc. Flotation Head	0.22	1.24	0.37	0.041	4.19	0.55
	Ave.-Current Test	0.16	0.95	0.31	0.031	3.84	0.52
	Other Laboratory	0.16	1.18	0.34	0.033	3.35	0.46
T176	Calc. Flotation Head	0.45	1.12	0.60	0.008	3.28	0.61
	Ave.-Current Test	0.39	1.10	0.56	0.004	2.95	0.60
	Other Laboratory	0.38	1.25	0.59	0.003	2.65	0.43
T172	Calc. Flotation Head	0.16	2.37	0.38	0.015	5.56	0.28
	Ave.-Current Test	0.14	2.30	0.36	0.014	5.06	0.29
	Other Laboratory	0.05	2.35	0.38	0.012	4.28	0.22

Again, generally consistent comparisons were obtained, especially for Cu, while Au, Fe and S comparisons are in good agreement given the nature of the data being compared. Ag and Mo again do not give good correlations and reflects the inherent differences uncertainties in the analytical determinations and procedures for the individual elements.

4.3 GRINDING TESTS

Following after the assay validation of the historical, 5 sample blends were selected for metallurgical testing as mentioned previously. It was considered that generic flotation schemes on the selected blends would show if prolonged storage was detrimental to the recovery of the valuable minerals.

The first sample selected for testing was the H61 blend which had showed the greatest sulphide sulphur/total sulphur discrepancy. The scoping test was conducted by grinding a 2kg batch for 22 minutes under standard conditions. This achieved a P_{80} of 106μm. Similar grinding tests were done on the other selected blend samples. The results are shown in Table 4.5 below.

Table 4.5 – Test Grind Results

Hole #	Blend H61	Blend T186	Blend T182	Blend T176	Blend T172
P_{80}, microns	106	104	111	88	119
Time, minutes	22	22	30	21	29

The results are comparative only but show that samples T172 and T182 were much harder relative to samples H61 and T186, while sample T176 was softer. The time required for grinding each sample to a target P_{80} of 75microns for the subsequent flotation tests was estimated from these grinding tests.

4.4 FLOTATION TESTS

Seven open-cycle flotation tests were conducted on the 5 selected drill hole sample blends in order to confirm that the samples were suitable for more extensive testing. Sample H61 was the first sample selected since it represented a worst-case scenario of low head grades, low sulphide sulphur/total sulphur ratio, a relatively coarse grind size of 106 microns and flotation with low reagent dosages.

4.4.1 Flotation Test F1 at P_{80} = 106μm

The flotation test was conducted on sample H61 ground to P_{80} of 106 microns at an initial pH of 10 with lime and collector reagents added to the grind. Bearing in mind the age of the sample and its unknown response to flotation, the rougher-scavenger flotation test was broken up into 8 distinct stages with different reagents added to each stage (see Appendix III for details). The results would assist in the subsequent assessment of the effect of different reagents and dosage levels to the flotation of this material. The results are summarized in Table 4.6 and Figure 4.1.

Table 4.6 – Flotation Test F1 Results for Sample H61

Sequential Stage Number	pH	Reagents Added	Bulk Rougher Grades				Cumulative Recoveries, %				
			Au g/t	Cu %	Mo %	Fe %	Mass	Au	Cu	Mo	Fe
1	10	SEX/A3302	0.81	3.39	0.05	6.75	4.9	26.8	49.3	15.4	9.1
3	9.5	MIBC	0.62	2.21	0.04	5.63	10.3	43.0	67.4	28.8	15.9
6	8.5	SEX/A3302	0.56	1.56	0.04	6.03	19.3	72.7	89.4	49.5	32.0
7	10	Na_2S	0.52	1.45	0.04	5.83	20.9	73.1	89.6	50.5	33.4
8	6	$PAX/CuSO_4/H_2SO_4/A404$	0.45	1.24	0.03	5.63	24.9	74.7	91.1	53.0	38.5

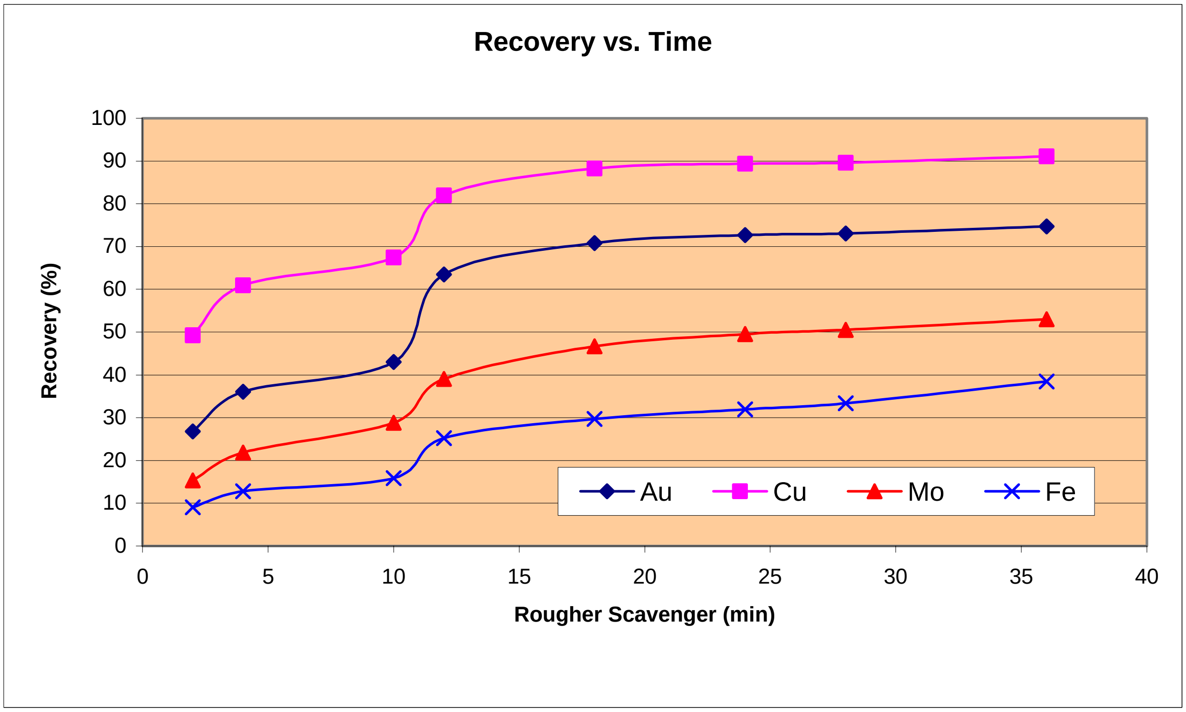

Figure 4.1 – F1 Rougher-Scavenger Kinetics

The results as summarized in Table 4.6 were generally encouraging ultimately yielding a recovery of 75% Au and 91% Cu in 25% of the mass. The high yield in the first scavenger flotation mainly indicated the need for higher reagent dosages from the start, while different reagent additions in the last stages of flotation were less effective. The tailings graded 0.05g/t Au, 0.04% Cu, <0.01% Mo and <0.01% $S^{=}$ which is close to the detection limits for these metals.

Despite the age of the drill core sample and the large discrepancy between the sulphide sulphur and total sulphur values, the sample from this drill core material responded in the normal fashion to the standard flotation procedure and was considered suitable for metallurgical testing.

4.4.2 Flotation Tests at P80 ~ 75μm

Based on the previous flotation scooping test result, the dosage of SEX and A3302 added to the grind was doubled, and additional SEX was provided to the scavenger stages with elimination of all other types of reagents (see Appendix III for details). As a result, the recoveries in the first three rougher stages increased dramatically, but the overall performance remained quite stable as shown in Table 4.7 below.

Table 4.7 – Baseline Flotation Tests at P_{80} ~ 75μm

Test No.	Hole #	P_{80} μm	Bulk Rougher Grades				Total Recoveries, %				
			Au g/t	Ag g/t	Cu %	Mo %	Weight	Au	Ag	Cu	Mo
F2	T186	73	1.05	8.4	1.57	0.070	23.2	88.8	89.5	93.5	68.0
F3	T182	72	0.68	4.3	1.34	0.167	25.1	88.3	99.3	93.5	84.9
F4	T176	74	1.80	5.1	2.85	0.024	20.5	92.1	99.2	95.8	37.8
F5	T172	76	0.65	11.4	1.93	0.066	19.5	88.8	93.3	95.1	61.6
F6	H61	76	0.36	1.8	1.10	0.027	28.2	70.3	98.6	92.3	51.3

With a 19.5% mass pull to the rougher-scavenger concentrate, test F5 using sample from T172 displayed the highest upgrading factors for Cu and Au when compared with the head grade; both are of the order of 5:1.

The recovery of Cu varies from 92 to 95% and silver varies between 90 and 99%, while that for Au is good at between 88 and 92%, except for sample H61 which again gives a relatively low recovery of 70% compared with 75% for the coarser grind (Test F1). The recovery of Mo varies from 38% to 85%. It is probably dependent on the mineralogy and appears to be very sensitive to the head grade. The tailings grades for these tests are also low, hovering near the limits of detection.

4.4.3 Cleaner Flotation Test on Sample T186

One rougher-cleaner flotation test was conducted on medium-grade sample T186 at P_{80} of 75 microns using a 4kg batch size and similar flotation conditions as in Test F2 for the rougher stage. The rougher concentrate was cleaned once prior to re-grinding this concentrate. This was followed by a further 3-stage cleaning step to reject the gangue minerals. The results are summarized in Table 4.8 while Figure 4.2 shows the recoveries of the various metals relative to the Cu grades obtained. The detailed results are given in Appendix III.

Table 4.8 – Cleaner Flotation Test F7 Results for Sample T186

Product	Product Grades				Product Recoveries, %				
	Au g/t	Ag g/t	Cu %	Mo %	Mass	Au	Ag	Cu	Mo
Ro.+Sc.Conc.	1.76	15.9	2.39	0.112	15.7	79.4	59.6	91.7	76.8
1st Cl.Conc.	6.05	48.5	8.14	0.361	4.3	73.9	49.4	84.9	67.2
2nd Cl.Conc.	18.3	136.6	24.8	0.995	1.3	66.4	41.3	76.7	54.9
3rd Cl.Conc.	22.6	158.6	30.1	1.150	1.0	63.5	37.1	72.2	49.2

The mass pull and hence the rougher recovery came in lower than for the Test F2 likely because of the larger batch size and this also affected the flotation kinetics. Test F7 rougher results (Table 4.8 and Figure 4.2), however, were generally comparable to those of Test F2.

The product grade obtained (3rd Cleaner Concentrate) was 30% Cu at an overall recovery of 72% after a relatively mild regrind of the bulk concentrate for 20 minutes in a ceramic pebble mill. The gold recovery for the 30% Cu product grade was 64%, Ag 48% and Mo 49%. Similarly constructed graphs would assist with the commercial evaluation of the project.

The results in Appendix III show that minor values are lost with the scavenger concentrates that were of lower grade than the head sample. Bulk cleaner tails, however, can be scavenged to reduce the losses, which seem to be entrained mainly by the gangue components.

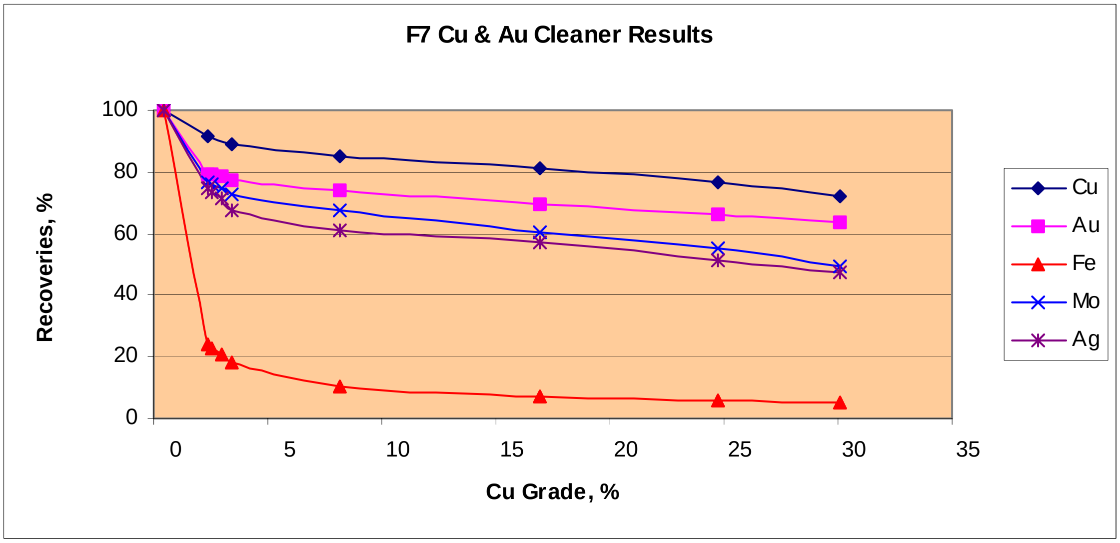

Figure 4.2 – Value Deportment vs. Product Cu Grades in Test F7

4.5 ACID BASE ACCOUNTING RESULTS

The environmental impact of exploiting a largely undisturbed sulphidic ore deposit is mainly defined by the potential for acid rock drainage (ARD) generation. The AP (acid generation potential, expressed as kg of its $CaCO_3$ equivalent per tonne of sample) is commonly estimated by assuming that all of the sulphides contained in the rock can be oxidized to generate sulphuric acid. This amount of generated sulphuric acid reacting with minerals in the rock is measured by titration of the pulverized sample with dilute hydrochloric acid, according to the Sobek acid consumption test. This determination of the neutralization potential (NP) is the approved B.C. method for modified acid-base-accounting (ABA) by US-EPA 600/2-78-054 standard procedures. The results of such measurements on the 16 selected interval samples are given in Appendix VI and is summarized in Table 4.9 below.

Table 4.9 – ABA Summary Results for 16 Selected Samples

Item	$S^{=}$, %	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

In all cases, the (NP-AP) values are positive and the excess NP is at least double the estimated AP. Also, the average of the (NP/AP) ratios exceeds 6 which is the limit adopted for ensuring that the potential for acid generating problems is remote. This is confirmed by the paste pH values that range from neutral to alkaline. Also, lime is added to the flotation process which itself removes nearly all of the sulphides from the tailings. The ABA on these tailings will therefore yield higher positive (NP-AP) results.

5.0 Conclusions and Recommendations

Head assays on 16 samples selected from the existing Schaft Creek drill core inventory were determined to correlate well with the assays from the historical database. This was especially the case with reference to the Cu and Au contents of individual drill hole intervals. The Ag assays values were found to be lower than those in the historical database. However, the Mo values were significantly lower in this validation study. These differences may be attributed to either non-homogeneous distributions of minerals in separate core splits, or differences in analytical methods. Weathering may have affected the archived samples to a small extent.

Flotation test results further confirmed the reliability of the current assay data as well as confirming that the existing sample inventory could be used for process design purposes. All 5 of the selected blends responded well to baseline bulk flotation tests using the PRA standard reagent scheme for porphyry Cu deposits. Even the scoping test on the aged and lower-grade sample H61 gave encouraging recoveries of 75% Au, 91% Cu and 53% Mo at the relatively coarse grind of P_{80} = 106μm. The other samples yielded recoveries of 88 to 92% for Au, 90 to 99% for Ag, 93 to 96% for Cu and 38 to 85% for Mo using a simplified reagent scheme and at a finer grind of P_{80} ~75μm. An open-cycle cleaner flotation test on sample T186 produced a concentrate grade of 30% Cu in 1% of the original mass with recoveries of 63.5% for Au, 72.2% for Cu and 49.2% for Mo. This test was done at a primary grind of P_{80} = 74μm with a 20-minute re-grind of the bulk rougher-cleaner concentrate. The baseline rougher-scavenger tailings assayed between 0.02 to 0.06g/t Au, <0.1 to 0.3g/t Ag, 0.02 to 0.04% Cu, 0.003 to 0.005% Mo and <0.01 to 0.03% sulphide sulphur. The calculated head assays and recoveries are very sensitive to these values which are on the border of the analytical detection limits of the respective elements. It is evident, therefore, that flotation was quite effective in recovering these values from the test samples in a nearly quantitative fashion. However, further tests may require more sensitive analytical methods, or require a higher sample weight for assaying in order to achieve the desired level of accuracy in comparative tests.

The ABA test results on the head samples established an excess neutralizing potential in all cases that would be further augmented by the flotation at alkaline conditions.

Appendix II: Valuation Opinion of 955528 Alberta Ltd's Option to acquire the Teck Cominco Limited interest in the Schaft Creek Mineral Deposit, British Columbia, Canada - author: Keith McCandlish, P. Geol, Associated Mining Consultants Ltd. July 20th, 2004.

Associated Mining Consultants Ltd.

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July 20, 2004

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: Valuation Opinion of 955528 Alberta Ltd.'s Option to Acquire the Teck Cominco Limited Interest in the Schaft Creek Mineral Deposit, British Columbia, Canada

Associated Mining Consultants Ltd. (AMCL) has been retained by 955528 Alberta Ltd. to act as Project Manager for a preliminary assessment (Scoping Study) of the Schaft Creek project's viability. As well, AMCL has been asked to provide an expert opinion on the valuation of the Option Agreement to acquire Teck Cominco Limited's interest in the property. This letter provides a preliminary estimate of the valuation.

Introduction

The Schaft Creek copper-gold-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 Schaft Creek mineral deposit is currently owned by Teck Cominco Limited. Teck Cominco holds a 70 % direct participating interest (the "Direct Holding") in the property as well as 23.4 % carried interest (the "Indirect Holding") through its' 78 % shareholding in Liard Copper Mines Ltd.. Liard holds a 30 % direct interest in the property. The Teck Cominco interest is subject to certain underlying Agreements.

Messr. G. Salazar S. (Salazar) has entered into an Option Agreement with Teck Cominco Limited to acquire 100 % of Teck's Direct Holding by incurring C$5,000,000 in expenditures on or before December 31, 2006 and aggregate expenditures of C$15,000,000 on or before December 31, 2011. As well, Salazar may acquire the Indirect Holding at any time after incurring the C$5,000,000 in expenditures by completing a "positive bankable feasibility study" and delivering a Feasibility Notice to Teck Cominco. If the Feasibility Notice is delivered prior to the aggregate C$15,000,000 in expenditures then Salazar will be deemed to have exercised the Option to acquire the Direct Holding. If Salazar acquires the Direct Holding then there is no requirement to acquire the Indirect Holding.

The Option Agreement is subject to a "back-in" provision whereby Teck Cominco has the right to acquire up to a 75 % interest in Salazar's Direct Holding and Indirect Holding in the Property. The back-in right will expire if Teck Cominco fails to give due notice of its' wish to exercise this right to Salazar within 120 days of the delivery of the Feasibility Notice.

The back-in right may be exercised by Teck Cominco in three ways:

> Teck Cominco may acquire a 20 % interest in the Property by funding forward expenditures in an amount equal to prior expenditures by Salazar,
>
> Teck Cominco may acquire a 40 % interest in the Property by funding forward expenditures in an amount equal to three times the prior expenditures by Salazar, and;
>
> Teck Cominco may acquire a 75 % interest in the property by funding forward expenditures in an amount equal to four times the prior expenditures by Salazar, and, providing notice of Teck Cominco's commitment to use its' best efforts to arrange project debt financing for not less than 60 % of the project capital cost requirements, and, should the project costs exceed the available project financing, Teck Cominco shall provide Salazar's equity portion on a subordinated loan basis at a defined prime rate plus 2 %. In this case Teck Cominco at its' sole discretion may make a decision to place the Property into commercial production

In exercising the back-in right, Teck Cominco will form a Joint-Venture to be managed by a Management Committee and each partner will own their respective interest in the Property and bear all expenditures and liabilities proportional to their respective interests.

If the Option Agreement is not exercised in respect of the Indirect Holding or Teck Cominco does not exercise its' back-in right then Salazar will grant to Teck Cominco (at Teck's sole discretion) either a 1 % net smelter royalty or (if Salazar has assigned the Option Agreement) shares in that company having a value of C$1,000,000.

Salazar has assumed the existing environmental liabilities and obligations associated with the Schaft Creek Property. Salazar has assigned the Option Agreement to 955528 Alberta Ltd..

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The technical basis for this opinion is based on AMCL's review of all of the available historical data. As well, a NI 43 101 compliant technical report has been commissioned by 955528 Alberta Ltd., 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.*

Mr. Peter Lacroix, P.Eng., Chief Mining Consultant, AMCL visited the property from April 16-17, 2004.

Associated Mining Consultants Ltd. considers that the Schaft Creek Property should be regarded as a mineral resource property *ie.* an advanced stage exploration project with a classified mineral resource. Results of metallurgical testing currently underway and the planned preliminary assessment or scoping study will determine whether to seek financing to move the project to a development phase.

Valuation Methodologies

Associated Mining Consultants Ltd. uses valuation methodologies for mineral properties which conform to:

> *Standards and Guidelines for Valuation of Mineral Properties. Prepared by the Special Committee of the Canadian Institute of Mining, Metallurgy and Petroleum on Valuation of Mineral Properties (CIMVAL) and Dated February 2003.*

Three broadly accepted valuation approaches are available to the mineral property valuator. These include the:

Income approach,

market approach, and;

cost approach.

The income approach is based on the analysis of the potential cash flows (discounted cash flow-return on investment or DCFROI) that may accrue from a mining project. This approach is only suitable for development or production properties (in rare cases in may be applied to mineral resource properties after a preliminary assessment has been completed) and is not considered an appropriate method for providing a valuation of Schaft Creek.

Option pricing is an evolving methodology classified as an income approach which may have some application to Schaft Creek. The Agreement to acquire Schaft Creek could be described as a "**call**-like" real option where the optionee (Salazar) has the right, but not the obligation, to invest money in the project and acquire an interest in the property. Most authors feel that the option valuation methodology is most appropriate for projects that have progressed beyond the exploration stage to at least the pre-feasibility stage where there is a defined resource and a reasonable estimate of capital and operating costs.

The market approach involves sales comparisons between comparable mineral property transactions and/or the analysis of option or farm-in agreement terms. Both methods require open market, arm's length transactions to be valid. This is the most applicable approach to the valuation of mineral resource properties where suitable comparable transactions or agreements have been documented.

The cost approach involves a subjective assessment of the contribution of past and proposed exploration expenditures to the potential of the property. The principle of contribution to value is also known as the appraised value methodology. While this can be a useful technique, or as a check on another methodology, its' application to Schaft Creek is limited by the historic nature of the expenditures and the subsequent evolution of exploration models for porphyry type deposits as well as advances in metallurgical techniques.

AMCL is strongly of the opinion that, where possible, multiple methods should be used in estimating the value for any mineral project.

Valuation of the Schaft Creek Project

Market Approach-Comparable Transaction

In order to apply this approach the valuator must identify an open market, arm's length transaction involving a similar mineral project to that being valued. The projects should be in the same stage of development, have a common suite of commodities and a similar geological setting, involve similar project risks and be located in a common part of the world. AMCL searched for recent transaction involving copper (± gold, ± molybdenum) porphyry deposits of the northwestern cordillera including the Mt. Polley, Galore Creek, Kemess North, Kemess South and Red-Chris deposits.

The NovaGold Resources Inc. and Spectrum Gold Inc. Galore Creek project seems like an obvious candidate but AMCL believes that this project is much further advanced than Schaft Creek, in that, a 3,000 m exploration program completed in the fall of 2003 has been used as the basis for an updated resource estimate which will form the backbone of a preliminary economic assessment.

The Kemess North and Kemess South projects are undergoing feasibility studies or in production, respectively.

The Mount Polley deposit is undergoing a re-evaluation of the deposit model as a result of the successful identification of additional resources by drilling based on recent geophysics.

The recent acquisition by bcMetals Corporation (formerly American Reserve Energy Corporation) in 2003 of an 80 % interest in the Red Chris deposit from American Bullion Minerals Ltd. (in a very complex transaction involving a joint-venture agreement and subsequent litigation) and an Option

July 20, 2004

Agreement with Teck Cominco allows AMCL to place a value on this deposit.

Details of the various transactions involving the Red-Chris Deposit are available on the bcMetals Corporation website. As well, a recent NI 43-101 compliant technical report was filed on SEDAR by the Company, entitled:

> *Report on the Red Chris Copper-Gold Project, Liard Mining Division. Prepared by G.H. Giroux, P.Eng., Giroux Consultants Ltd., Robert Rodger, P.Eng. and J.D. Blanchflower, P.Geo., Minorex Consulting Ltd. and Dated November 18, 2002.*

There are some differences between Schaft Creek and Red Chris which will have to be considered or factored into the valuation. The Red Chris deposit is located closer to power and other infrastructure and is higher grade (both copper and gold credits) than Schaft Creek but has a considerably lower tonnage base. A review of the Schaft Creek resource estimate suggests that it is likely that higher-grade zones may be identified within the deposit. The larger resource base may be able to sustain the higher infrastructure development costs.

The biggest difference between the two properties is the issue of ownership. Currently, the Red Chris Development Company (a wholly owned subsidiary of bcMetals Corporation) holds the rights to 80 % of the property with Teck Cominco holding 20 % and back-in rights to 43.75 % of the 80 % interest. bcMetals Corporation has entered into an Option Agreement to acquire Teck Cominco's interest. Schaft Creek is the subject of an Option Agreement with Teck Cominco.

Despite these differences AMCL believes that the Red Chris acquisition is a suitable analogue on which to base a valuation of the Schaft Creek Deposit.

In a press-release dated January 15, 2003 American Reserve Energy Corporation (the predecessor to bcMetals Corporation) acquired a 100 % interest in the privately held Red Chris Development Company for C$267,167.14 and the issuance of 2,000,000 shares. Red Chris Development's sole asset was a joint-venture agreement with American Bullion Minerals Ltd. in respect of the Red Chris Property whereby Red Chris Development (as operator) would have a 70 % interest and American Bullion a 30 % interest in American Bullion's 80 % interest (Teck Cominco holding the other 20 %) in the Property.

The terms of the Joint-Venture Agreement required Red Chris Development to pay an aggregate of C$2,000,000 to American Bullion broken down as follows:

> C$250,000 was paid on the signing of the Joint-Venture Agreement,
>
> C$500,000 was due on the assignment of the Joint-Venture Agreement to a publicly listed company-bcMetals Corporation (the "Approval Date"),

> C$625,000 on the first anniversary of the Approval Date, and;
>
> C$125,000 on the second, third, fourth, fifth and sixth anniversary of the Approval Date.

American Bullion's interest constitutes a 30 % reversionary carried ownership interest. Upon attaining commercial production Red Chris Development would recover 100 % of its' costs incurred on the property and American Bullion would be repaid (on a *pro rata* basis) C$10,000,000 in prior expenditures on the property. American Bullion's interest would then revert to a 30 % working interest.

A long period of litigation followed whereby American Bullion attempted to overturn the Joint-Venture Agreement. The Supreme Court of British Columbia has recently ruled in the favour of bcMetals Corporation and upheld all of the terms of the Joint-Venture Agreement.

The terms of the Joint-Venture Agreement valued a 56 % interest in the property at C$2,000,000 (recognizing Teck Cominco's 20 % interest and back-in rights and the 30 % reversionary carried ownership interest. This effectively valued 100 % of the Red Chris Property, with all of its' encumbrances, at C$3,571,429.

Subsequently, bcMetals Corporation entered into an Option Agreement with Teck Cominco to acquire their 20 % interest. Exercise of this Option would result from fulfilling the following conditions:

> Payment of C$300,000 in cash 250,000 warrants (exercisable until December 31, 2006) at C$0.60/share,
>
> issuance of 500,000 shares and 500,000 warrants on March 31, 2004,
>
> issuance of 500,000 shares and 500,000 warrants on July 31, 2004, and:
>
> issuance of 500,000 shares and 250,000 warrants on December 31, 2004.

The share purchase warrants have a three year term (subject to a four month hold period) and are exercisable at a price equal to the greater of C$0.60 or the average trading price 10 days prior to the issuance of the warrants plus a 20 % premium in the first year, a 40 % premium in the second year and a 50 % premium in the third year. An additional C$1,000,000 would be paid to Teck Cominco within one year of the commencement of commercial production. This type of complex share transaction is virtually impossible to value realistically.

As an academic exercise, if we assume that the share purchase warrants are all exercised at C$0.60 (2,000,000 warrants for C$1,200,000), the cash component of C$1,300,000, and, 1,500,000 shares sold at bcMetals Corporation the average of the 52 week rolling high and low share price (C$1.10 for

July 20, 2004

C$1,650,000) the value of Teck Cominco's 20 % interest is a minimum of C$4,150,000. This puts a minimum value on a 100 % interest in the project, assuming it proceeds to production, of C$20,750,000. The curiosity of this calculation is that this is almost exactly the same valuation Teck Cominco places on the exercise of the Option Agreement for Schaft Creek suggesting that Teck Cominco believes them to be of comparable worth.

At the same time, bcMetals Corporation also entered into an Agreement whereby a 0.571428 % carried interest held by one of the original staking prospectors was purchased for C$1,500 and 57,000 shares (trading at C$0.57 at the time of issuance) for an estimated cash equivalent of C$33,900. This effectively values 100 % of the property at C$5,932,506.

AMCL is of the opinion that the current free market value for the Red Chris Property is in the range of C$3.5 million-C$6.0 million. AMCL believes that a similar valuation can be ascribed to the Schaft Creek Property. The difficulty then becomes determining a value for the Option Agreement between Teck Cominco and 955528 Alberta Ltd..

In discussions with Messr. Salazar and various parties working on the preliminary assessment, it has been estimated that somewhere between C$2 million-C$2.5 million will need to be spent on the project over the next two years to reach the first major decision point in assessing the Schaft Creek Project's viability. This is consistent with the work programme proposed in the independent technical report on the property. In AMCL's opinion this expenditure is warranted *ie* is the minimum required to reach the next development decision point and therefore represents a probable minimum value for the Schaft Creek Property.

Conclusions

> In AMCL's opinion the value of the Option Agreement for the Schaft Creek Property as determined from a comparable transaction is somewhere between C$2.5 million-C$3.5 million (lower end of the valuation for Red Chris).

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 McCandlish, P.Geol.,
Manager of Mineral Services

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Appendix III: Preliminary Assessment of the Schaft Creek Deposit, British Columbia; Project Status Report No. 1; by Keith McCandlish, P. Geol., Associated Mining Consultants Ltd., September 20^{th}, 2004.

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

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

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September 20, 2004

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

September 20, 2004

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.

Signature Not Verified
Keith McCandlish, P.Geol.
Digitally signed by Keith McCandlish, P.Geol.
DN: cn=Keith McCandlish, P.Geol., o=AMCL, c=CA
Date: 2004.09.22 09:29:05 -06'00'

Keith McCandlish, 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. McCandlish 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. McCandlish 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

Signature Not Verified

Keith McCandlish, P.Geol.

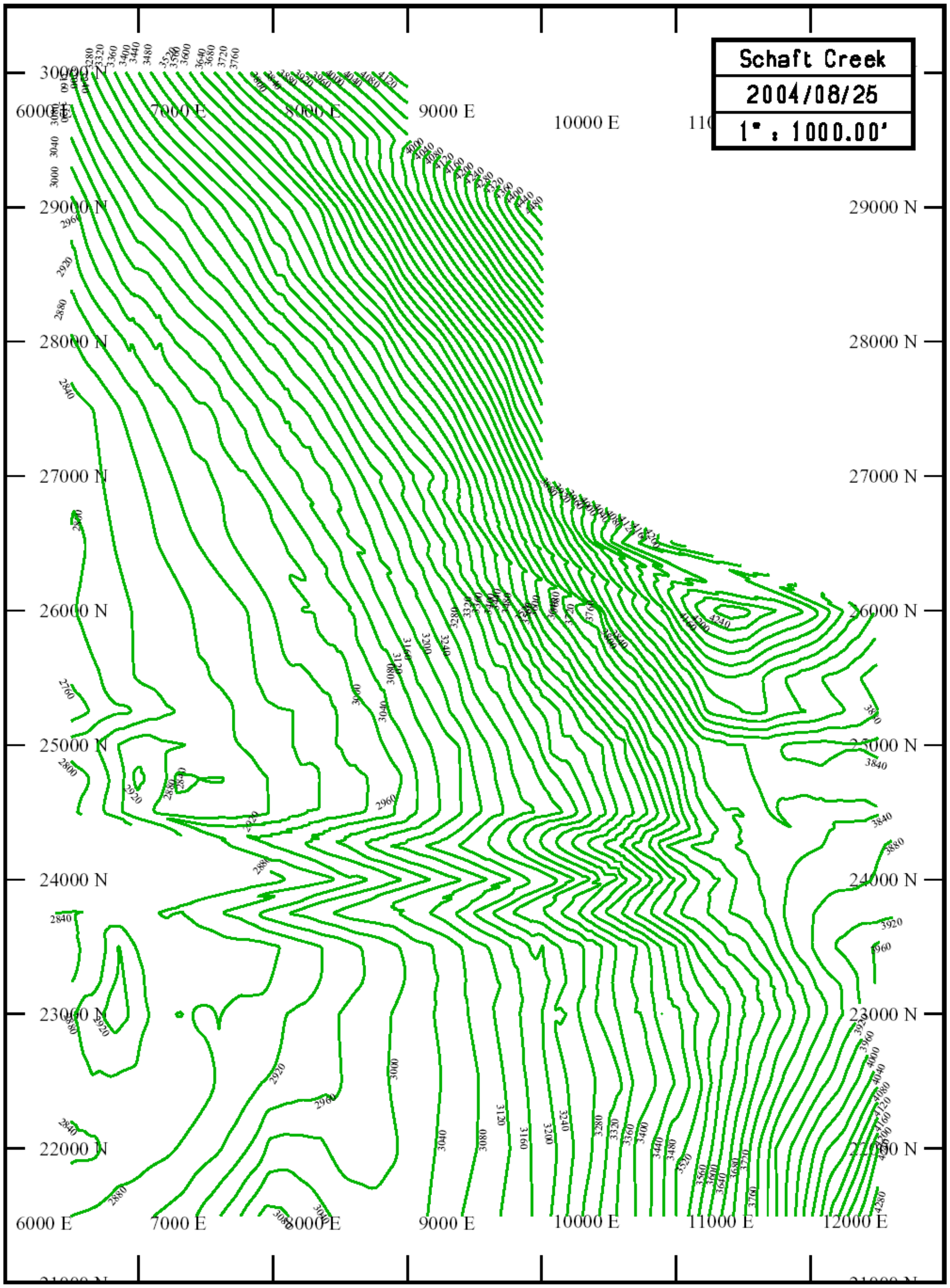
Digitally signed by Keith McCandlish, P.Geol.
DN: cn=Keith McCandlish, P.Geol., o=AMCL, c=CA
Date: 2004.09.22 09:29:40 -06'00'

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

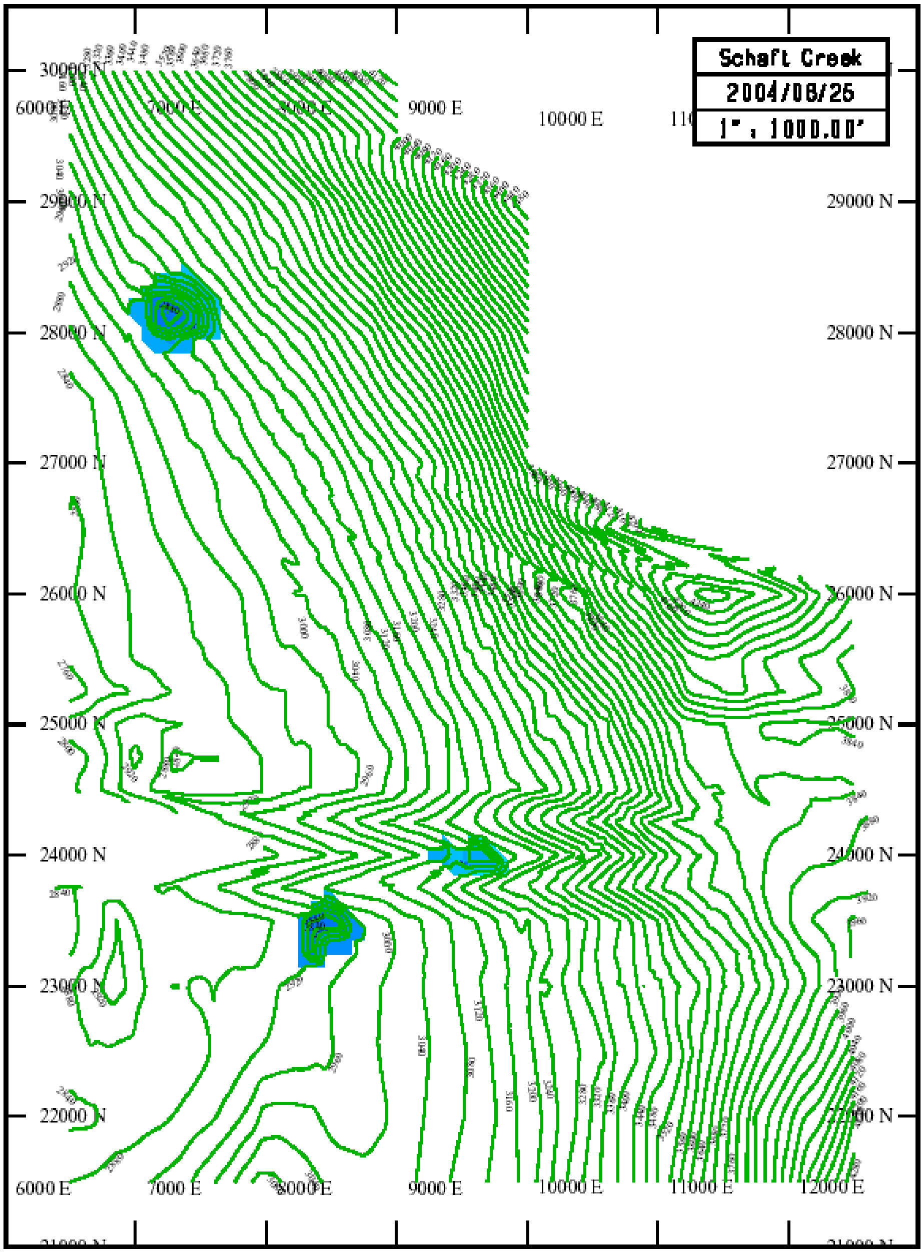
APPENDIX 1
PLAN VIEWS

File: 04PM76

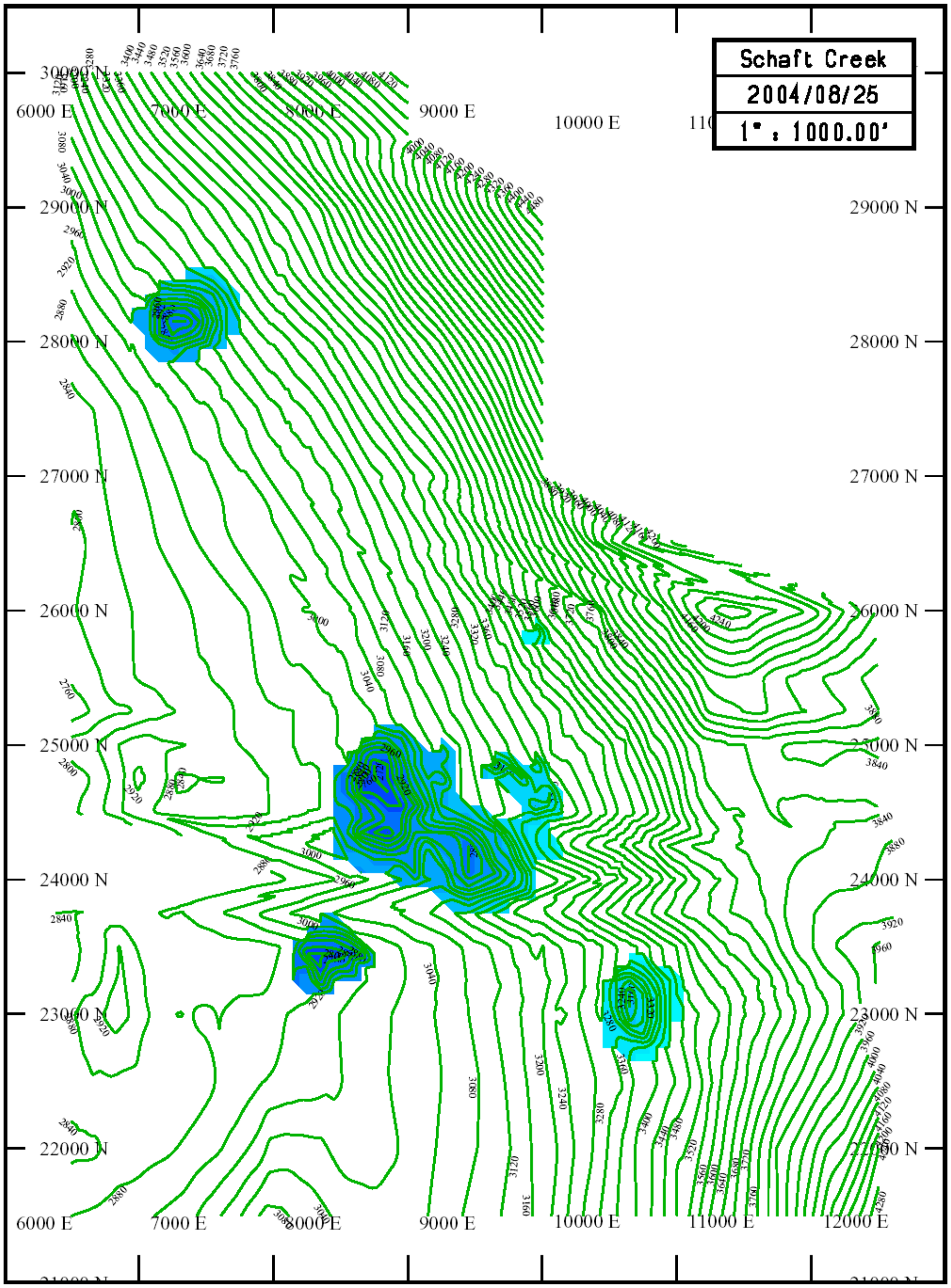
September 20, 2004



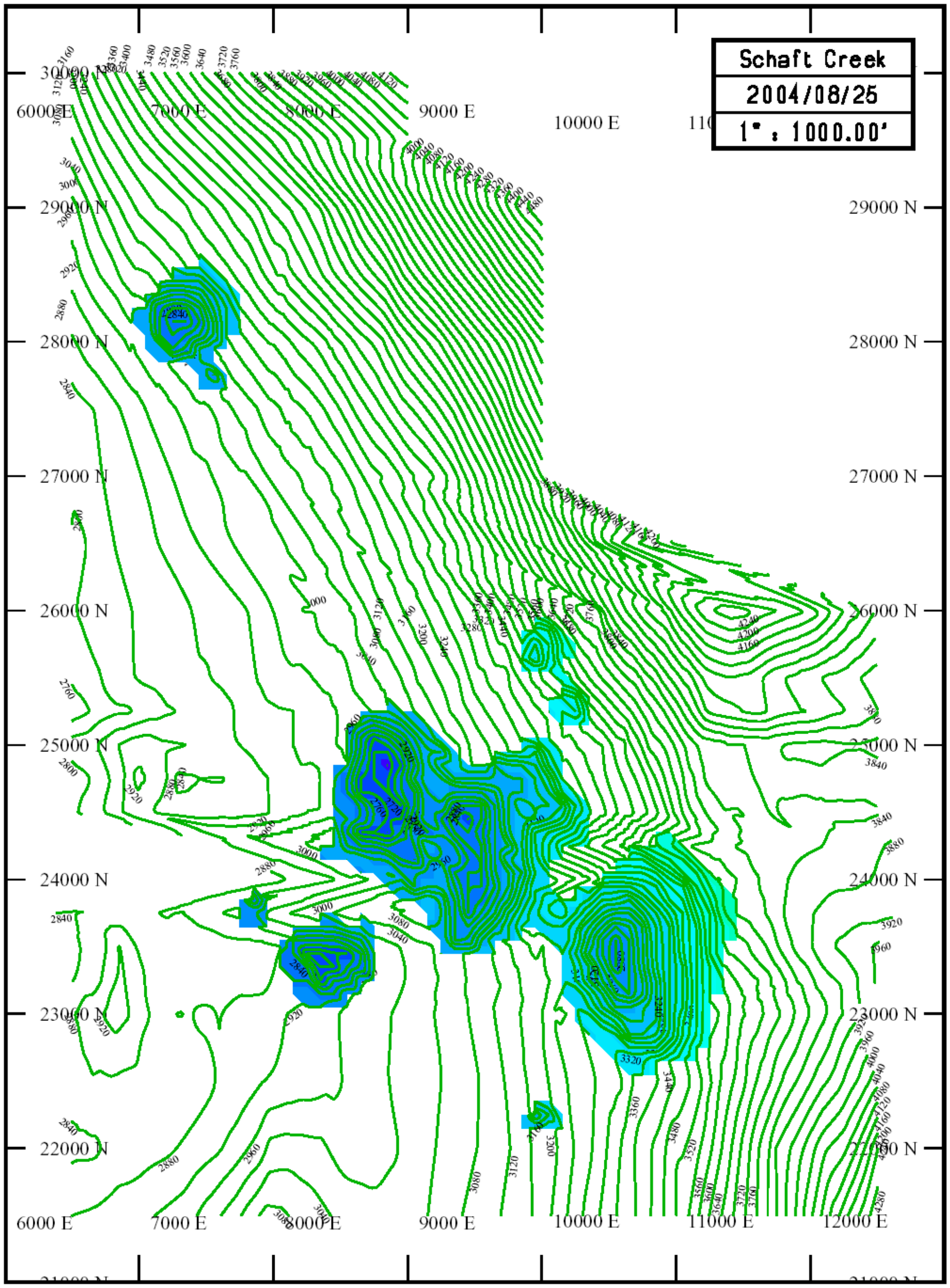
Initial Topography from Overburden Solid



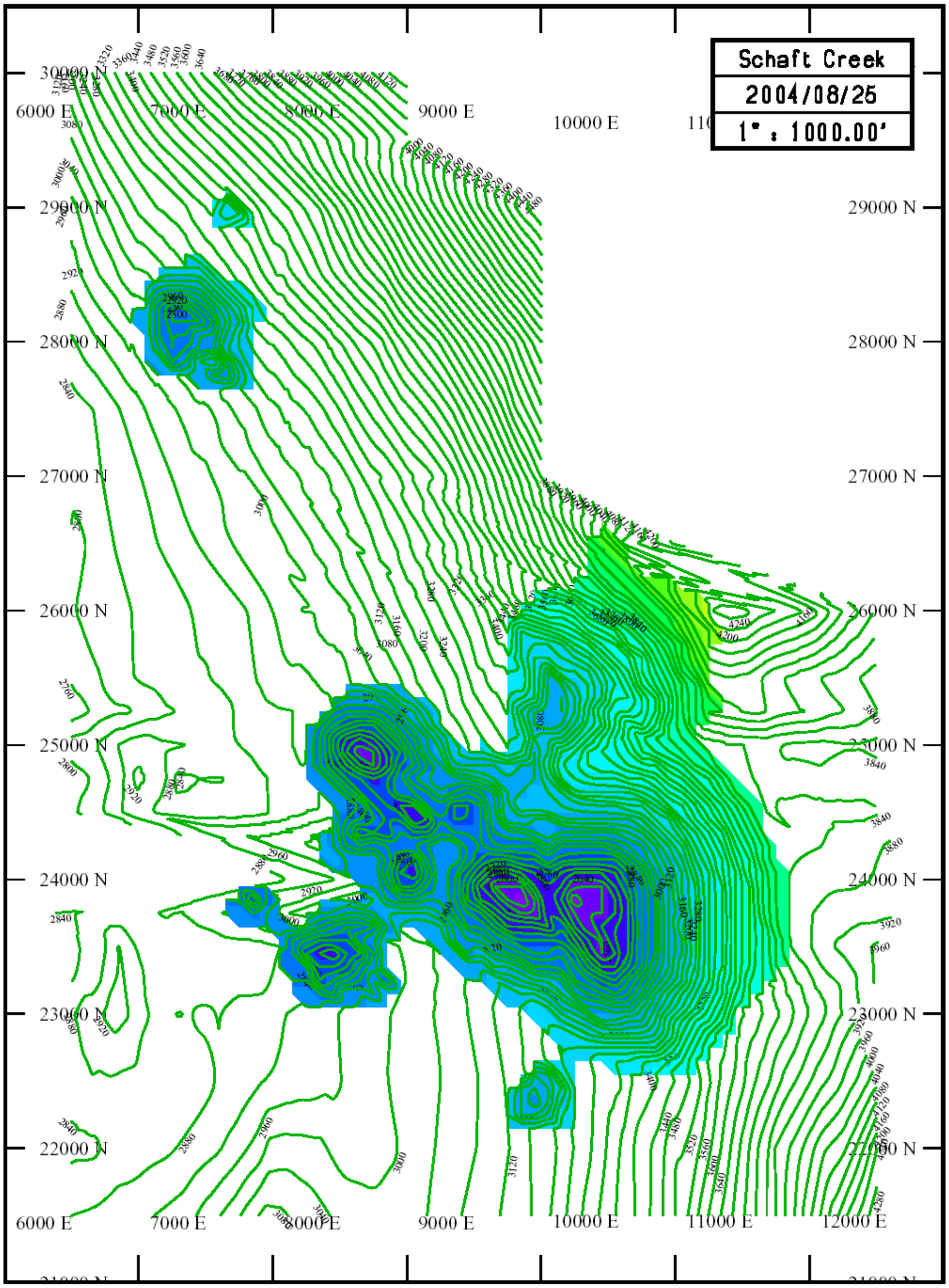
US$0.70/lb Cu Equivalent Optimized Pit Contours (Shaded)



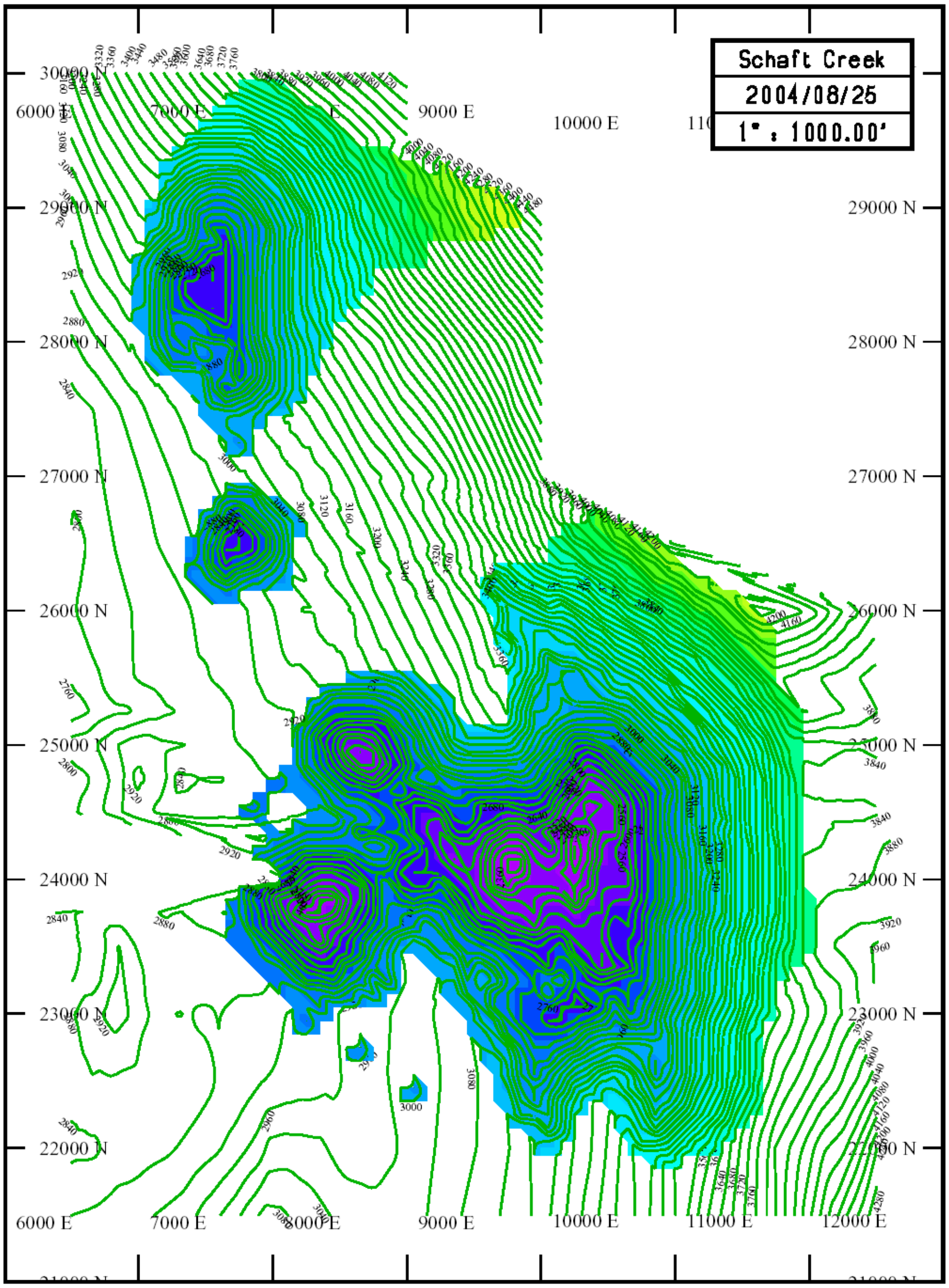
US$0.80/lb Cu Equivalent Optimized Pit Contours (Shaded)



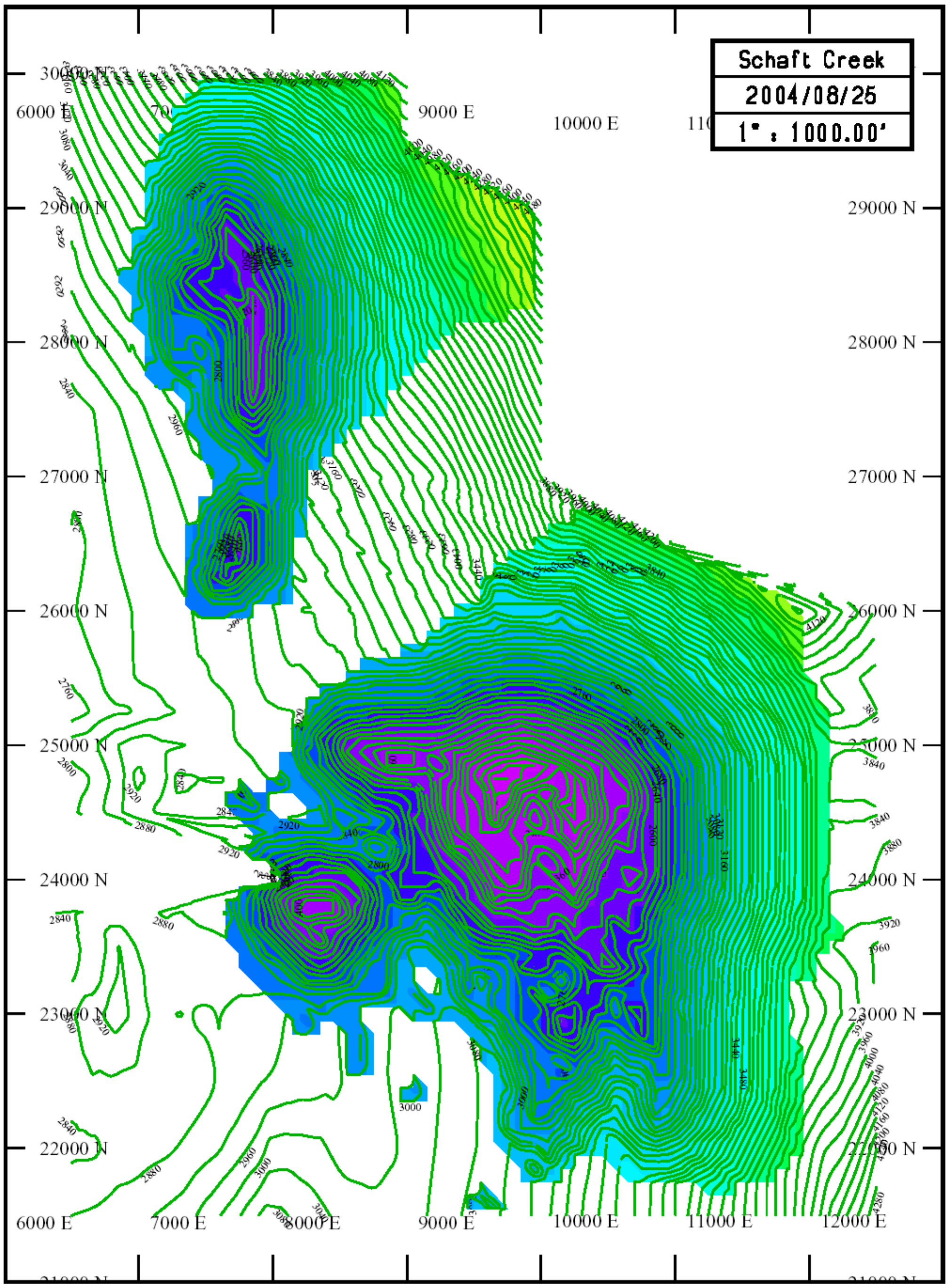
US$0.90/lb Cu Equivalent Optimized Pit Contours (Shaded)



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

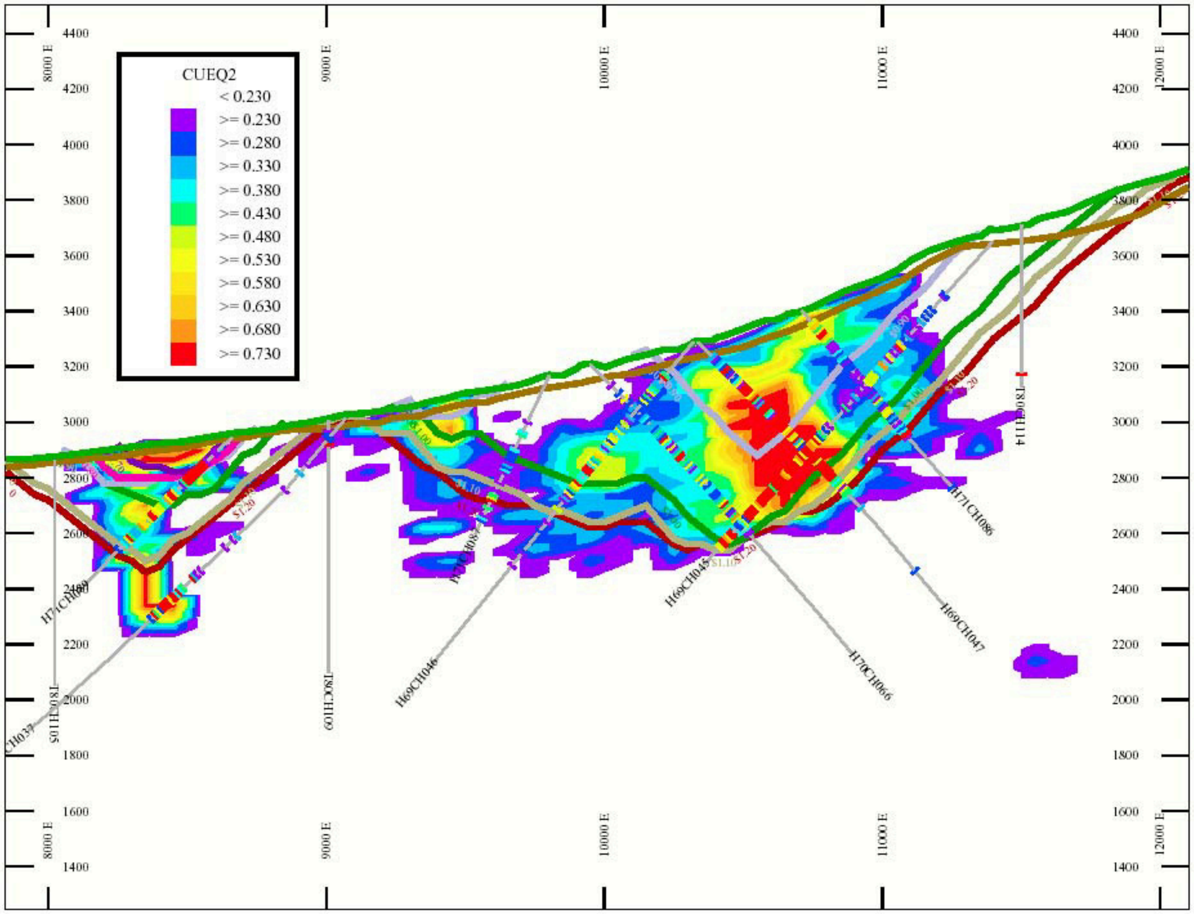


US$1.10/lb Cu Equivalent Optimized Pit Contours (Shaded)

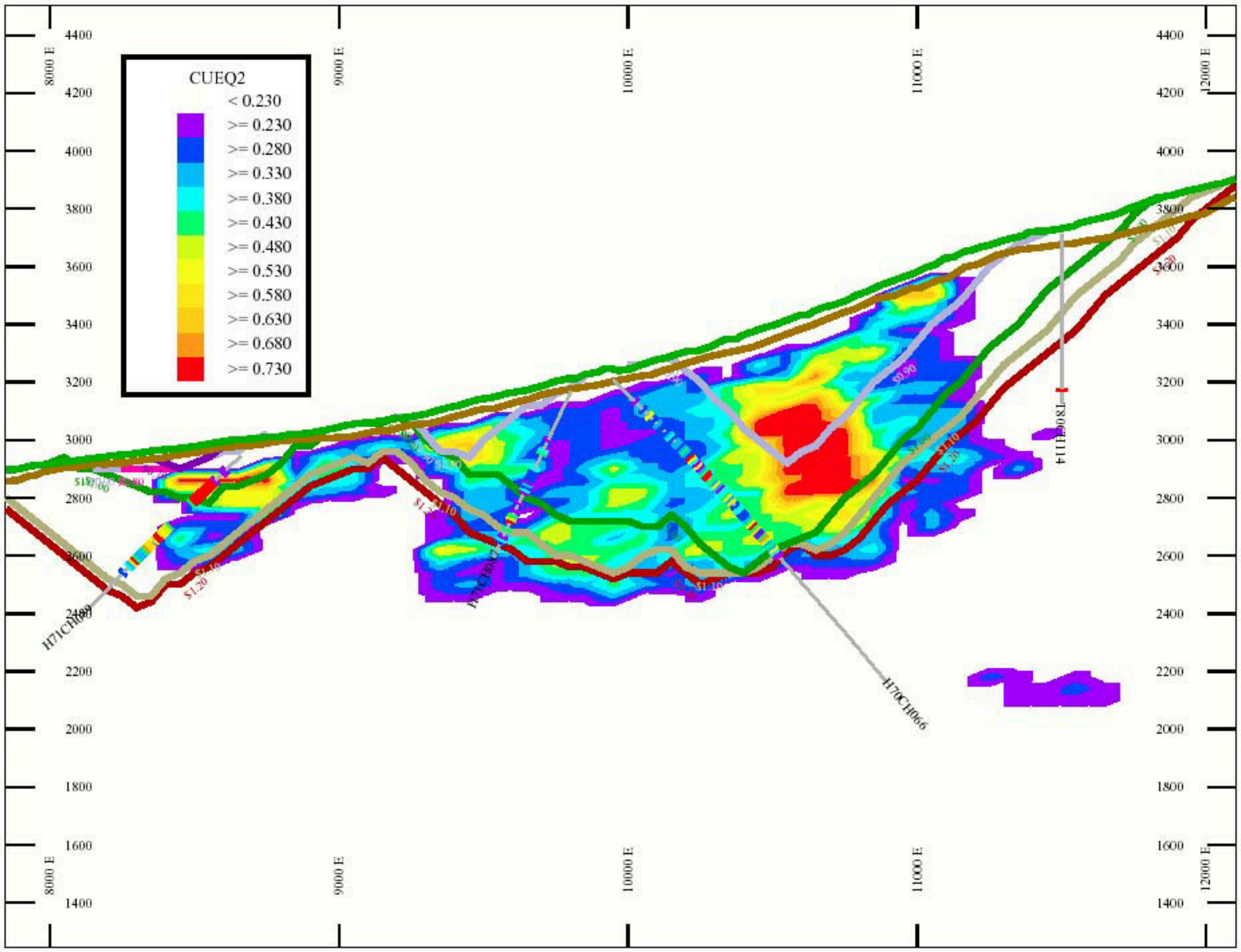


US$1.20/lb Cu Equivalent Optimized Pit Contours (Shaded)

APPENDIX 2
CROSS SECTIONS



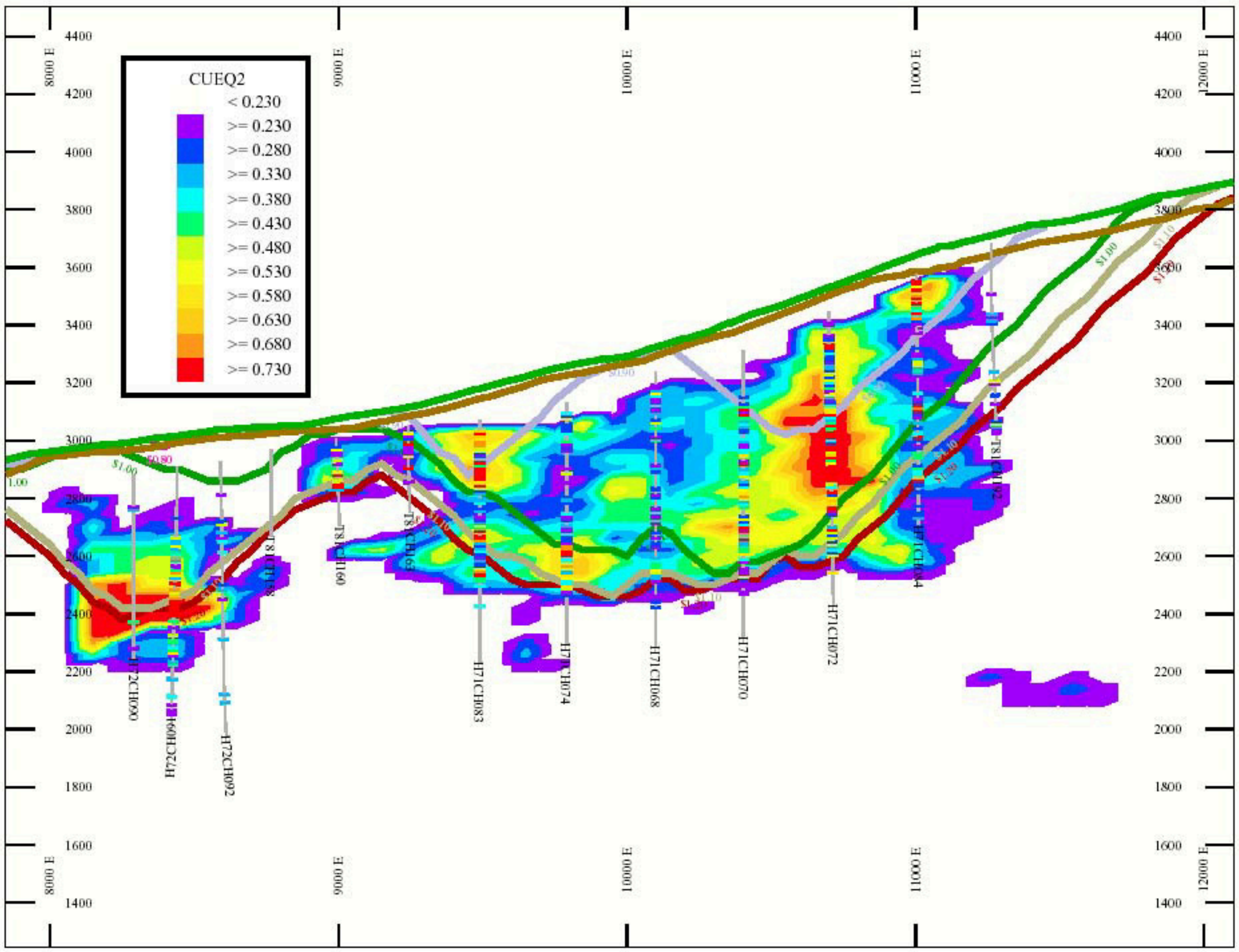
Cross Section at 23500N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



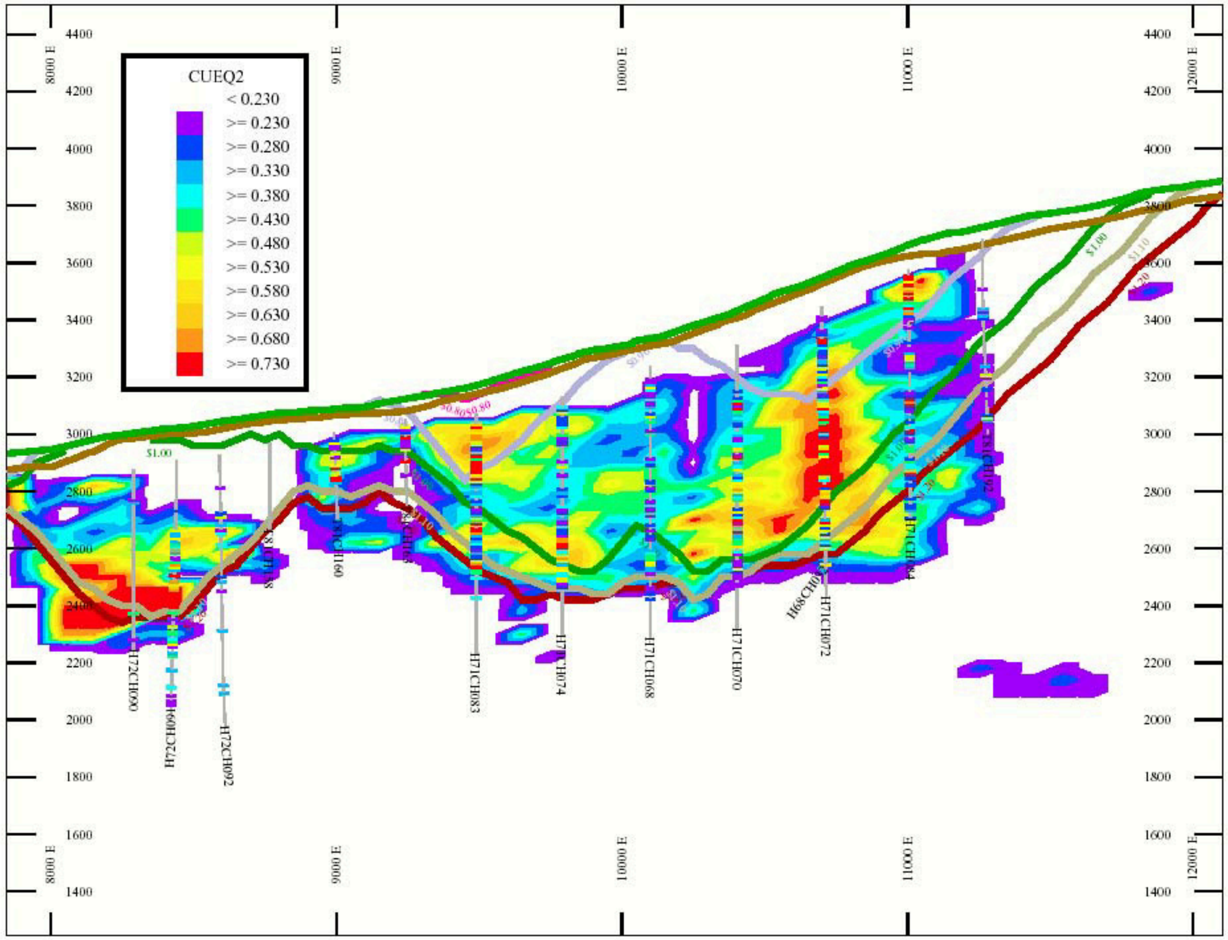
Cross Section at 23600N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

File: 04PM76

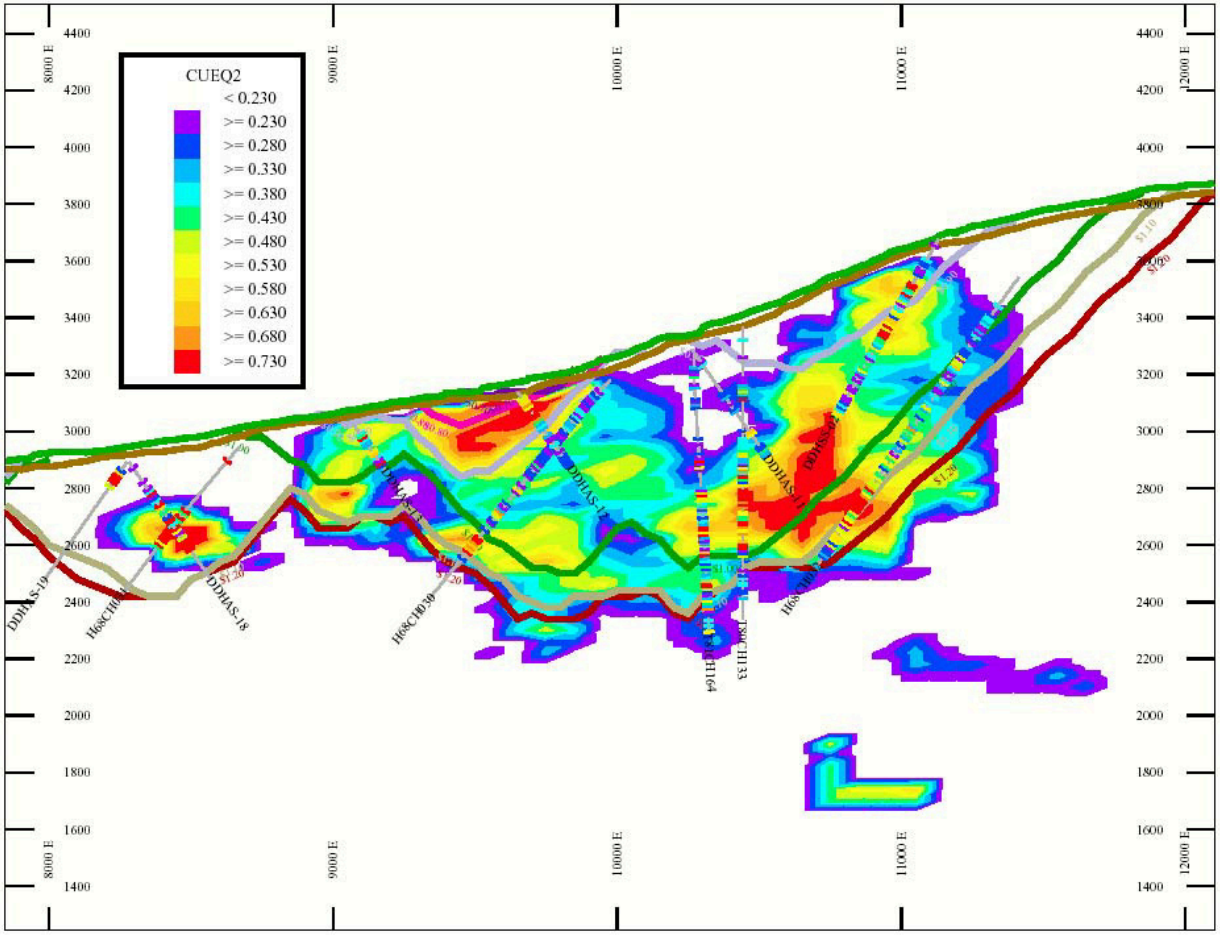
September 20, 2004



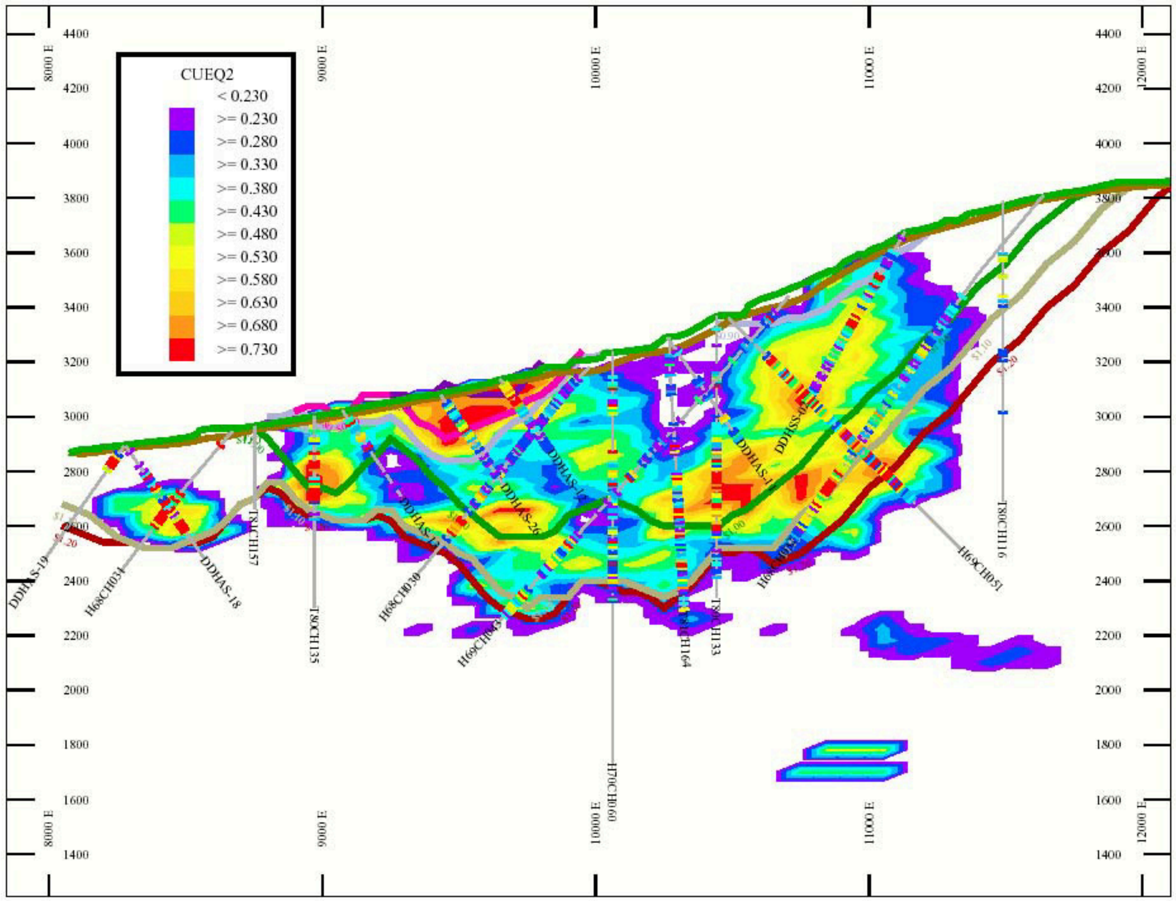
Cross Section at 23700N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



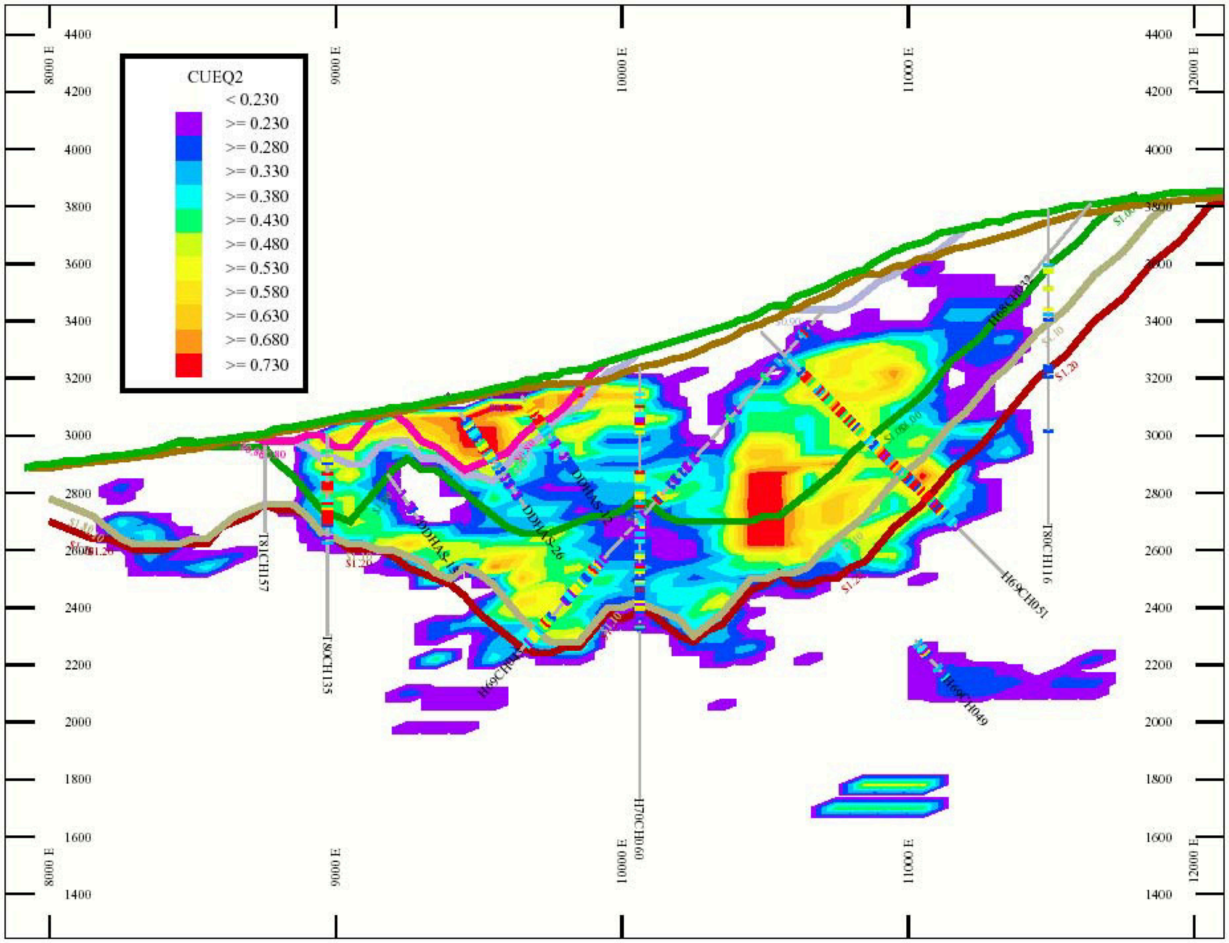
Cross Section at 23800N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



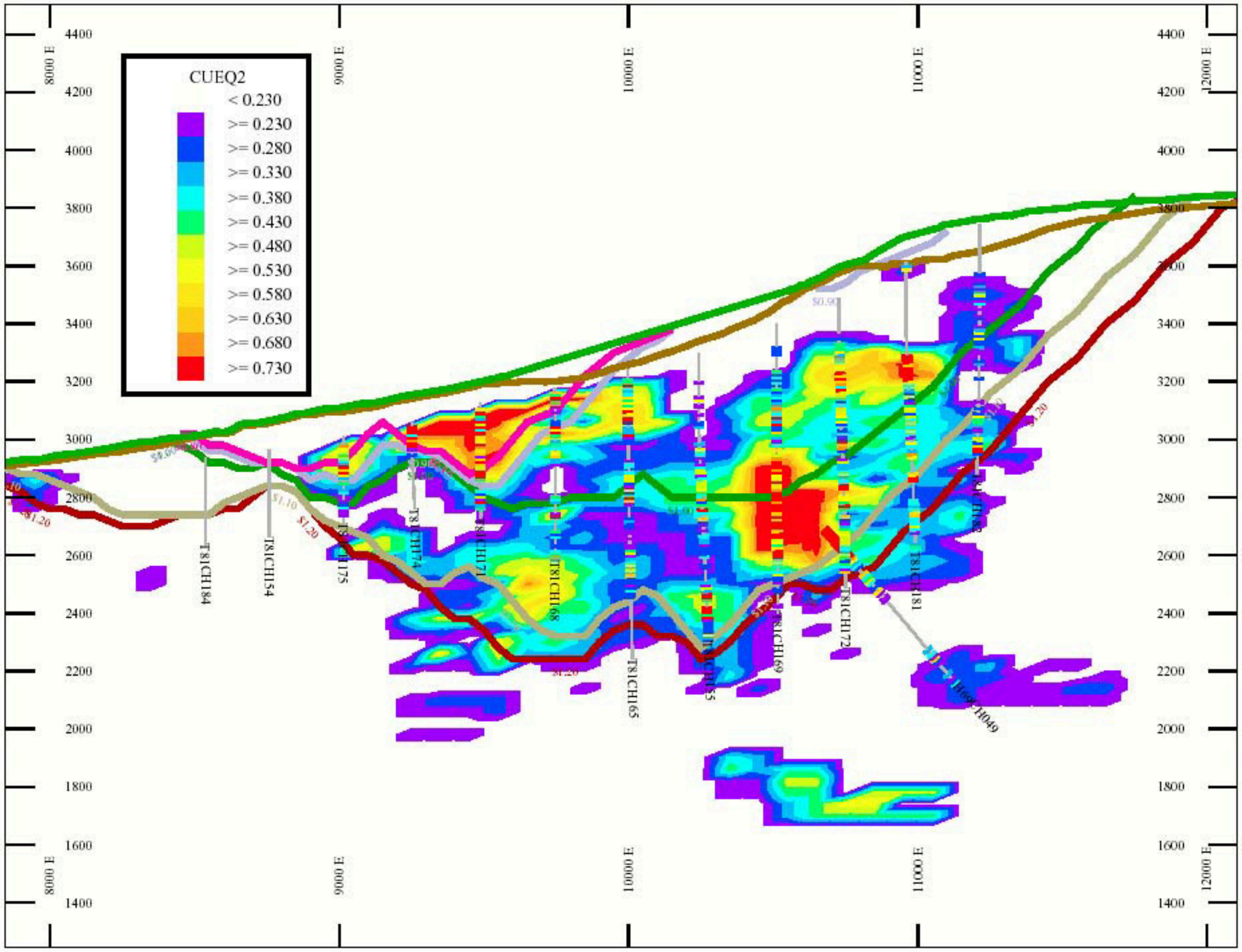
Cross Section at 23900N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



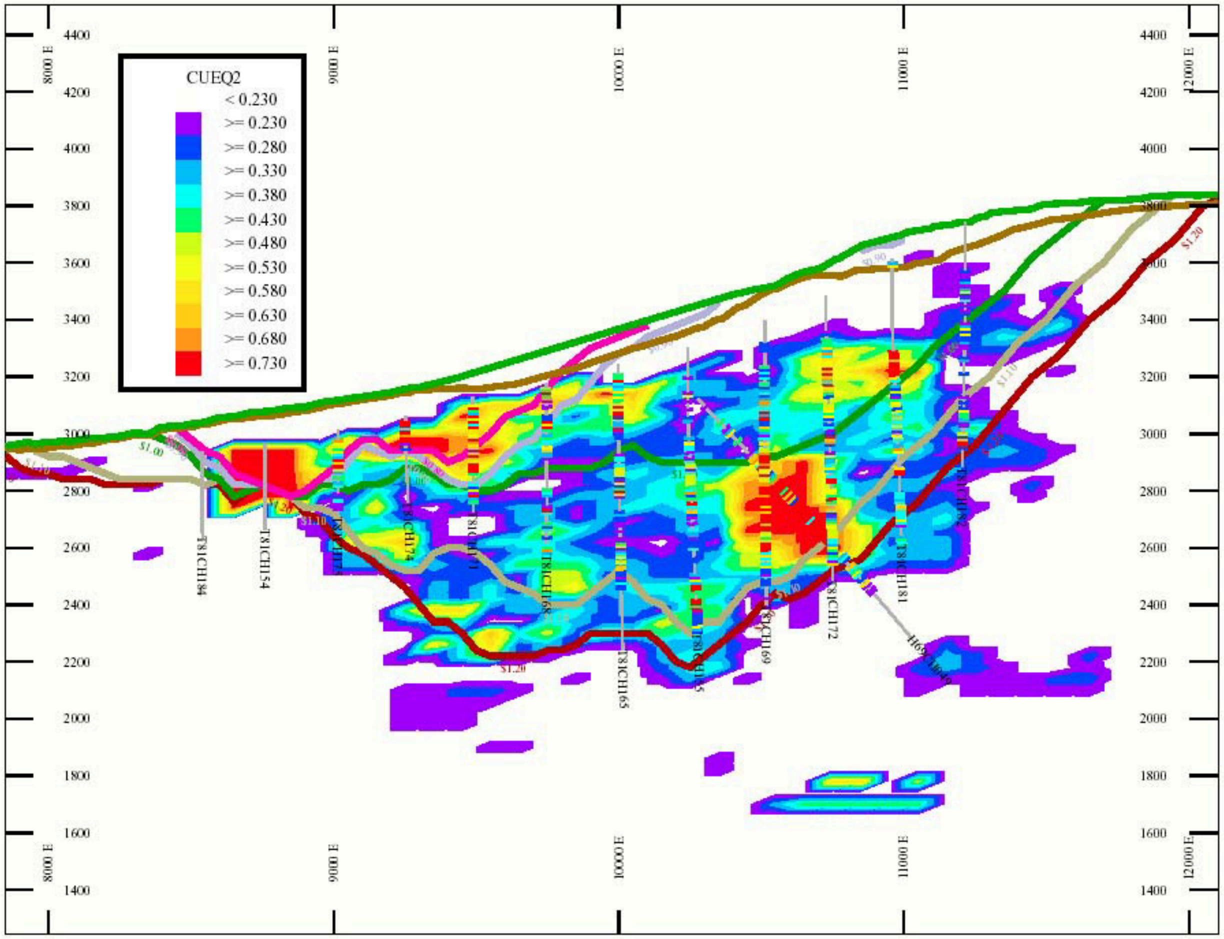
Cross Section at 24000N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



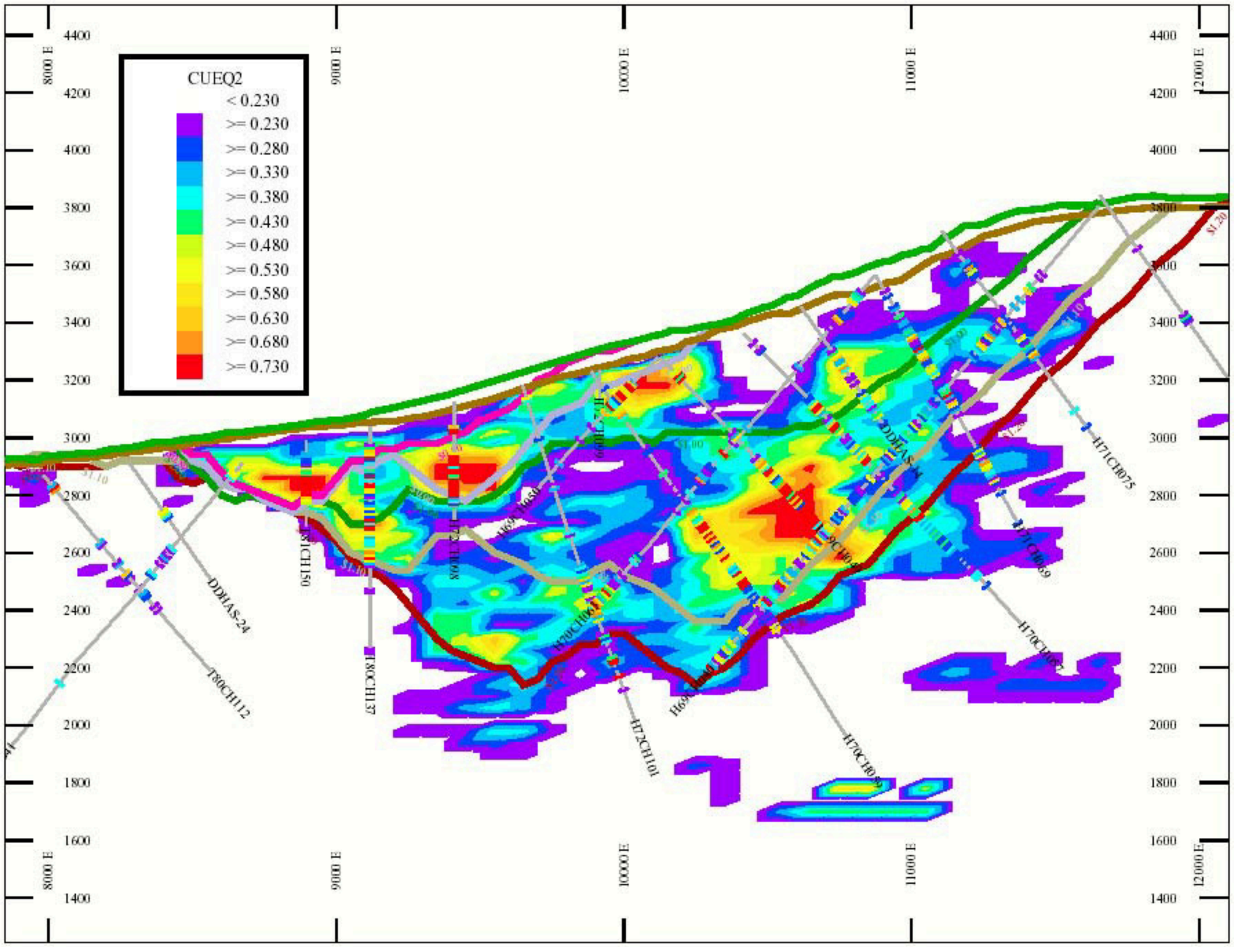
Cross Section at 24100N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



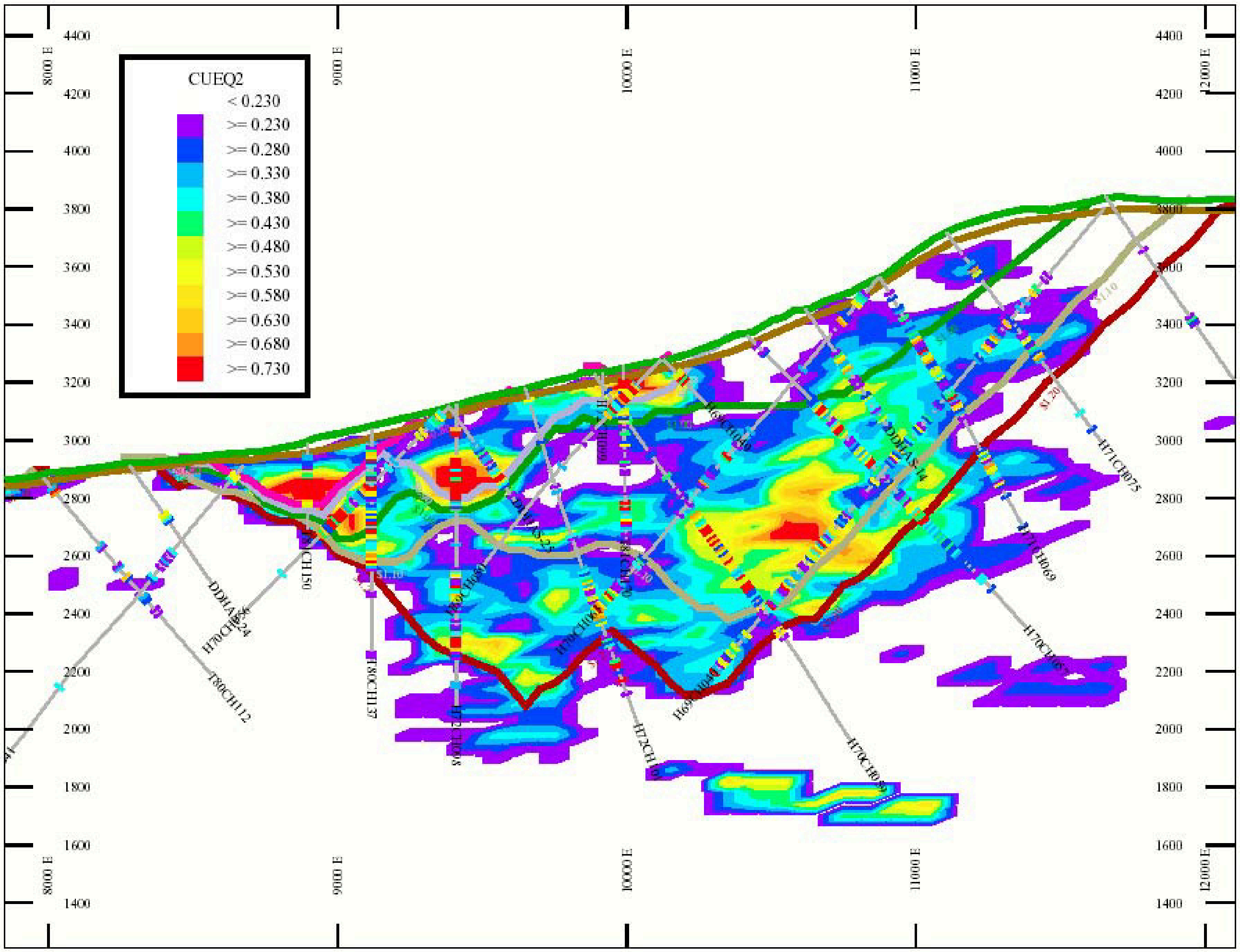
Cross Section at 24200N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



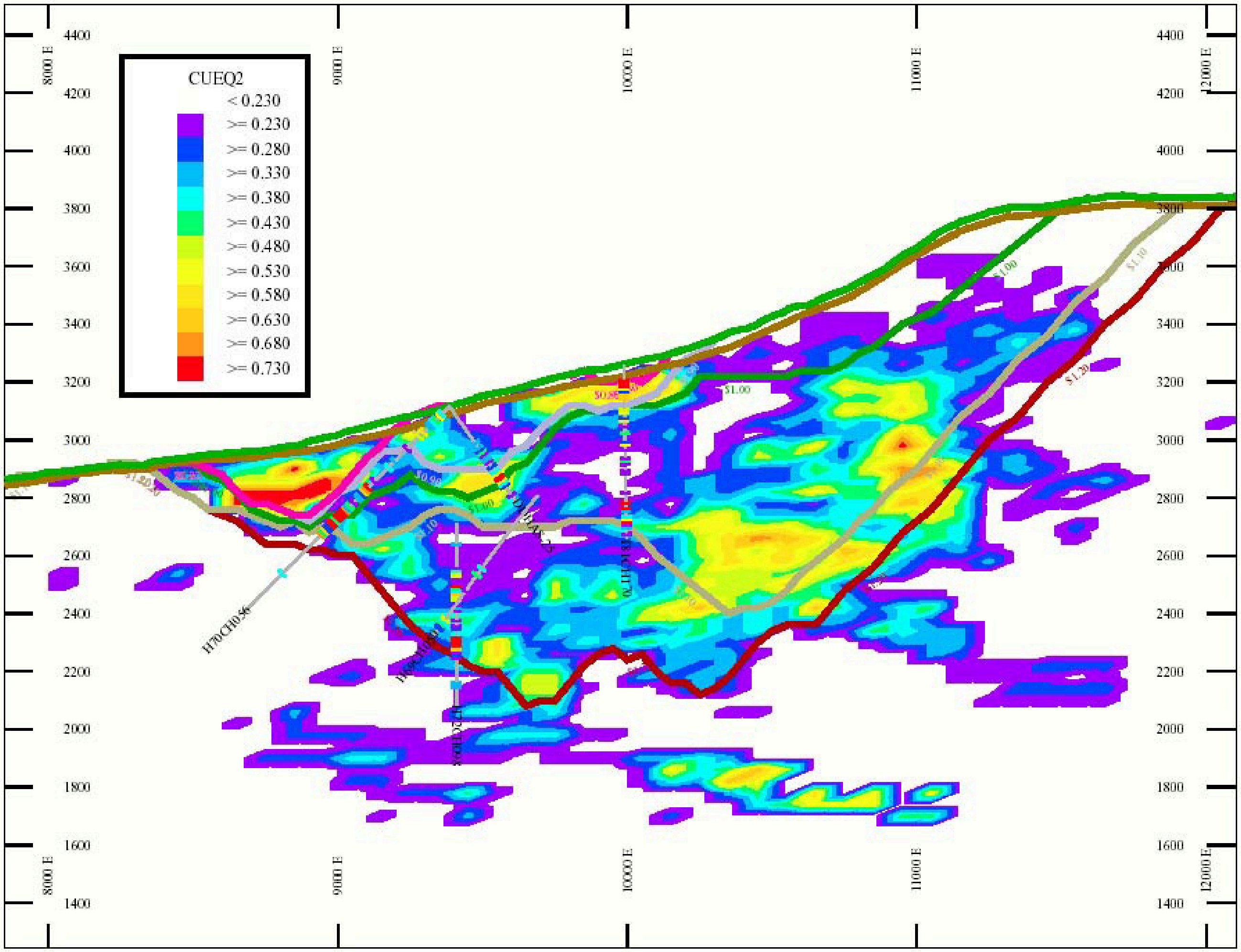
Cross Section at 24300N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



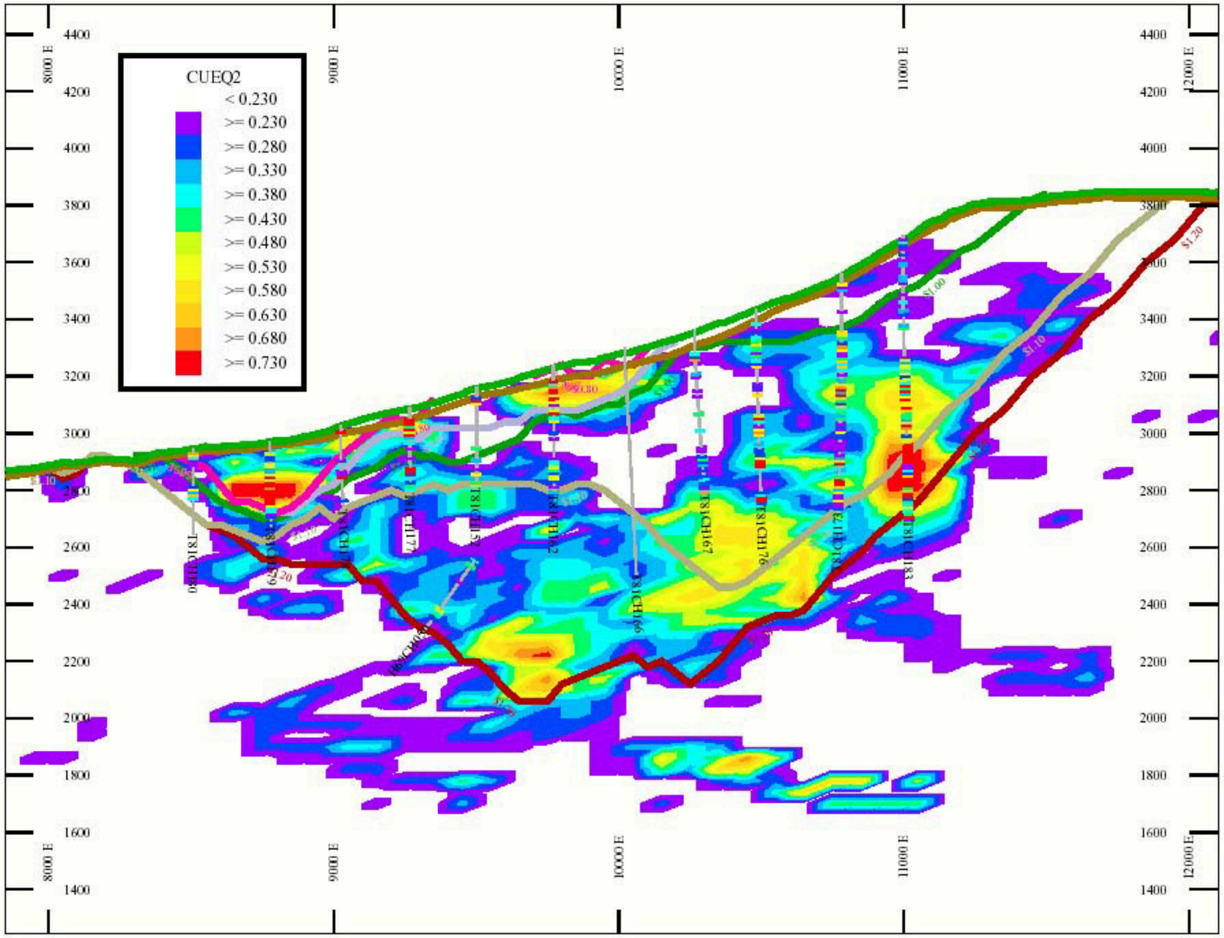
Cross Section at 24400N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



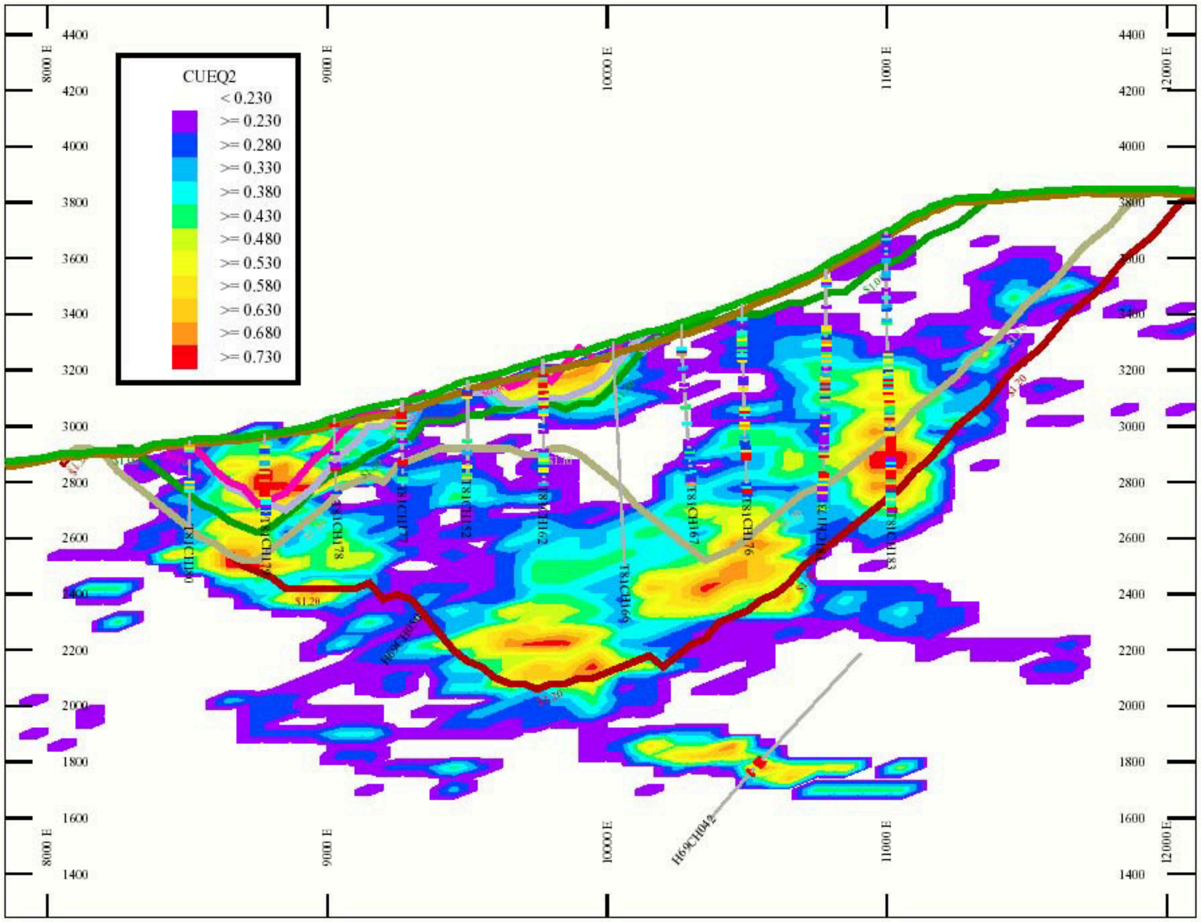
Cross Section at 24500N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



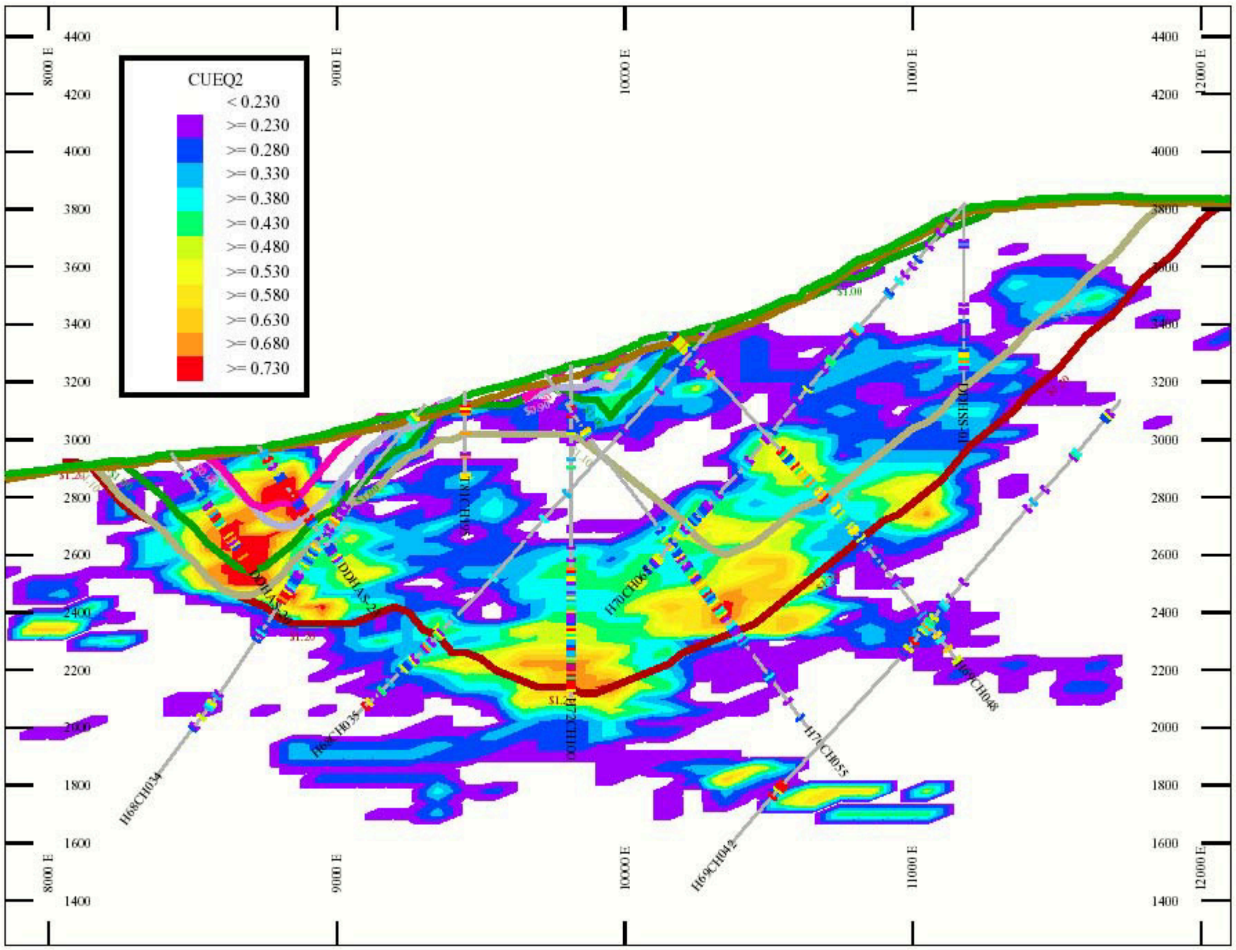
Cross Section at 24600N Showing Optimized Pit Outline and Grade Model with Net Copper Equivalent



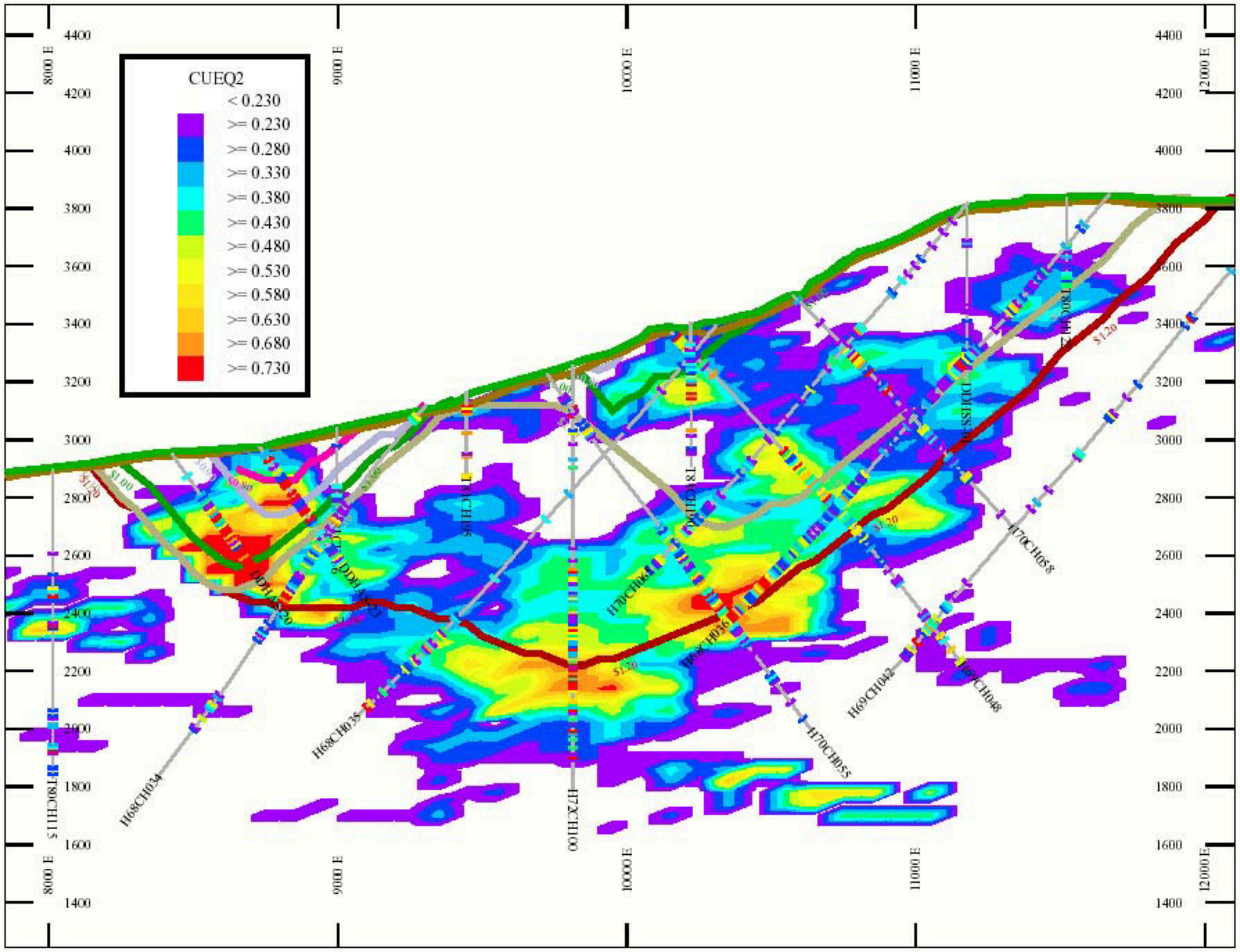
Cross Section at 24700N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



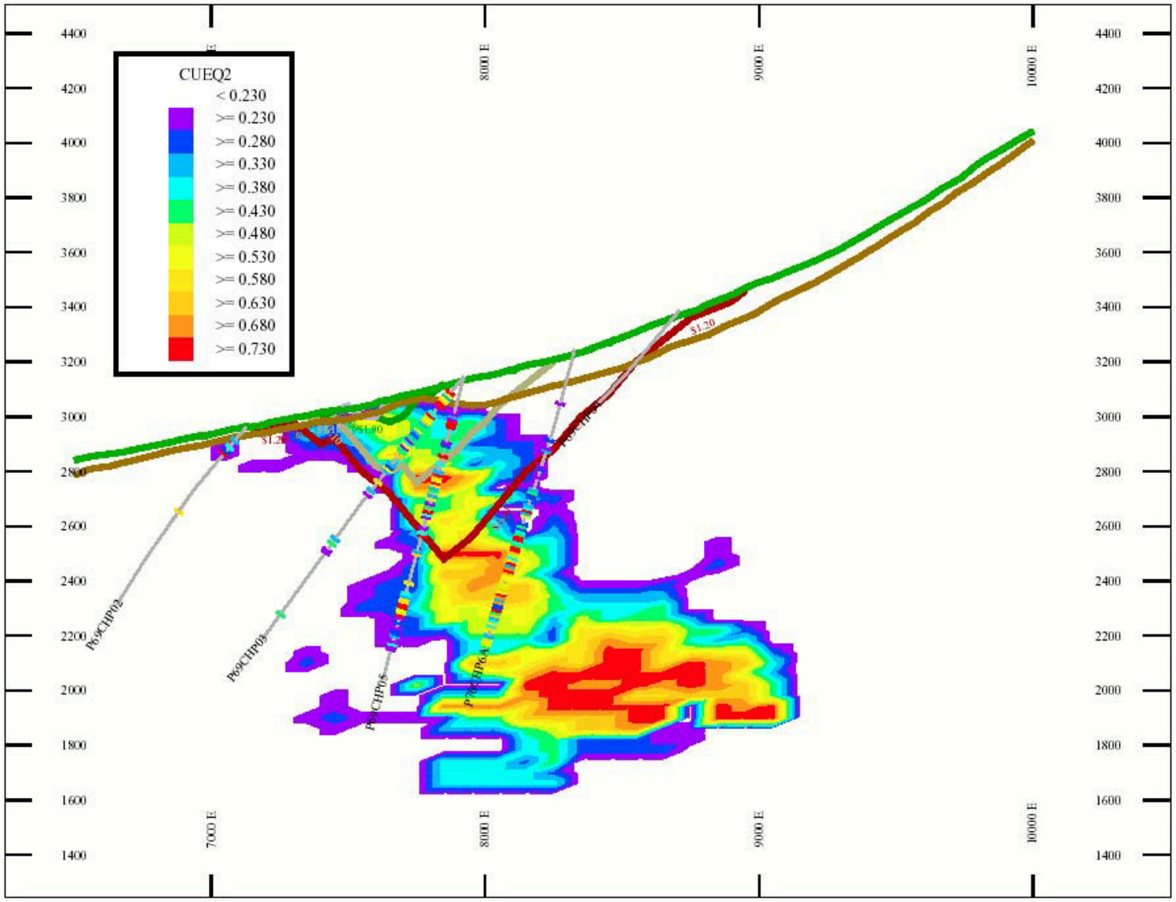
Cross Section at 24800N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



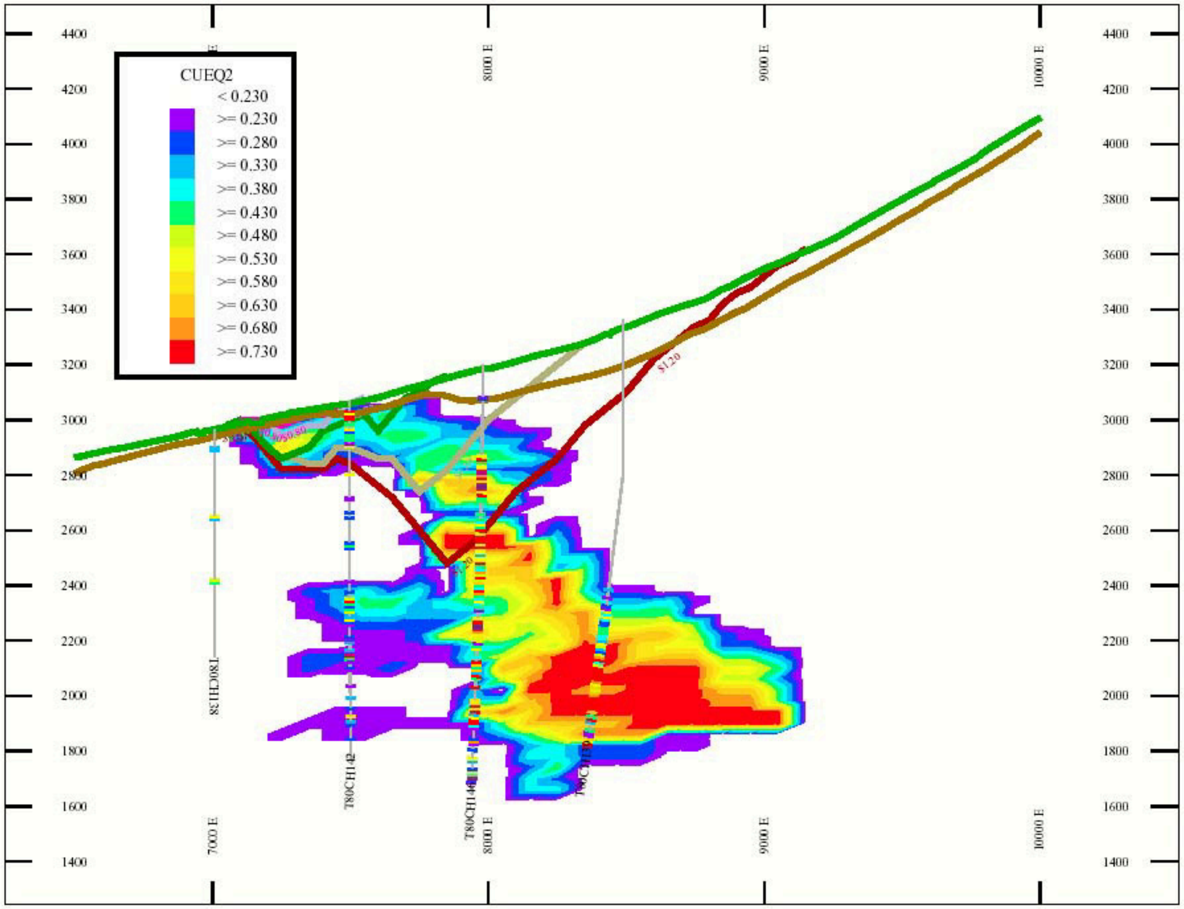
Cross Section at 24900N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



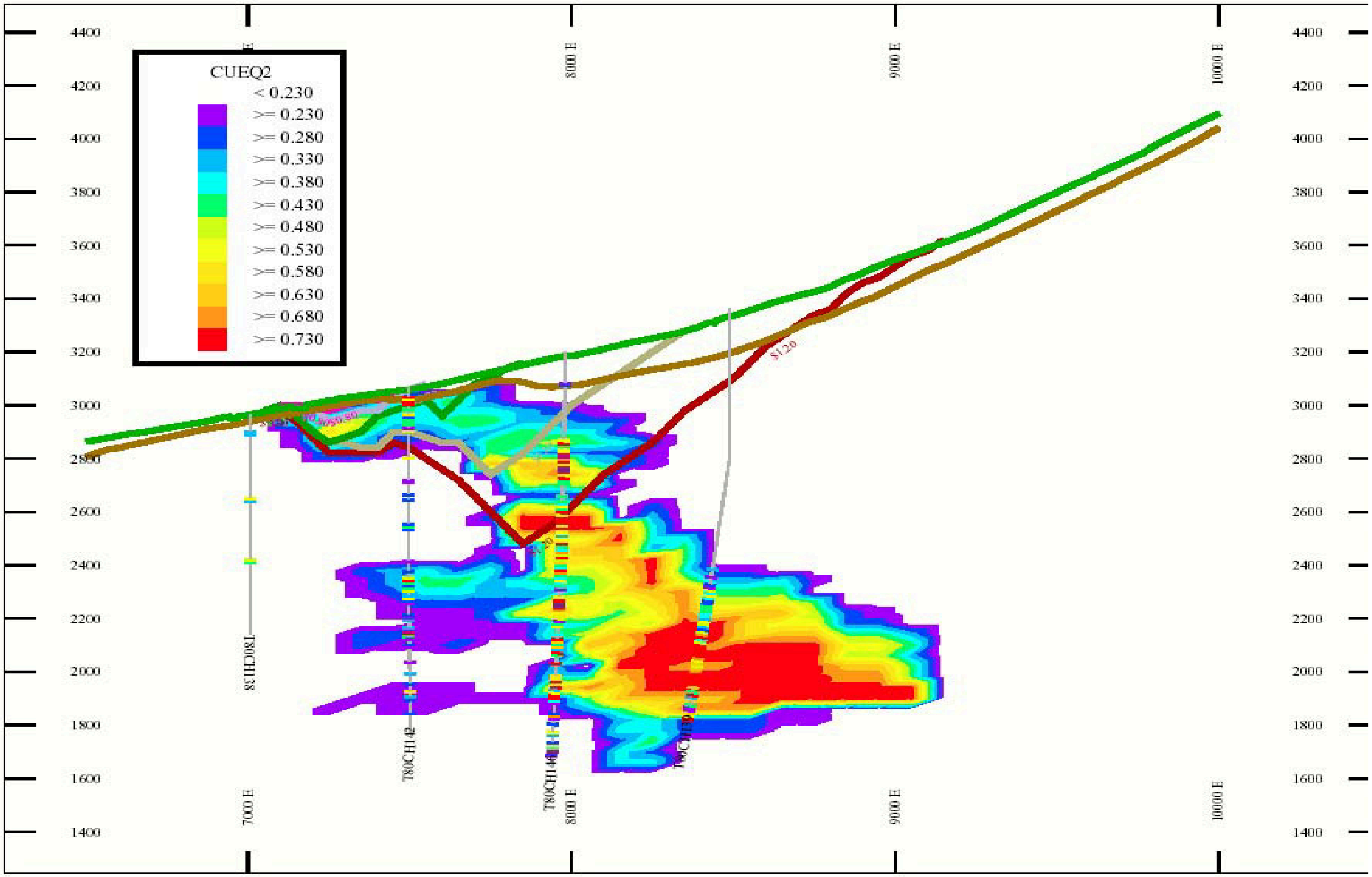
Cross Section at 25000N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



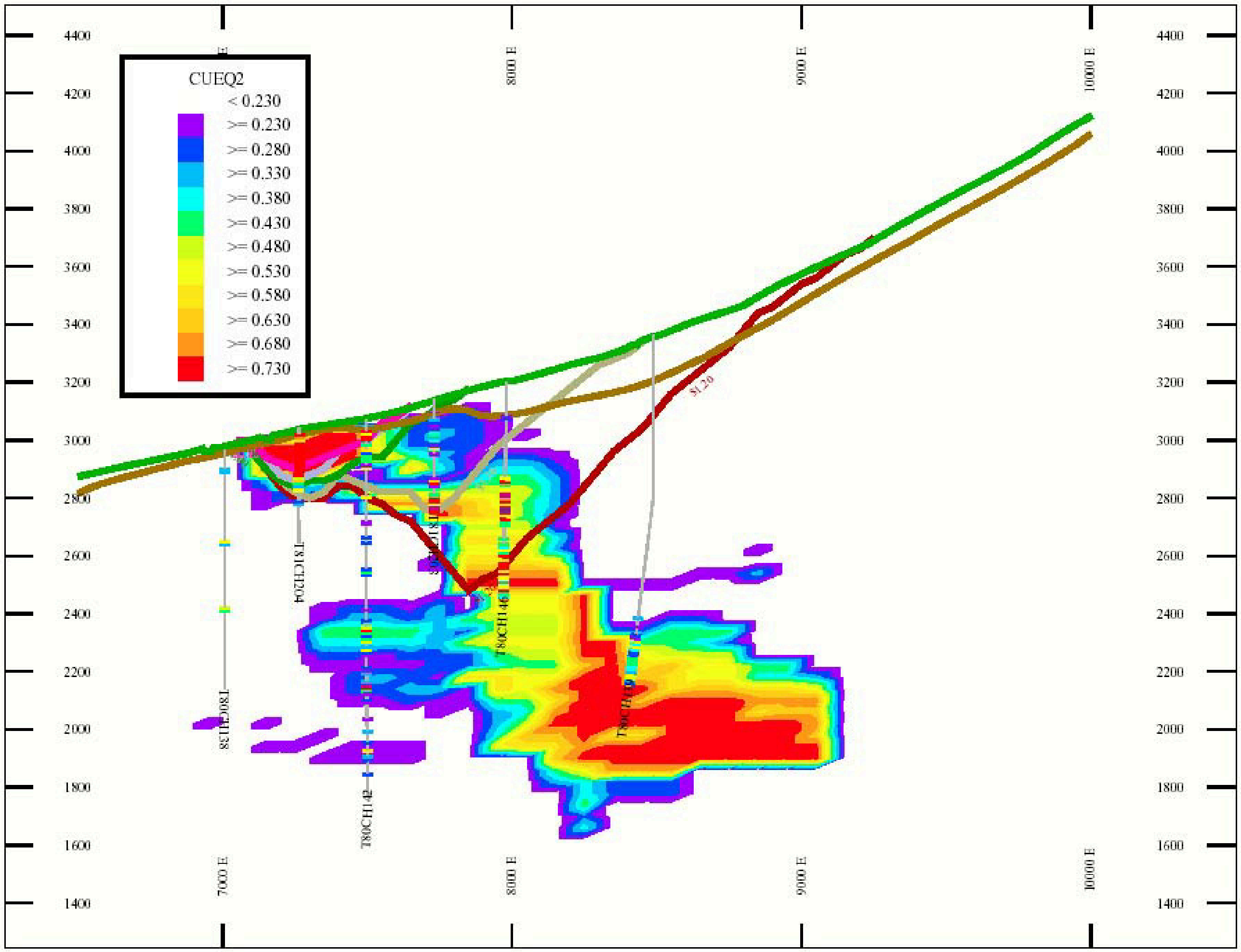
Cross Section at 27700N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



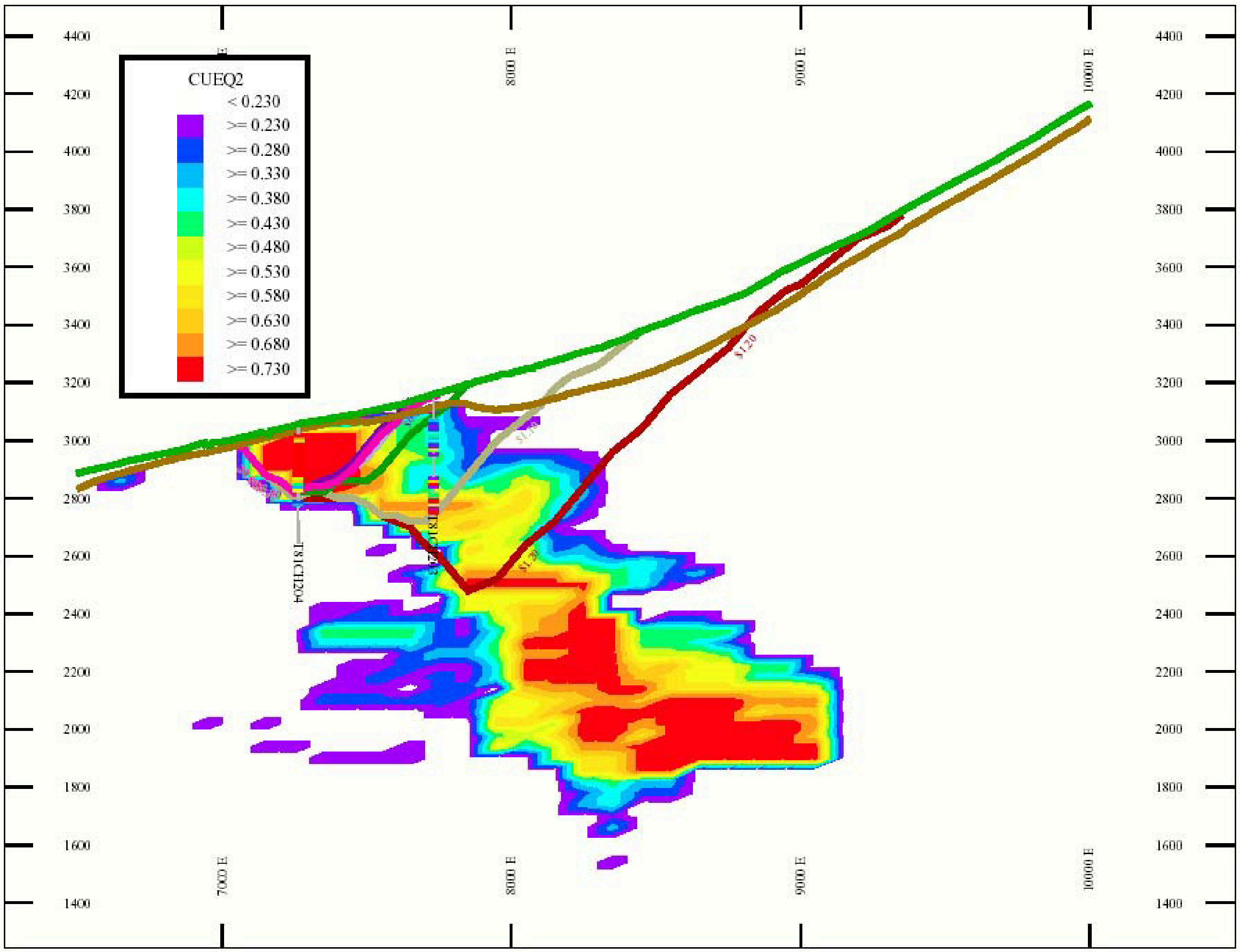
Cross Section at 27800N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



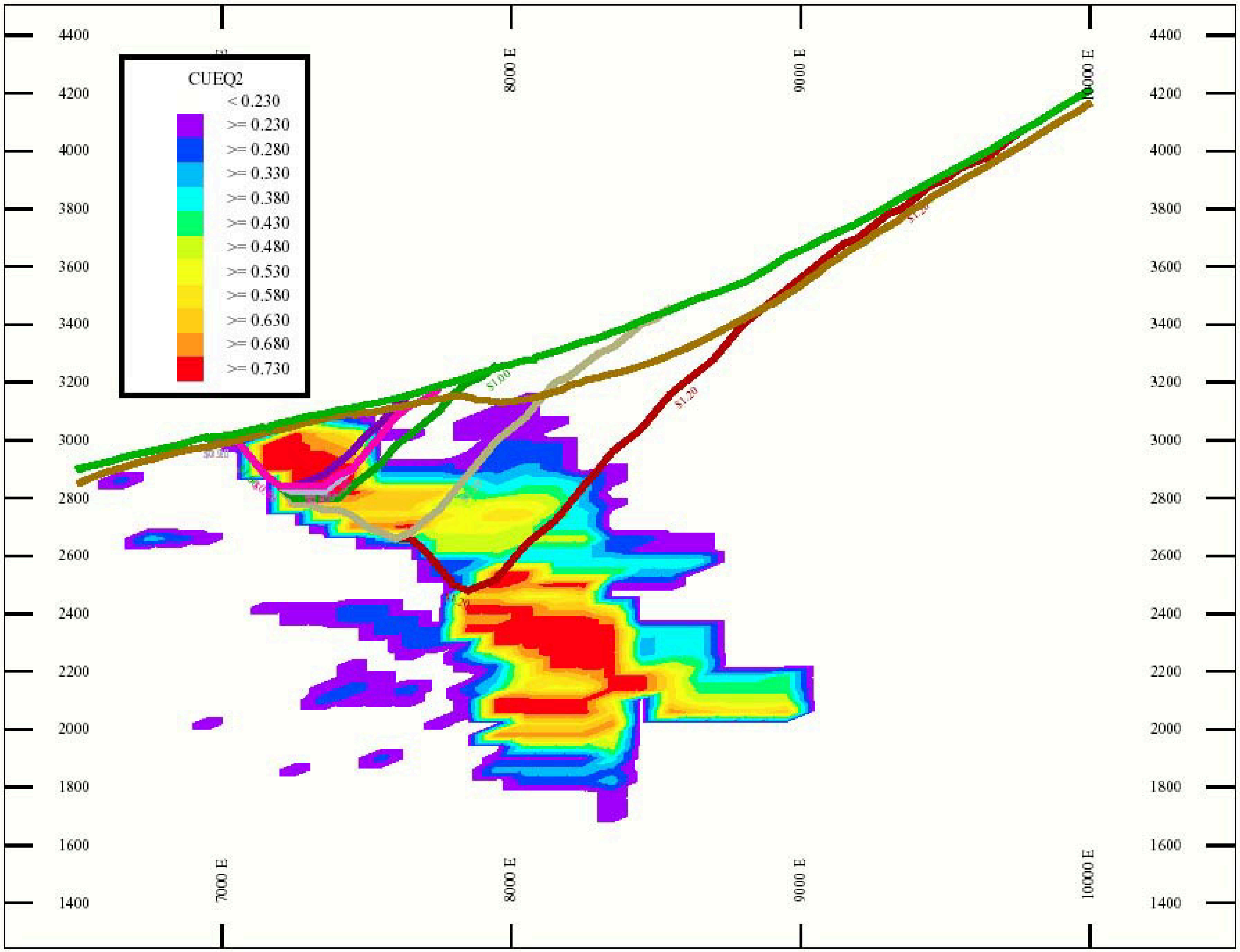
Cross Section at 27900N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



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

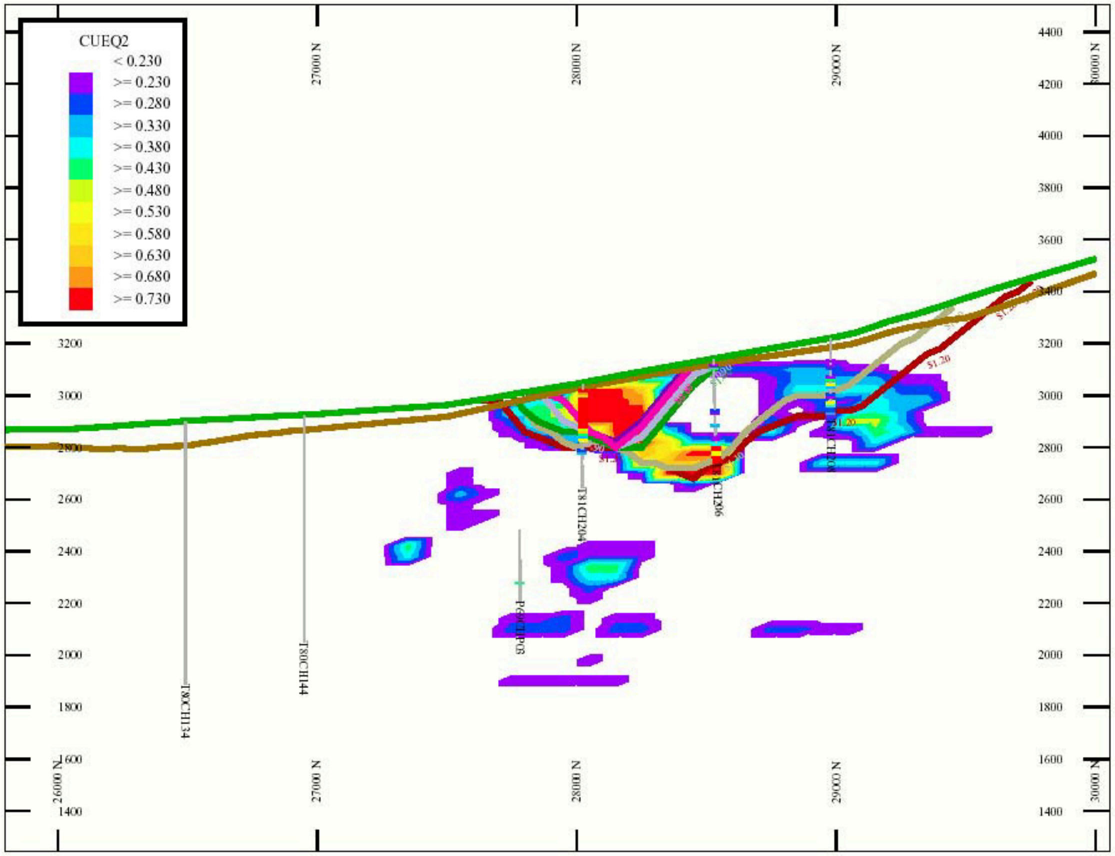


Cross Section at 28100N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

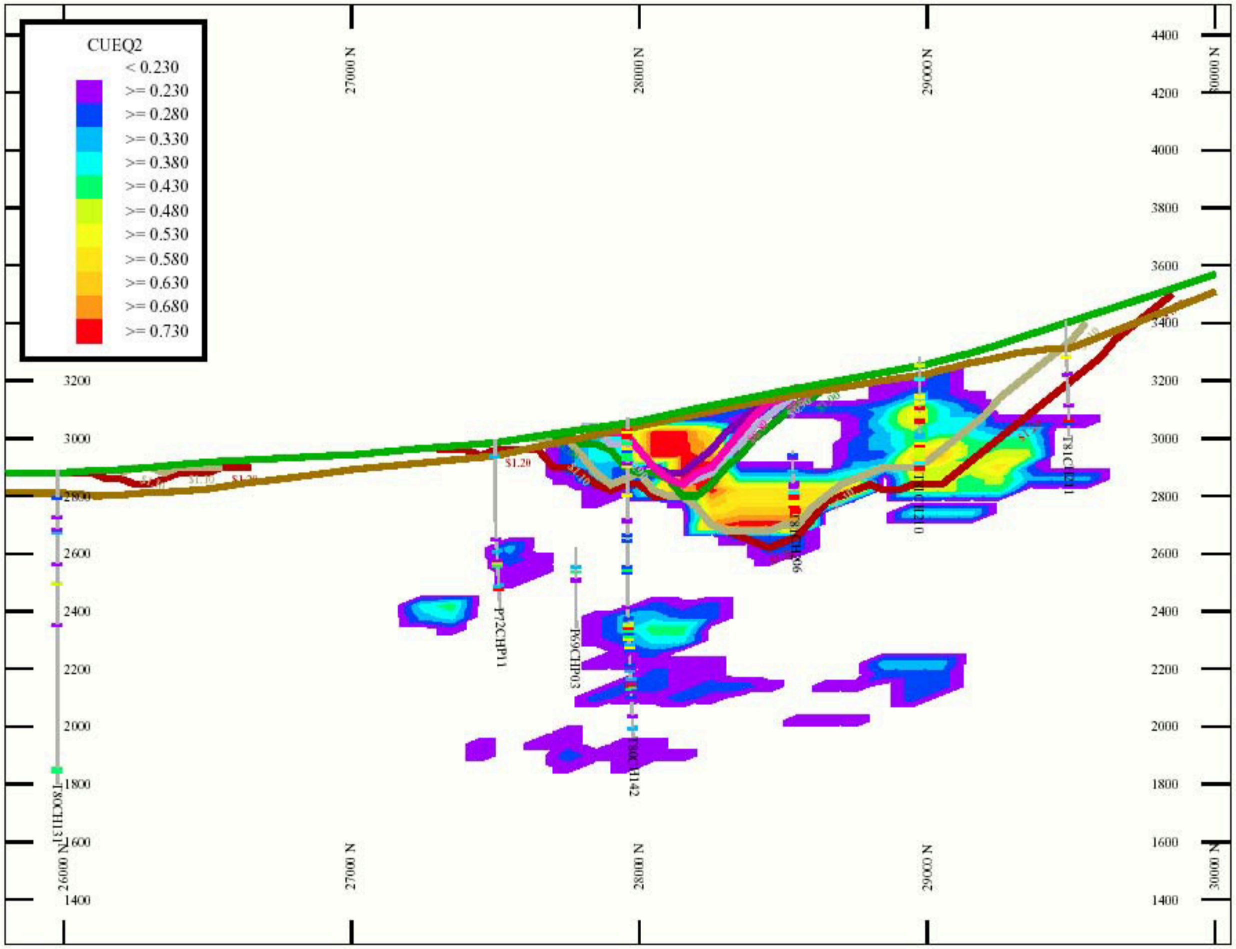


Cross Section at 28200N Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

APPENDIX 3
LONG SECTIONS



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



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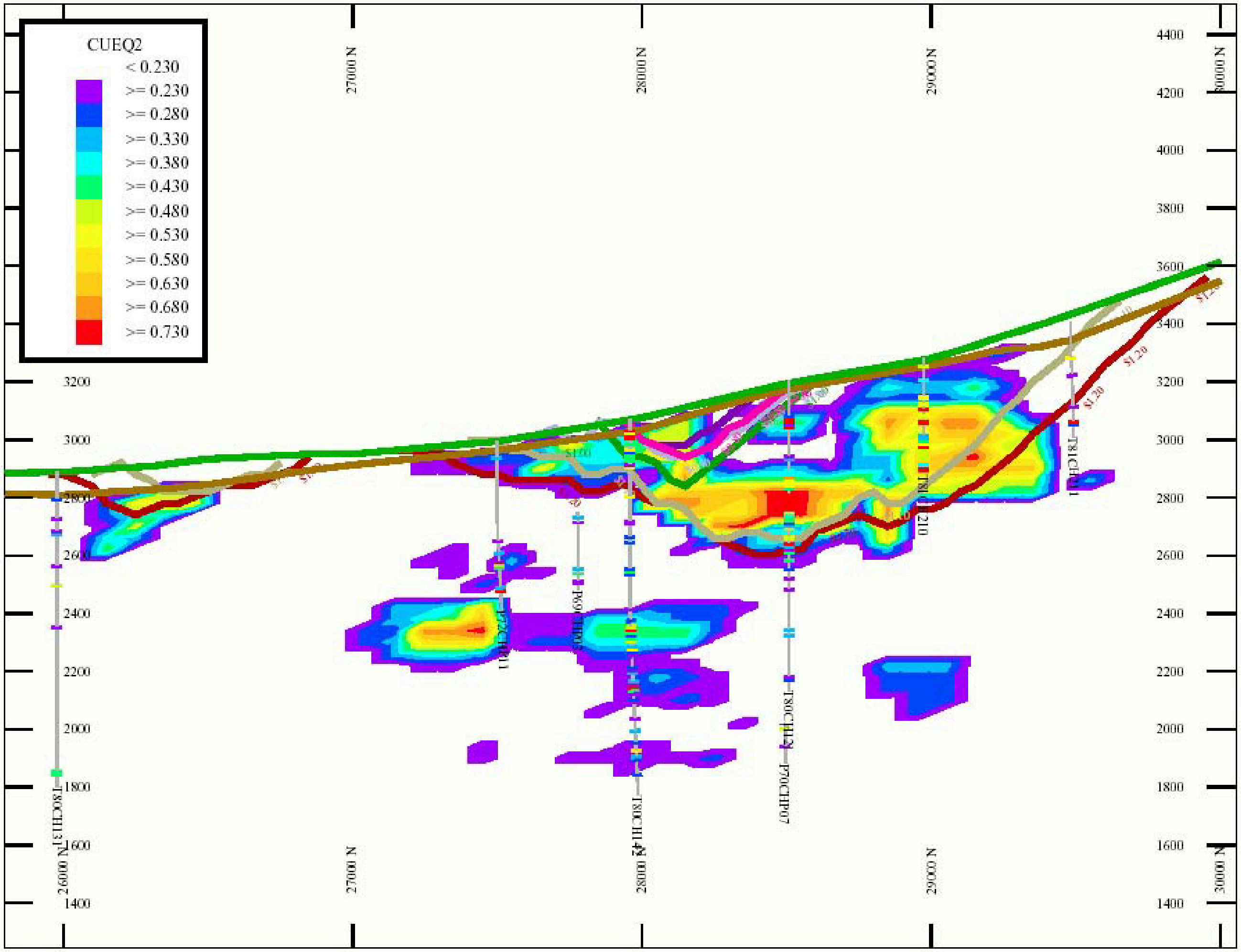

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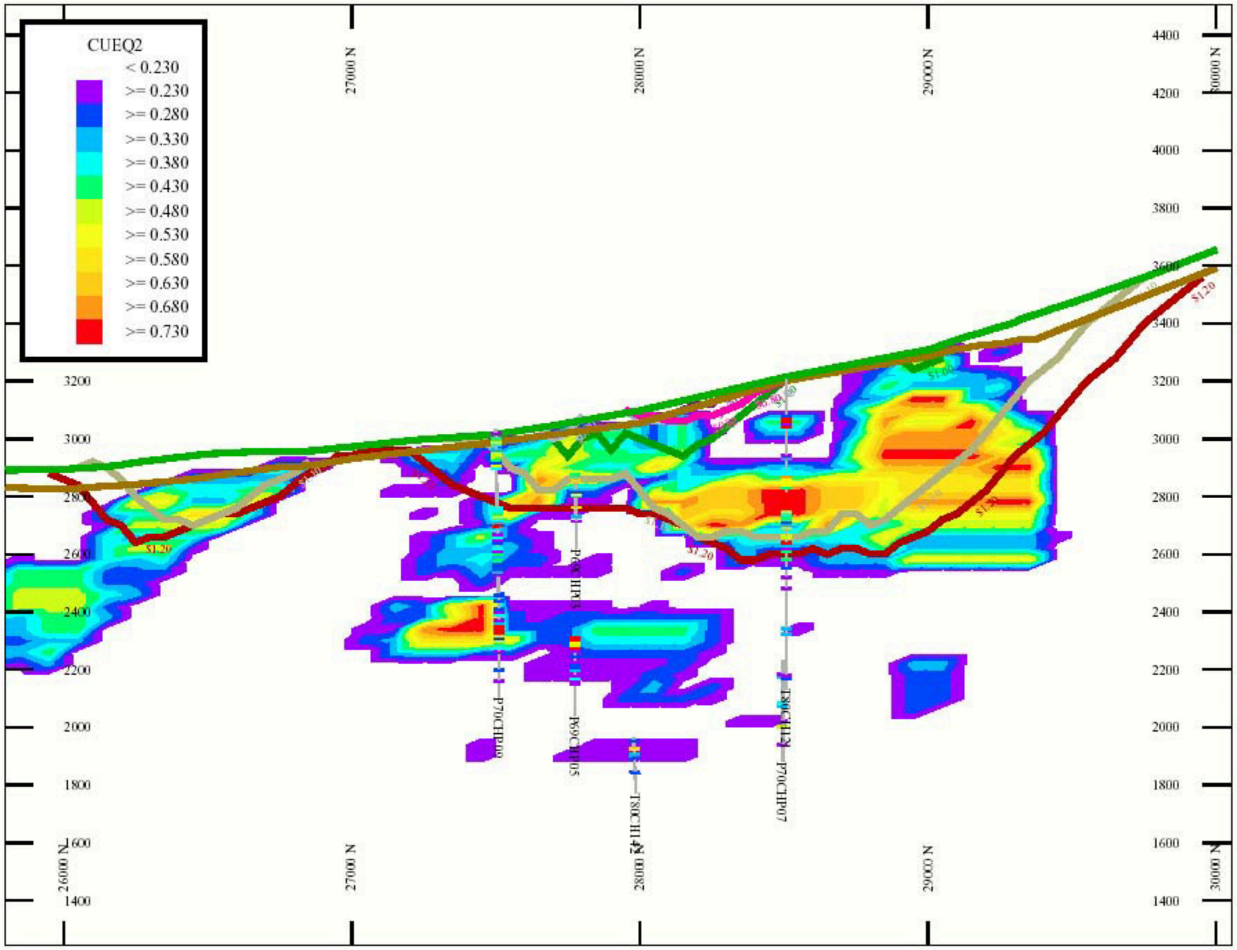
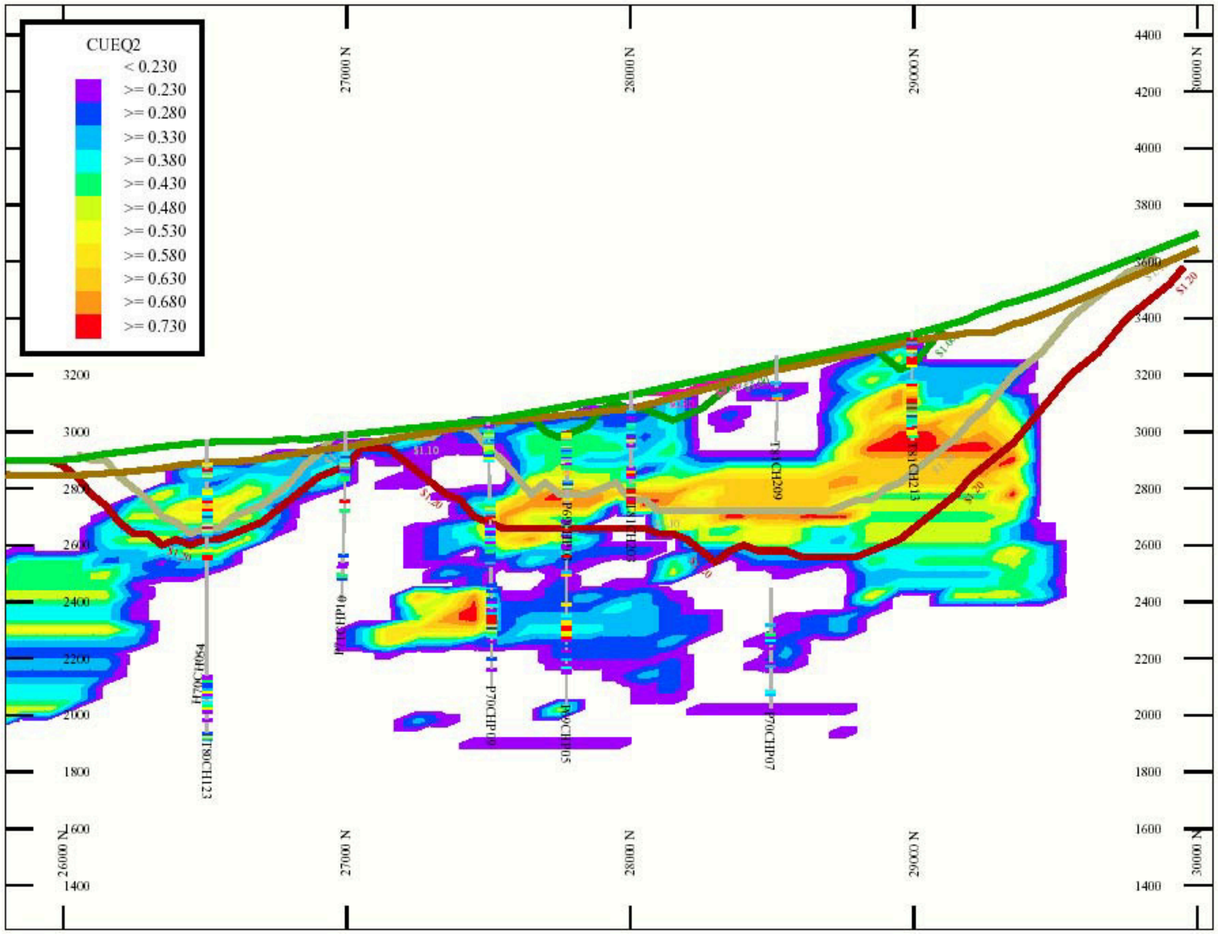
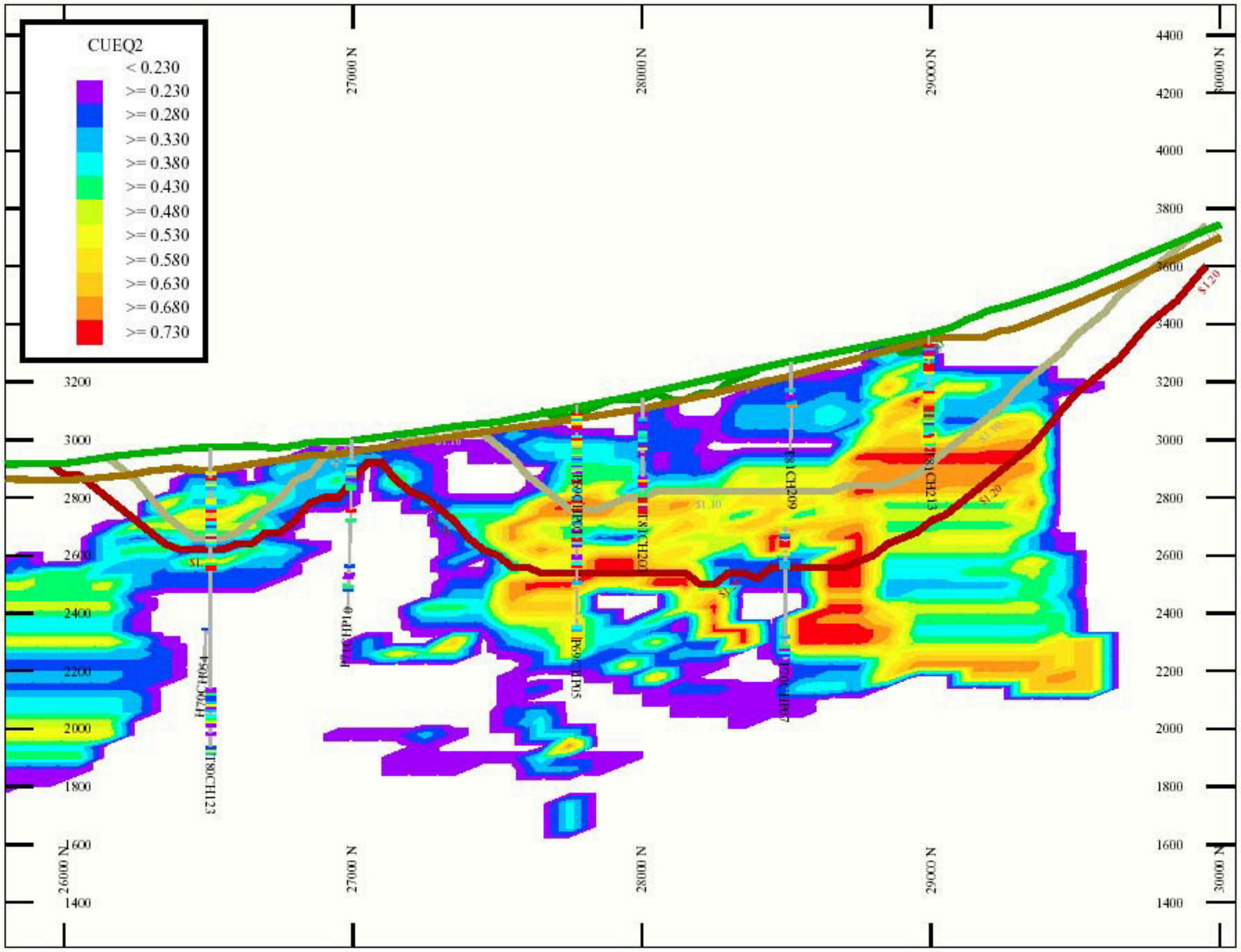


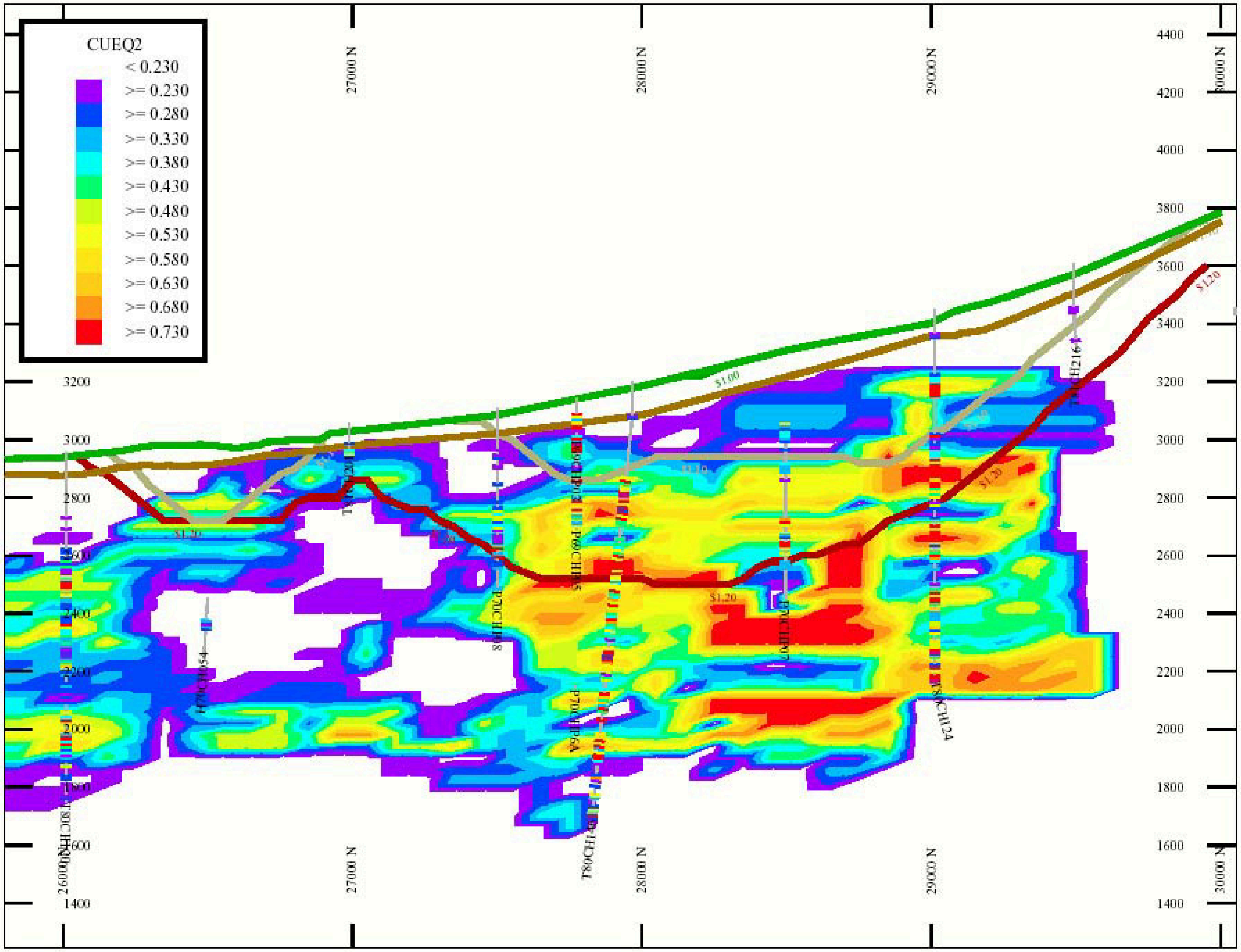
Long Section at 7600E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



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

File: 04PM76

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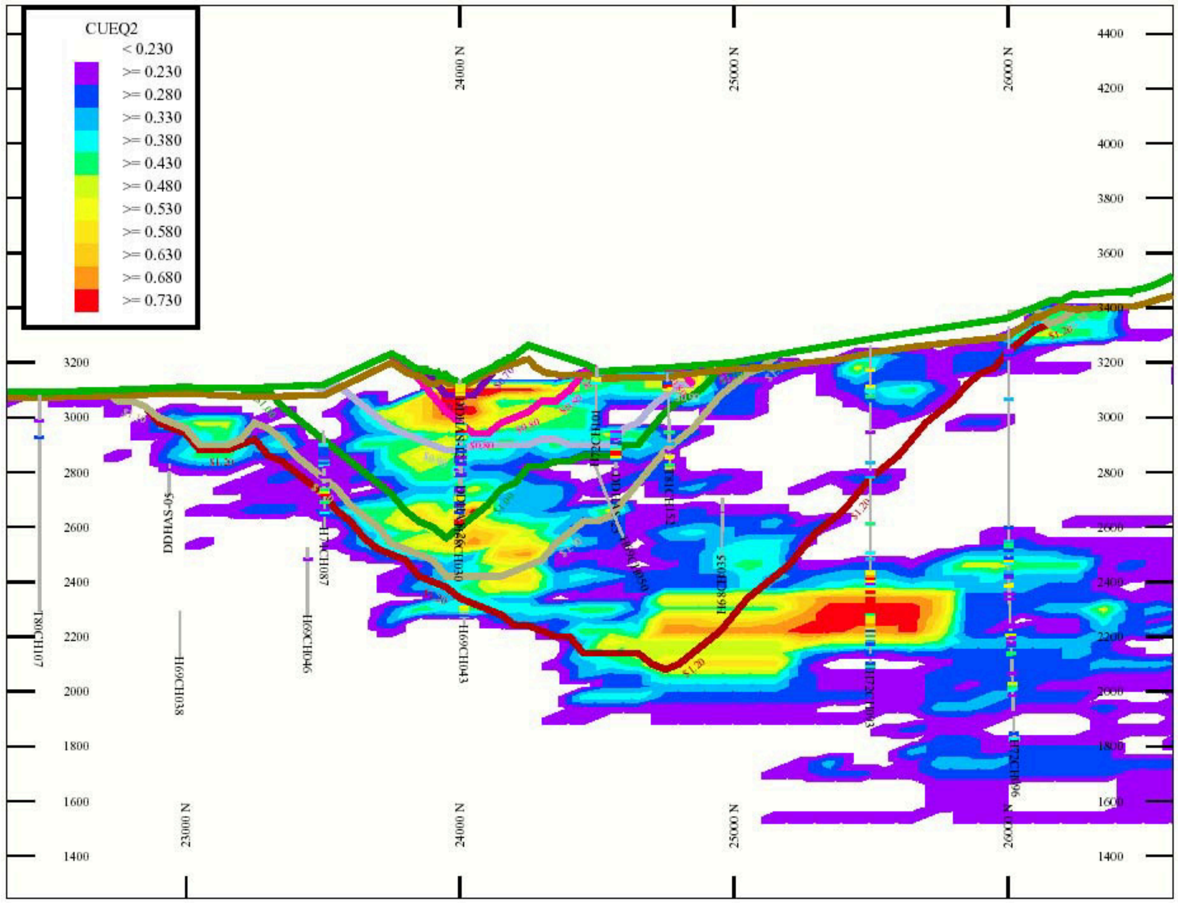
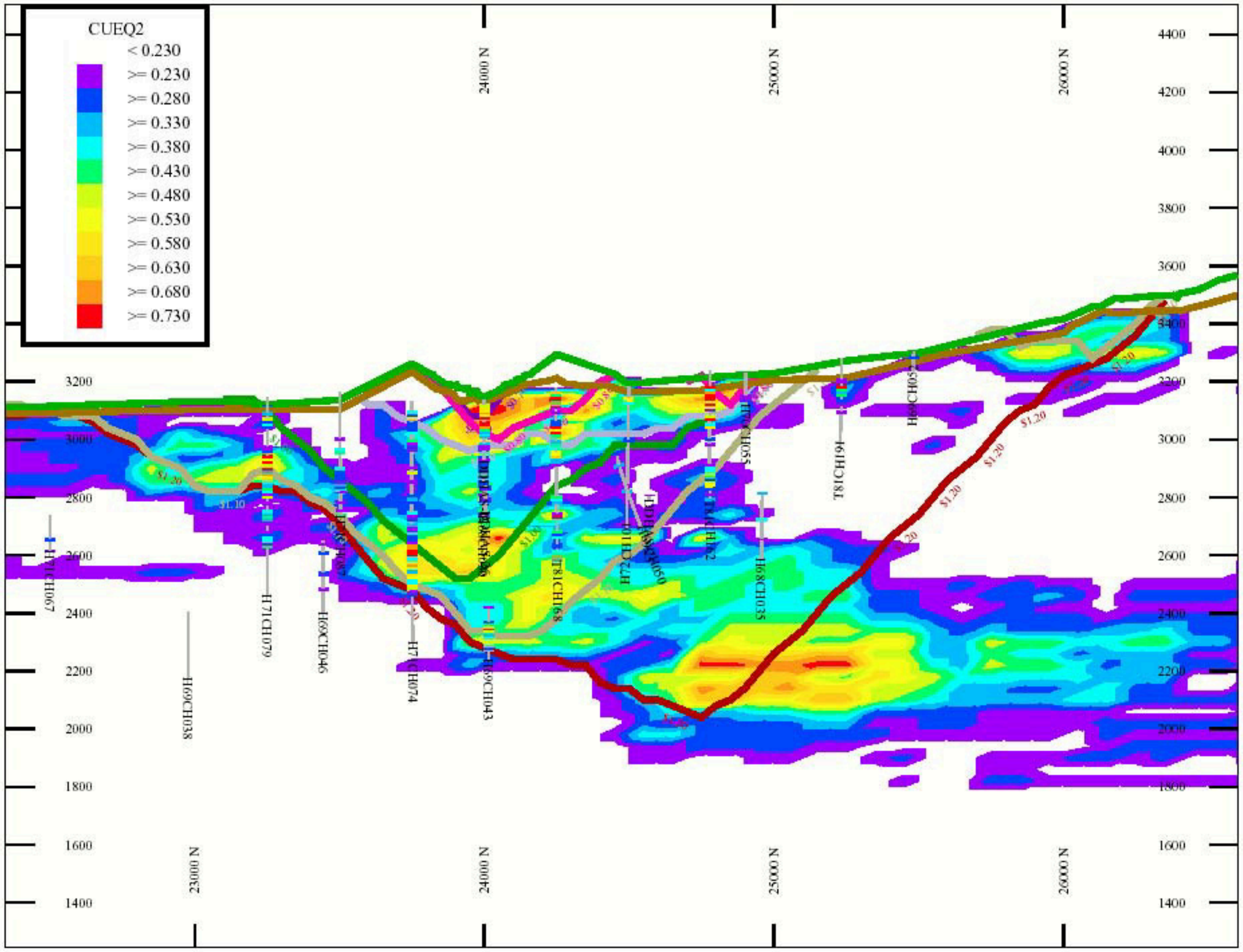
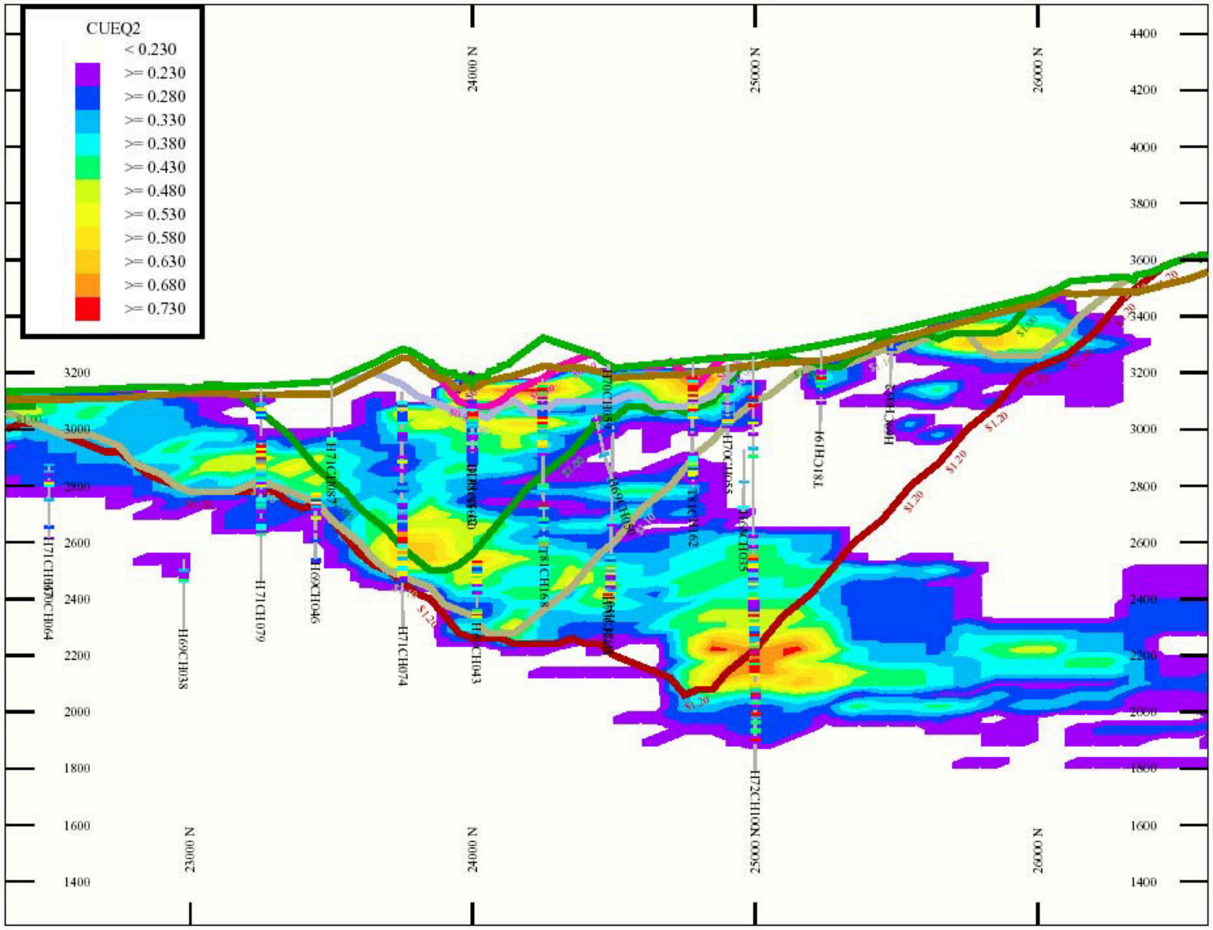


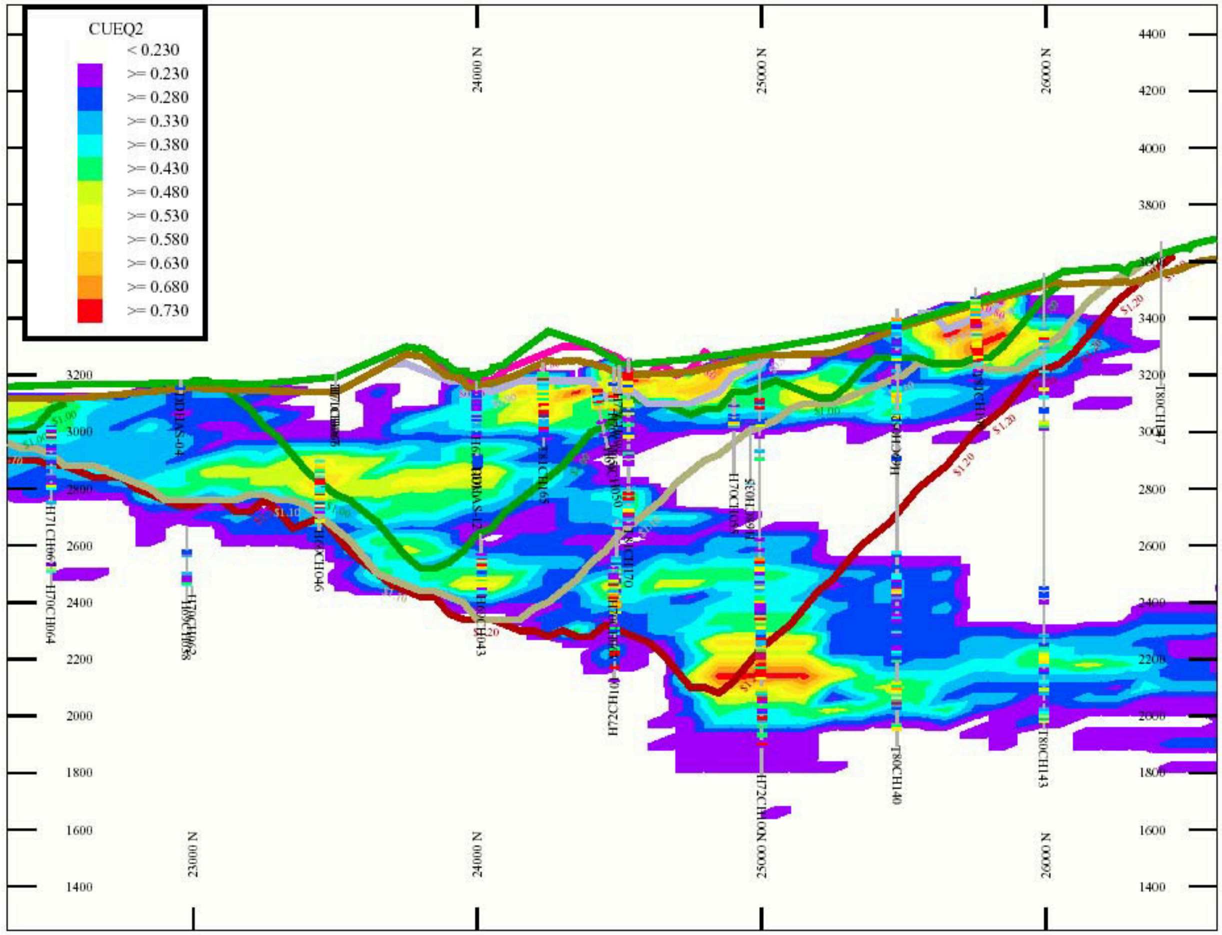
Long Section at 9600E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



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



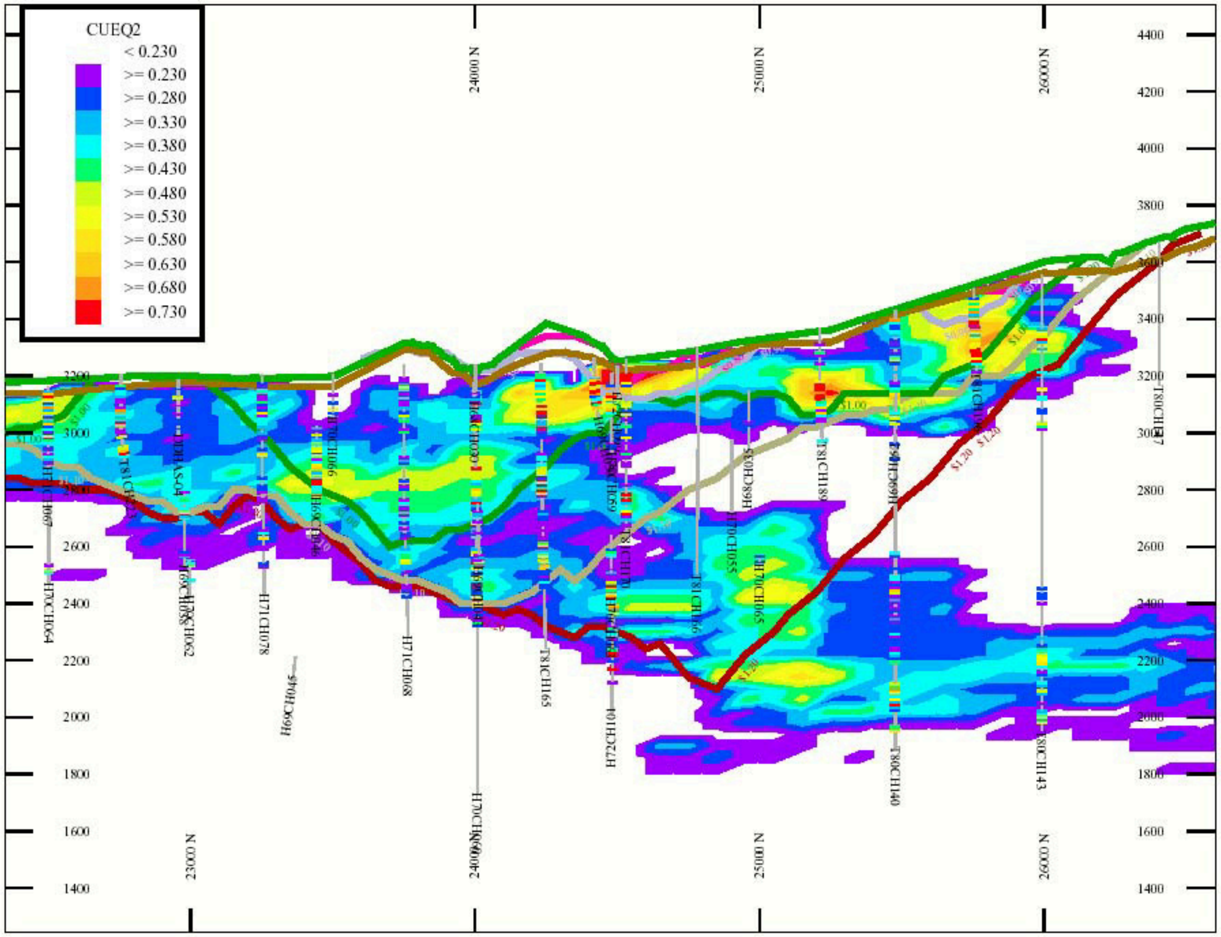
Long Section at 9800E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



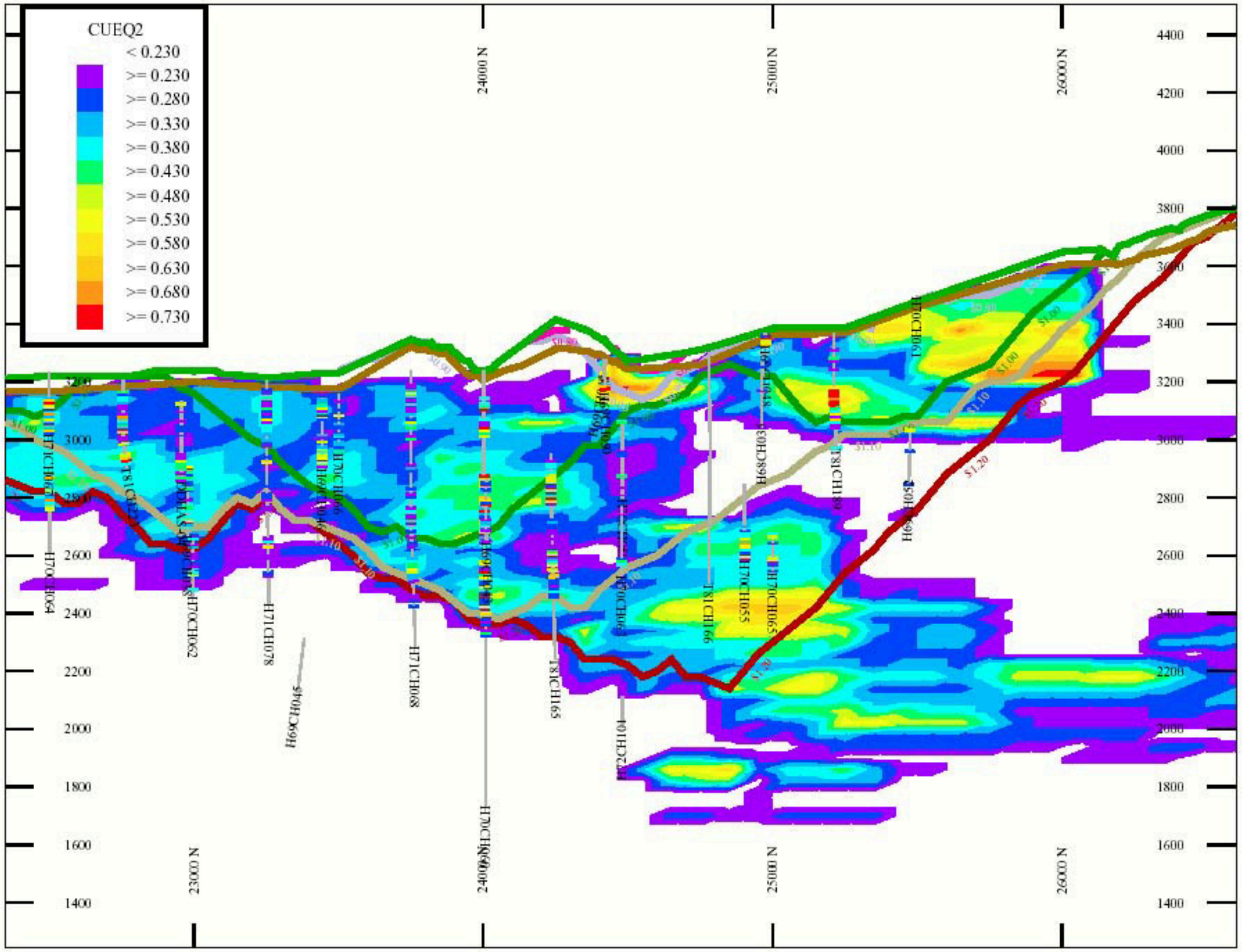
Long Section at 9900E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent

File: 04PM76

September 20, 2004



Long Section at 10000E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent



Long Section at 10100E Showing Optimized Pit Outlines and Grade Model with Net Copper Equivalent