Silver Coin Gold Project NI 43-101 Preliminary Economic Assessment Report

Stewart, British Columbia, Canada

Prepared for:

Pinnacle Mines, Ltd

Suite 1980
1075 West Georgia Street
Vancouver, BC,   V6E 3C9
Canada
Phone: (604) 688-9588
Fax: (778) 329-9361

Prepared by:

TETRA TECH

350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700
Fax (303) 217-5705

Tetra Tech Project No. 114-310920

December 30, 2009

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF APPENDICES

APPENDIX A: "Metallurgical Study on the Silver Coin Gold Project" by F. Wright Consulting Inc., January 8, 2009

APPENDIX B: Metso Test Report, prepared for Pinnacle Mines dated October 21, 2009

1.0 SUMMARY

In October 2009 Pinnacle Mines Ltd ("Pinnacle" or "the company") commissioned Tetra Tech of Golden, Colorado to conduct a Canadian National Instrument 43-101-compliant Technical Report and Preliminary Economic Assessment (PEA) on the company's majority controlled Silver Coin Gold Project. The project is located 25 Km north of Stewart, British Columbia and is centered at 130° 02' west longitude and 56° 06' north latitude. It is a joint venture with Mountain Boy Minerals Ltd. ("Mountain Boy Minerals") and includes 26 contiguous claims with a net area of 1,255 Ha. Pinnacle owns 70% of the project in areas with known mineralization and together with co-owner Mountain Boy Minerals, 55% of some of the land lying outside of the known gold resource.

1.1 Historic Drilling

Historic drilling, prior to Pinnacle involvement in 2005 to 2008, totals 422 drillholes on the property for 37,401 meters. This historic drilling included 293 underground drillholes totaling 17,500 meters from the period 1988-1994.

Pinnacle drilled 292 surface drillholes totaling 48,443 meters between 2005 and 2008. There was no drilling completed on the project in 2009.

1.2 Geology and Mineralization

The geology of the property is dominated by Triassic-Jurassic basin filling sediments and volcanic rocks of the Stuhini Group, Hazelton Group and Bowser Lake Group. These rocks have been metamorphosed to greenschist facies and have been intruded by plutons of both Mesozoic and Cenozoic age. North-south faulting controls the distribution of the rocks and certain faults are critical in defining the location of gold mineralization. In the area of the deposit, alteration is intense and has complicated both surface and underground interpretation of the geology.

Mineralization on the property occurs above a major north-south west-dipping listric fault and defines a crudely cylindrically shaped body of mineralization in highly altered and stockwork quartz-veined, Jurassic aged, andesitic Hazelton Group volcanic rocks. Historically, predecessor companies mined a zone of gold-bearing sulfide mineralization. The current mineralization concept involves an early event of Kuroko type sea floor massive sulfide mineralization that was remobilized and enriched by later intrusive-related mineralization as the rocks were uplifted and accreted to the continent.

1.3 2004-2008 Pinnacle Exploration Drilling Program

After nearly 10 years of inactivity, Mountain Boy Minerals resumed active exploration of the property in 2004 by drilling 38 surface core drill holes. Most of these holes were drilled in the Main Breccia Zone to confirm and expand the known mineralization. In 2005, Pinnacle became involved on the property and continued active surface exploration through the 2008 season. During the 2004-2008 period, the two companies drilled a total of 324 surface core holes totaling 50,305 meters. The majority of these holes were drilled in the Main Breccia Zone to expand and confirm the known mineralization. A minor amount of drilling was done in the Terminus, West No Name Lake and Road Zones to continue exploration of these targets.

1.4 Resource Estimation

Tt completed an independent mineral resource and reserve estimate of the contained gold in the Silver Coin deposit. Several computer programs were used in this analysis. Geostatistics and resource estimation was done with MicroModel®. Additional statistical analysis was done with Statistica®, and Excel®. Three-dimensional wireframes of modeled faults and model visualization was done with GemCom software.

Tt calculated resources for the Silver Coin deposit using both current and historical data from trenches, surface drilling and underground drilling. Both the new and historical data was verified using the original assay certificates. Tt had the advantage to carefully critique the methodologies used by two earlier resources estimates. The 2007 Minefill resource estimate used grade-shell wireframes to constrain ordinary kriging. The 2008 Snowden resource estimate employed both grade-shell wireframes and mapped faults to constrain multiple indicator kriging. Tt agrees with these earlier estimators observation that geologic wireframes of lithology are suspect due to the complex and discontinuous three dimensional distributions of silification, brecciation and sulfidation. This complexity is a result of interpretations done by different geologists from at least five companies and the inherent complexity of the subsurface geology. Tt used Pinnacle's re-interpretation of subsurface faulting to constrain its estimate using multiple-pass ordinary kriging.

Geologic Modelling

The block model consists of blocks 10x10x5 m in dimension. The total model contains a potential of 131 rows, 121 columns, and 161 levels. The model has no rotation and is 1210m east-west by 1310m north-south by 805m high. A large percentage of the blocks are "air blocks" (i.e. above topography). Sample, composite and block grade labels are silver, gold, copper, lead and zinc. Respectively, they are denominated as: silver (AgG, cAgG, kAgG), gold (AuG, cAuG, kAuG), copper (Cu%, cCu%, kCu%), lead (Pb%, cPb%, kPb%), and zinc (Zn%, cZn%, kZn%).

Two subsurface faults are believed to act as a floor to mineralization, particularly gold. Blocks below the faults are coded as 99 (blue). Blocks above the first fault are coded as 1. To the north end of the deposit there seems to be a second splay to this surface lying below the first. The blocks above the second fault, but below the first fault have been coded as 2. Further analysis has shown there is no significant distinction in composites coded with rock codes 1 or 2. Hence, the two codes have been lumped together.

There is no coding for overburden, as a significant part of the Silver Coin resource essentially outcrops. The steep northern face of the deposit has zones of transported overburden where down-slope movement has resulted in local thicker accumulations of loose material. However, a review of the site suggests that the effects of overburden are generally negligible given a vertical block size of 5-meters.

Assay Database

The assay database is comprised of drillhole and trench assay values. Total count of all entries is 774; 412 Surface drillholes, 287 under-ground drillholes and 75 trenches. The original assay values have sampling intervals that vary from 1 to three meters.

The average gold grade for surface drill holes (SDH) is 1.25 Au g/t, while for underground drill holes (UDH) it is 1.76 gAu/t. There is an apparent enhancement of the average gold grade of almost a half a gram between surface and underground data.

Compositing

Statistics were developed from log transformed assay data. In all cases, the statistical review shows that the compositing appears valid and did not inappropriately distort the underlying assay distribution. Gold composite data appears to follow a unimodal, somewhat triangular distribution. Silver composite data appears to follow a more classic lognormal distribution. The low composite grades for copper, lead and zinc, truncated by detection limits, apparently distort the lower end of the bell-shaped lognormal distribution. This truncation effect is most noticeable for lead and zinc.

Geostatistical Analyses

Kriging was done using ordinary kriging. The parameters used for the kriging analysis are summarized in FIGURE 1-1.

Matching Codes			Anisotropy				MIF Search Ranges					Variogram Parameters						
xAu (Gold Value Composited to 5m, cut at 30 g/t)																		
Composite Codes	Block Codes	Zone Name	Axis	Anisotropy Axis Length (m)	Anisotropy Rotation	Type[3]	Resource Class[4]	Resource Code[3]	Maximum Search Range	Number Closest Pts /Max Pts Single Drillhole	Min Pts Required to Estimate	Rotation	Length	Nugget[1]	Nested	Model Type[4]	Sill[1]	Range (m)
1&2	1&2	Above Faults 1&2	Primary	35	90	Az	M	1	11	15/99	2	90	100	6.0	1	Sph	4	10
			Second	35	45	Dip	I	2&3	20	15/99	2	45	100		2	Sph	3	40
			Tertiary	15	0	Tilt	F	4&5&6	50	15/99	2	0	40		3	Sph	2	120
99	99	Below Faults	Primary	35	90	Az	M	1	11	15/99	2	90	100	6.0	1	Sph	4	10
			Second	35	45	Dip	I	2&3	20	15/99	2	45	100		2	Sph	3	40
			Tertiary	15	0	Tilt	F	4&5&6	50	15/99	2	0	40		3	Sph	2	120
xAg, cCu, cPb, cZn uses the gold variogram and search parameters and produces no classified results																		
1&2	1&2	Above Faults 1&2	Primary	35	90	Az						90	100	6.0	1	Sph	4	10
			Second	35	45	Dip						45	100		2	Sph	3	40
			Tertiary	15	0	Tilt			50	15/99	2	0	40		3	Sph	2	120
99	99	Below Faults	Primary	35	90	Az						90	100	6.0	1	Sph	4	10
			Second	35	45	Dip						45	100		2	Sph	3	40
			Tertiary	15	0	Tilt			50	15/99	2	0	40		3	Sph	2	120

Notes
1 Unitize General Relative (All variogram structures are transformed to relative variograms from log variograms)
2 Kriging Error is used to adjust preliminary class 1,3,5 to a final final resource class of 1,2,3,4,5&6
3 Az=Azimuth is clockwise (CW) from North, Dip is positive when downward, Tilt rotates CW around primary axis.
4 Sph=Spherical, Lin=Linear, Exp=Exponential, Gau=Gaussian
5 M=Measured, I=Indicated, F=Inferred

FIGURE 1-1: SUMMARY OF THE KRIGING PARAMETERS USED IN RESOURCE ESTIMATION

Resource Classification

Resource classification was accomplished by combining a number of different variables into the designation of resource class. Drillhole spacing, minimum and maximum numbers of drillholes and composites, and kriging error were all used in the classification system. Kriging is a mathematical algorithm that has many similarities to regression. For every estimate, kriging also produces a kriging error. The kriging error embodies a quantitative measurement of the quality of the kriging estimate. It is much more sophisticated than a simple measure of sample spacing. Kriging error takes into account both the anisotropy of the deposit, hence the direction that samples are from the block and whether there are areas that are over sampled (i.e. clustering of data). This combination of various components results in a robust method of resource classification because it embodies the strengths of various components; i.e. special relationships of the data, quantities of the data, and a measure of confidence in the estimated grade.

Estimated Resources

TABLE 1-1 shows the Silver Coin resources tabulated by gold grade and resource classification of measured, indicated and inferred. It is Tt's opinion that the reported mineral classes comply with current CIMM definitions for each mineral class. The **BOLDED** line indicates the base case cutoff grade scenario.

TABLE 1-1: SILVER COIN TOTAL CLASSIFIED RESOURCES
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

MEASURED RESOURCES

ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	8,895	1.28	7.04	0.29	365	2,012	55,967
ALL	0.50	5,957	1.73	8.16	0.35	331	1,562	46,569
ALL	0.75	4,308	2.16	8.96	0.40	299	1,241	38,246
ALL	1.00	3,219	2.59	9.64	0.44	268	997	31,140
ALL	1.25	2,505	3.01	10.27	0.47	243	827	26,017
ALL	1.50	2,052	3.38	10.93	0.50	223	721	22,723

INDICATED RESOURCES

ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	18,385	1.02	5.99	0.20	602	3,544	82,522
ALL	0.50	11,811	1.38	6.92	0.25	526	2,627	65,174
ALL	0.75	8,009	1.75	7.54	0.28	451	1,942	49,527
ALL	1.00	5,608	2.13	8.13	0.30	384	1,465	37,511
ALL	1.25	4,073	2.51	8.56	0.32	329	1,121	28,949
ALL	1.50	3,048	2.90	9.17	0.35	284	898	23,297

MEASURED + INDICATED RESOURCES								
ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	27,279	1.10	6.33	0.23	967	5,556	138,441
ALL	0.50	17,767	1.50	7.33	0.29	857	4,189	111,750
ALL	0.75	12,317	1.89	8.04	0.32	749	3,184	87,762
ALL	1.00	8,827	2.30	8.68	0.35	652	2,462	68,635
ALL	1.25	6,578	2.70	9.21	0.38	572	1,949	54,962
ALL	1.50	5,101	3.09	9.88	0.41	507	1,620	46,029

INFERRED RESOURCES								
ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)				Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	49,189	0.76	6.60	0.22	1,209	10,433	243,019
ALL	0.50	24,861	1.17	8.50	0.28	937	6,792	154,999
ALL	0.75	15,343	1.52	8.43	0.30	750	4,158	99,920
ALL	1.00	10,380	1.84	9.47	0.33	612	3,160	76,363
ALL	1.25	6,787	2.22	10.89	0.38	484	2,375	57,217
ALL	1.50	5,031	2.51	12.04	0.41	407	1,948	45,508

Virtually the entire known resource is located on 22 claims of the total 26 claims that make up the Silver Coin project. Pinnacle owns 70% and Mountain Boy Minerals owns 30% of these 22 claims and the known resource. Pinnacle has an option to acquire an additional 10% of the 22 claims (for a total of 80%) by spending CDN$2,000,000 on exploration expenses on or before June 30, 2014. The remaining four INDI claims lie on the eastern edge of the resource and Pinnacle owns 28.05% of these four claims with Mountain Boy owning an additional 26.95% for a total of 55%. Nanika Resources Inc. owns the balance of 45%.

1.5 Potentially Mineable Resources

Silver Coin contains no mineral reserves as defined by CIMM standards. All categories of the estimated mineral resources - Measured (M), Indicated (I), and Inferred (I), have been used in the determination of potentially mineable mineral resources. All categories have been used in developing production schedules and preliminary cash flow analyses.

The potentially mineable resources are developed from open-pit mining scenarios. The potentially mineable resource estimates were derived from 3D grade and geologic block models developed by Tt as described in SECTION 17.

Whittle Pit Design Parameters

To guide design of an ultimate pit and to determine the best extraction sequence, Tt utilized floating cones and the Whittle algorithm to establish guides to mineable shapes within the mineral resource block model. The ordinary kriging estimate of total gold in the model was imported to Gemcom's® Whittle® mine optimization software. Estimates for mine and processing costs were prepared from a comparison of actual and handbook costs for similar operations.

For the Silver Coin Gold Project, two potential operations are being considered. One that involves creating a bulk sulfide concentrate that is shipped to Asia for smelting and one that involves flotation followed by cyanidation on site that produces a precious metals dore. TABLES 1-2 and 1-3 list the input parameters used for the LG cone runs for these two potential development scenarios. No additional constraints such as property boundaries or existing infrastructure locations were applied to the preliminary Whittle run. The average achievable pit slope was estimated at 45°. Slope measurements on historical benches are approximately 45° in several areas of the existing pit. The gold price is based on the 3-year trailing average gold price.

TABLE 1-2: WHITTLE LG PARAMETERS – ALL FLOTATION SCENARIO
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

Parameter	Units	Value
Average Pit Slopes	Degrees	45
Metal Price (3-year average)		
Gold	US$/t. oz	833
Silver	US$/t. oz	14.24
Metal Produced		
Gold	%	95
Silver	%	88
Mining Cost	$US/tonne mined	2.55
Processing Cost	$US/tonne milled	7.42
Freight & Refining	$US/ounce gold	25
General & Administrative Costs	$US/tonne milled	1.33
Environmental & regulatory Costs	US$/tonne milled	0.25

TABLE 1-3: WHITTLE LG PARAMETERS – FLOTATION - CYANIDATION SCENARIO PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009		
Parameter	**Units**	**Value**
Average Pit Slopes	Degrees	45
Metal Price (3-year average)		
Gold	US$/t. oz	833
Silver	US$/t. oz	14.24
Metal Produced		
Gold	%	88
Silver	%	60
Mining Cost	$US/tonne mined	2.55
Processing Cost	$US/tonne milled	8.42
Freight & Refining	$US/ounce gold	10.00
General & Administrative Costs	$US/tonne milled	1.33
Environmental & regulatory Costs	US$/tonne milled	0.25

TABLES 1-4 and 1-5 summarize the results of the two Whittle LG scenarios. For this PEA, the ore production rate was set at 3,500,000 ore tonnes per year or approximately 10,000 ore-tonnes per day. A one-year build up is expected with Year one ore production set at 3,500,000 tonnes and 6,354,000 tonnes of waste. Subsequent years will continue to produce 3,500,000 ore tonnes through year 15 and have waste tonnes dropping to approximately 2,000,000 tonnes in year 15.

TABLE 1-4: POTENTIALLY MINEABLE RESOURCES – ALL FLOTATION SCENARIO PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009						
Ore Tonnes	**Avg. Metal Grades**			**Waste Tonnes**	**Total Tonnes**	**Stripping Ratio**
('000)	**Au (g/t)**	**Ag (g/t)**	**Zn (%)**	**('000)**	**('000)**	**(W:O)**
54,173	0.99	7.23	0.27	65,786	119,959	1.21:1

TABLE 1-5: POTENTIALLY MINEABLE RESOURCES – FLOTATION - CYANIDATION SCENARIO PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009						
Ore Tonnes	**Avg. Metal Grades**			**Waste Tonnes**	**Total Tonnes**	**Stripping Ratio**
('000)	**Au (g/t)**	**Ag (g/t)**	**Zn (%)**	**('000)**	**('000)**	**(W:O)**
42,840	1.13	7.82	.30	55,808	98,649	1.3:1

1.6 Mineral Processing and Metallurgical Testing

A scoping-level metallurgical program was conducted on selected drill core from the Silver Coin Gold Project during the period of 2005 to 2009. Laboratory studies were primarily performed by Process Research Associated Ltd. ("PRA") under the supervision of Mr. Frank Wright. This program investigated several different process routes for the recovery of the contained gold and silver values, including:

- Flotation
- Whole-ore cyanidation
- Cyanidation of flotation concentrates

Based on this work, the metallurgical data show that the most likely process routes for the Silver Coin Gold Project ore include:

- All Flotation
- Flotation followed by cyanidation of the flotation concentrate.

The first option does not require the use of cyanide and is considered the base-case process route due to concerns regarding the use of cyanide at the project site. This option would result in the production of a low grade flotation concentrate requiring shipment to an off-site smelter. The second option involves the use of cyanide and would result in the production of a readily marketable gold-silver dore' product at site.

1.7 Cash Flow Analyses

A cash flow analysis was developed for the mining the measured, indicated and inferred resources currently defined at Silver Coin and included the following input parameters:

Cash flow analyses was developed for the mining and processing the measured, indicated and inferred resources currently defined at Silver Coin. Both the all-flotation and flotation-cyanidation process alternatives were evaluated and included the following input parameters:

- Gold price at US$850 per ounce and silver price at US$14.25 per ounce
- All-flotation process gold recovery at 95 percent and silver recovery at 88 percent
- Flotation-cyanidation process gold recovery at 88 percent and silver recovery at 60 percent
- Mine operating cost at $2.31per tonne mined
- Process operating cost at $6.27 per tonne ore for the all-flotation process alternative and US$7.48 per tonne processed for the flotation-cyanidation process alternative.
- G & A at US$1.33 per tonne ore processed
- Concentrate transport and smelting costs were based on the following:
    - Trucking and port handling – US$5.00 per tonne of concentrate
    - Ocean freight – US$60.00 per tonne of concentrate
    - Smelter treatment charge – US$200 per tonne of concentrate
    - Gold refining charge – US$6.00 per oz.
    - Silver refining charge – US$0.50 per oz.

TABLE 1-6 provides a cash flow summary for the project based on processing the ore by the flotation-cyanidation process alternative. This cash flow indicates a before tax net present value (NPV) of US$58.3 million for the project at a 10 percent discount rate, and assumes 100 percent equity and a constant 2009 US dollar. TABLE 1-7 summarizes the sensitivity of the Net Present Value ("NPV") of the projected cash flows at discount rates of 5%, 8%, 10% and 12% for variation in Capex, Opex and gold price.

TABLE 1-8 provides a cash flow summary for the project based on processing the ore by the all-flotation-process alternative. This cash flow indicates a before tax net present value (NPV) of a negative US$82 million for the project. This economics of the all-flotation alternative is negatively impacted by concentrate transport and smelting charges. TABLE 1-9 summarizes the sensitivity of the Net Present Value ("NPV") of the projected cash flows at various discount rates.

TABLE 1-6: CASH FLOW FOR FLOTATION - CYANIDATION PROCESS ALTERNATIVE
PINNACLE MINES LTD - SILVER COIN PROJECT
December 2009

			PP 2	PP 1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Totals
PRICE SCHEDULE	$/oz Au				$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00		
	$/oz Ag				$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25		
MINE PRODUCTION SCHEDULE																			
ORE	tons (000s)				3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	840		42,840
	Au	g/t			0.83	0.90	0.95	1.02	1.14	1.22	1.17	1.07	1.00	1.04	1.38	1.79	1.05		
	Ag	g/t			6.22	5.50	5.56	6.15	6.61	7.25	8.33	9.52	10.13	8.79	10.14	9.16	9.17		
WASTE	tons (000s)			6,563	9,500	8,500	7,000	4,835	3,143	2,383	1,959	2,029	2,252	2,757	2,857	1,983	48		55,808
	TOTAL			6,563	13,000	12,000	10,500	8,335	6,643	5,883	5,459	5,529	5,752	6,257	6,357	5,483	889		98,648
	Stripping Ratio	1.30			2.71	2.43	2.00	1.38	0.90	0.68	0.56	0.58	0.64	0.79	0.82	0.57	0.06		
MILL PRODUCTION SCHEDULE																			
MILL FEED	tons (000s)				3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	840		42,840
	Au	g/t			0.83	0.90	0.95	1.02	1.14	1.22	1.17	1.07	1.00	1.04	1.38	1.79	1.05		
	Ag	g/t			6.22	5.50	5.56	6.15	6.61	7.25	8.33	9.52	10.13	8.79	10.14	9.16	9.17		
MILL RECOVERY	% of Au in feed	88%			88%	88%	88%	88%	88%	88%	88%	88%	86%	88%	88%	88%	88%		
	% of Ag in feed	60%			60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%		
Au PRODUCED		oz			82,578	88,825	94,212	101,214	112,474	120,714	115,073	106,087	99,134	103,155	136,411	177,372	25,035		1,382,864
Ag PRODUCED		oz			420,297	371,551	375,535	415,138	446,605	489,212	562,577	643,025	684,094	593,557	684,647	618,501	148,649		6,453,339
REVENUE ($000)																			
	Au Revenue	1.0			70,189	75,501	80,080	86,032	95,603	102,607	98,322	90,157	84,264	87,682	115,950	150,766	21,280		1,158,434
	Ag Revenue				6,069	5,295	5,351	5,916	6,364	6,971	8,017	9,163	9,748	8,458	9,756	8,814	2,118		91,961
	TOTAL	1.0			76,178	80,796	85,432	91,948	101,967	109,578	106,339	99,320	94,013	96,140	125,706	159,580	23,398		1,250,395
MINE OPERATING COSTS ($000s)																			
Mining Materials and Supply				7,427	11,846	13,132	12,532	11,246	10,131	9,102	8,416	7,885	7,885	7,885	7,714	7,457	2,490		125,157
Mining Labor				1,582	6,909	10,616	10,616	10,616	10,097	10,097	9,338	6,694	6,107	6,065	5,757	4,999	3,510		103,003
Total Mining		$2.31		9,009	18,755	23,748	23,148	21,862	20,228	19,199	17,754	14,579	13,992	13,950	13,471	12,456	6,000		228,160
Processing	$/ tonne processed	$7.45			26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	6,285		
G&A	$/ tonne processed	$1.33			4,655	4,655	4,655	4,655	4,655	4,656	4,655	4,655	4,655	4,655	4,655	4,655	1,118		
Environmental	$/ tonne processed	$0.25			875	875	875	875	875	873	875	875	875	875	875	875	210		
	TOTAL	1.0		9,009	50,466	55,418	54,856	53,673	51,938	50,909	49,464	46,289	45,702	45,660	45,161	44,166	13,622		
FREIGHT & MARKETING COST ($000s)																			
Refining, Freight and	$/oz Au recovered	$25			2,064	2,221	2,355	2,530	2,812	3,018	2,892	2,652	2,478	2,579	3,410	4,434	626		
Marketing	% of revenue	0.0%			0	0	0	0	0	0	0	0	0	0	0	0	0		
	TOTAL				2,064	2,221	2,355	2,530	2,812	3,018	2,892	2,652	2,478	2,579	3,410	4,434	626		
ROYALTY ($000s)		0%			0	0	0	0	0	0	0	0	0	0	0	0	0		
	TOTAL				0	0	0	0	0	0	0	0	0	0					
TOTAL COSTS ($000s)					52,529	57,679	57,213	56,102	54,750	53,927	52,356	48,941	48,180	48,239	48,591	48,600	14,246		641,356
TOTAL COSTS PER TON MILLED					15.01	16.48	16.35	16.03	15.64	15.41	14.96	13.98	13.77	13.78	13.88	13.89	16.96		
NET BEFORE TAX ($000s)				(9,000)	23,649	23,117	28,219	35,845	47,216	55,651	53,983	50,379	45,832	47,901	77,115	110,980	9,150		600,030
NET INCOME ($000)				[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	77,115	[illegible]	[illegible]		600,030
CAPITAL COST SUMMARY ($000s)																			
Road Access and Site Prep			5,000																5,000
Surface Buildings			1,600	2,000															3,600
Mining Equipment				15,346	13,921	120	120	1,060	120	4,060	1,020	1,060	720	720	120	120			38,527
Site Infrastructure, power, water			6,000	6,000															12,000
Process Plant			36,795	36,795															73,590
Office, warehouse, lab				5,000															5,000
Tailings			7,500	7,500				2,500					2,500						20,000
EPCM			10,559																10,559
Working Capital		3 Month Oper			12,616												(12,616)		0
Permitting, Reclamation, and Closure			5,000														5,000	10,000	20,000
TOTAL CAPITAL		1.0	72,454	72,641	26,537	120	120	3,560	120	4,060	1,020	1,060	3,220	720	120	120	-7,616	10,000	188,276
Contingency		20%	14,491	14,528	5,307	24	24	712	24	812	204	212	644	144	24	24	-1,523	2,000	
TOTAL CAPITAL WITH CONTINGENCY		1.0	86,946	87,169	31,846	144	144	4,272	144	4,872	1,224	1,272	3,864	864	144	144	-9,140	12,000	225,931
NET PRETAX CASH FLOW ($000s)			-86,946	-96,178	-8,196	22,973	28,076	31,573	47,074	50,784	52,759	49,107	41,968	47,037	76,971	110,836	18,290	-12,000	374,099

NPV (10%) $0

IRR 0%

TABLE 1-7: FLOTATION–CYANIDATION ALTERNATIVE SENSITIVITY ANALYSIS
PINNACLE MINES LTD. - SILVER COIN GOLD PROJECT
December 2009

Net Present Value Calculations ($000s) Capital Sensitivity			
Discount %	**Base**	**CAPEX-20%**	**CAPEX+20%**
0	374,099	419,286	328,913
5	170,130	210,485	129,776
8	95,428	133,442	57,415
10	58,349	94,970	21,728
12	28,861	64,201	-6,479
Net Present Value Calculations ($000s) Au Price Sensitivity, US$/oz			
Discount %	**850**	**900**	**800**
0	374,099	442,242	305,956
5	170,130	214,352	125,908
8	95,428	130,353	60,503
10	58,349	88.457	28,242
12	28,861	54,990	2,733
Net Present Value Calculations ($000s) Operating Cost Sensitivity			
Discount %	**Base**	**Op Cost-20%**	**Op Cost+20%**
0	374,099	497,358	280,841
5	170,130	254,687	85,574
8	95,428	164,340	36,516
10	58,349	118,962	-2,263
12	28,861	82,494	-24,771

TABLE 1-4: CASH FLOW FOR ALL - FLOTATION PROCESS ALTERNATIVE
PINNACLE MINES LTD - SILVER COIN PROJECT
December 2009

			PP 2	PP 1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Totals
PRICE SCHEDULE	$/oz Au				$ 850.00	$ 850.00	$ 800.00	$ 850.00	$ 853.55	$ 850.00	$ 800.00	$ 850.00	$ 850.00	$ 850.00	$ 800.00	$ 850.00	$ 850.00	$ 800.00	$ 850.00	$ 850.00	
	$/oz Ag				$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	
MINE PRODUCTION SCHEDULE																					
ORE	tons (000s)		0	0	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	1,673	54,173
	Au	g/t	0.00	0.00	0.74	0.72	0.78	0.83	0.85	0.87	1.00	1.27	1.03	0.98	0.83	0.90	0.86	1.38	1.73	1.24	
	Ag	g/t	0.00	0.00	6.91	5.79	5.53	5.36	5.43	6.30	6.48	6.83	7.64	8.50	8.25	8.65	7.74	8.08	8.05	8.22	
WASTE	tons (000s)		0	8,311	7,831	8,615	5,639	4,681	4,135	3,560	2,993	2,547	3,364	3,453	2,541	2,725	3,315	3,041	2,454	851	63,786
	TOTAL		0	8,311	11,331	12,115	[illegible]	[illegible]	7,635	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	119,959
	Stripping Ratio	1.21			2.23	2.46	1.67	1.34	1.18	1.02	0.86	0.73	0.96	0.70	0.73	0.78	0.95	0.87	0.71	0.50	
MILL PRODUCTION SCHEDULE																					
MILL FEED	tons (000s)				3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	1,673	54,173
	Au	g/t			0.74	0.72	0.78	0.83	0.85	0.87	1.00	1.27	1.03	0.98	0.83	0.90	0.86	1.38	1.73	1.24	
	Ag	g/t			6.91	5.79	5.53	5.36	5.43	6.30	6.48	6.83	7.64	8.50	8.25	8.65	7.74	8.08	8.05	8.22	
MILL RECOVERY	% of Au in feed	85%			85%	85%	85%	90%	85%	85%	85%	95%	85%	85%	85%	92%	85%	91%	95%	85%	
	% of Ag in feed	88%			88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	
Au PRODUCED		oz			78,255	78,542	83,253	86,286	80,737	101,354	104,475	114,603	110,441	104,443	88,980	95,242	102,444	131,117	151,720	63,436	1,635,547
Ag PRODUCED		oz			664,522	673,712	547,853	530,437	577,376	423,831	643,349	641,216	760,582	641,425	819,630	851,653	764,437	864,741	739,377	389,129	11,061,854
CONCENTRATE PRODUCED	tonnes	Wt% 2.50%			87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	41,820	
CONCENTRATE GRADE																					
	Au	g/tonne			0.81	0.85	0.95	1.01	1.04	1.18	1.22	1.31	1.26	1.18	1.17	1.10	1.17	1.54	2.06	1.53	
	Ag	g/tonne			7.83	8.54	8.26	6.04	6.63	7.13	7.34	7.64	8.69	8.83	13.41	9.73	8.78	10.27	8.11	8.30	
Au PAYABLE	Deduct .05 oz Au and pay for 95%				71,135	64,587	74,834	78,714	83,044	84,533	94,905	104,607	100,783	93,065	80,875	87,275	93,169	124,206	154,478	54,281	
Ag PAYABLE	Deduct 1 oz Ag and pay for 95%				587,171	481,801	437,440	420,873	485,342	508,832	529,813	564,760	629,407	718,300	781,974	725,847	644,855	773,444	674,003	329,643	
REVENUE ($000)																					
	Au Revenue	1.0			63,465	58,357	63,664	67,760	68,737	73,833	83,446	89,171	85,643	80,605	75,364	74,184	78,195	105,574	145,204	48,538	1,264,004
	Ag Revenue				8,363	4,543	4,334	5,084	6,613	7,363	7,162	8,105	8,113	10,236	11,143	10,345	8,185	10,947	8,023	4,757	131,444
	TOTAL	1.0			[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	1,307,743
MINE OPERATING COSTS ($000s)																					
Mining Materials and Supply				7,437	11,846	13,132	12,822	11,248	10,135	9,102	6,416	7,885	7,885	7,885	7,714	1,457	3,409				125,157
Mining Labor				1,563	6,909	10,614	10,614	10,618	10,007	10,007	8,338	6,684	6,137	4,065	5,757	4,995	3,519				133,003
Total Mining		63.31		9,000	18,755	23,744	23,146	21,862	20,339	19,109	17,754	14,576	13,893	13,900	13,471	12,454	9,000	15,111	13,214	3,572	251,637
Processing	$/tonne processed	$6.27			21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	10,486	
G&A	$/tonne processed	$1.33			4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	2,225	
Environmental	$/tonne processed	$0.25			875	875	875	875	875	875	875	875	875	875	875	875	875	875	875	418	
	TOTAL	1.0		9,000	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	
FREIGHT & MARKETING COST ($000s)																					
Trucking and Port Handling	$/tonne of Conc	$5.00			875	875	875	875	875	875	875	875	875	875	875	875	875	875	875	418	
Ocean Freight	$/tonne of Conc	$60			5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	2,509	
Smelter Treatment Charge	$/tonne of Conc	$200			17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	8,364	
Au Refining Charge	$/oz Au	$6.00			427	413	450	476	497	564	583	629	606	575	530	524	558	745	1,011	350	
Ag Refining Charge	$/oz Ag	$0.50			344	311	373	393	335	255	263	264	320	359	391	365	372	385	337	165	
	TOTAL				[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	
ROYALTY ($000s)		0%			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
	TOTAL				0	0	0	0	0	0	0	0	0	0							
TOTAL COSTS ($000s)				9,004	70,545	75,431	74,916	73,651	72,053	77,318	83,680	69,533	64,016	65,879	65,501	64,443	67,635	67,341	65,483	30,308	1,056,137
TOTAL COSTS PER TON MILLED					20.16	21.57	21.40	21.04	20.59	20.32	19.81	11.83	18.29	18.85	18.71	18.41	18.57	19.24	18.73	18.12	
NET BEFORE TAX ($000s)				(9,004)	(2,016)	(10,601)	(4,642)	752	4,316	16,077	20,253	30,683	28,744	35,033	22,036	25,094	30,381	40,215	67,342	33,831	331,606
NET INCOME ($000s)				[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	331,606
CAPITAL COST SUMMARY ($000s)																					
Road Access and Site Prep			5,000																		5,000
Surface Buildings			1,600	2,000																	3,600
Mining Equipment				11,344	13,931	120	120	1,066	120	4,080	1,030	1,066	720	720	120	120					38,537
Site Infrastructure, power, water			5,000	8,000																	13,000
Process Plant			28,775	28,775																	58,540
Office, warehouse, lab				3,000																	3,000
Tailings			7,500	7,500				2,500					2,500								20,000
EPCM			8,154																		8,154
Working Capital		3 Month Opex			11,554															(11,554)	0
Permitting, Reclamation, and Closure			3,000															5,000	5,000	5,000	18,000
TOTAL CAPITAL		1.0	62,034	61,619	25,475	120	120	3,566	120	4,080	1,030	1,066	3,220	720	120	120	0			-4,554	153,831
Contingency		20%	12,400	13,123	5,099	24	24	712	24	816	204	213	644	144	24	24	0				
TOTAL CAPITAL WITH CONTINGENCY		1.0	74,435	74,739	30,574	144	144	4,272	144	4,896	1,234	1,279	3,864	865	144	144	0	0	0	-4,554	184,297
NET PRETAX CASH FLOW ($000s)			[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

NPV (10%) $0
IRR 0%

TABLE 1-9: ALL-FLOTATION ALTERNATIVE SENSITIVITY ANALYSIS PINNACLE MINES LTD. - SILVER COIN GOLD PROJECT December 2009			
Net Present Value Calculations ($000s) Capital Sensitivity			
Discount %	Base	CAPEX-20%	CAPEX+20%
0	137,310	178,169	98,450
5	-16,112	19,345	-51,569
10	-82,013	-49,572	-114,454
12	-96,000	-64,648	-127,352
Net Present Value Calculations ($000s) Au Price Sensitivity, US$/oz			
Discount %	850	900	800
0	137,310	211,783	62,836
5	-16,112	28,288	-60,512
10	-82,013	-53,666	-110,371
12	-96,000	-71,890	-120,110
Net Present Value Calculations ($000s) Operating Cost Sensitivity			
Discount %	Base	Op Cost-20%	Op Cost+20%
0	137,310	275,154	-535
5	-16,112	72,795	-105,019
10	-82,013	-20,980	-143,047
12	-96,000	-42,703	-149,297

1.8 Exploration Potential

There is excellent potential to grow the Silver Coin resource by 50 to 100%. The resource remains substantially open to the north and northwest; and many of the best intercepts in recent drilling come from the north end of the deposit. While the topography and rock conditions suggest that drilling costs will be higher in some areas of the north, drilling on the northern third of the deposit has been extremely productive to date, yielding approximately 400,000 oz of gold per 100 meters of strike. Pinnacle expects the next step-out drill fences at 50m intervals to be very productive. Discovery costs on the next increments of the resource could easily be $2.50 per oz or less.

1.9 Potential Limitations

Tt is not aware of any potential limitations to the project that would materially change any of the data, resource estimates, environmental considerations, socio-economic factors, or conclusions presented within this report that are outside of the normal factors that may impact mining projects, such as price variability, exchange rates, permitting time, etc. With respect to the Silver Coin Gold Project, the land tenure is secured by patented and unpatented claims, the existing environmental liabilities are well documented and have been adequately addressed, potential new environmental issues are part of this and future studies and are not anticipated to materially impact the path forward. The region has good existing infrastructure, power and water and excellent road access is provided by the Granduc Road. Exploration and development drilling, as well as metallurgical testing and analyses are expected to continue in 2010.

2.0 INTRODUCTION

Pinnacle commissioned Mr. Robert Perry and Tetra Tech (Tt) to prepare a Technical Report for the Silver Coin Gold Project near Stewart, British Columbia, Canada that meets the requirements of Canadian National Instrument 43-101 ("NI 43-101"). This report has been prepared in accordance with the guidelines provided in NI 43-101, Standards of Disclosure for Mineral Projects, dated December 23, 2005. The Qualified Persons responsible for this report are Mr. John W. Rozelle, P.G., Principal Geologist of Tt and Mr. Robert Perry. Mr. Perry is a Qualified Person and consultant to Pinnacle who visited the Pinnacle core shed in Stewart to look at core on April 20-22, 2009. On a subsequent visit August 18, 2009, Mr. Perry toured the property. On both visits, Mr. Alex Walus, an employee of Pinnacle who had been the project geologist for Silver Coin for at least four years, accompanied Mr. Perry.

2.1 Terms of Reference

The purpose of this report is to analyze and interpret all available data in order to produce a NI 43-101 compliant mineral resource estimate assuming the data adequately support such an estimate. Pinnacle currently has 70% ownership of all claims within the currently defined resource area. It is the intent of Pinnacle to continue to drill on the site in order to better define and expand the mineralization and its boundaries.

2.2 Scope of Work

The scope of work undertaken by Tt involved compiling or creating the three-dimensional computerized geologic model and updated resource estimate, metallurgical review, and mine planning, scheduling, and capital and operating cost estimation studies on the Silver Coin Gold Project as contracted by Pinnacle. Based on this information Tt, in conjunction with Mr. Robert Perry, has developed this Preliminary Economic Assessment (PEA) and prepared recommendations on further work needed to advance the project to pre- and/or full-feasibility stage.

2.3 Effective Date

The effective date of the mineral resource statements in this report is December 30, 2009.

2.4 Qualifications of Consultant

This report has been prepared based on a technical review by consultants sourced from Tt's Golden, Colorado office and Pinnacle professionals (TABLE 2-1). These professionals are specialists in the fields of geology, geostatistics, mineral resource estimation, mineral reserve estimation and classification.

TABLE 2-1: KEY PROJECT PERSONNEL PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009		
Company	**Name**	**Title**
Robert Perry Consulting LLC	Robert Perry	Geological Consultant
Tetra Tech, Inc.	John Rozelle	Principal Geologist
	Rex Bryan	Sr. Geostatistician
	Steve Krajewski	Sr. Geologic Modeller
	Lee Aga	Sr. Mine Planner
	Eric Olin	Principal Metallurgical Engineer
FGM Mining Consultants	Landy Stinnett, P.E.	Mining Engineer

2.5 Basis of Report

This report draws heavily on information contained in a prior Technical Report on Silver Coin prepared by MineFill Services Inc. written in April 2007, a partially completed draft Technical Report prepared by Snowden Resources in latter 2008, a partially completed draft Technical Report prepared by Mr. Alex Walus, a Pinnacle staff geologist, in early 2009 and an internal report prepared for Pinnacle by Bitterroot Group LLC in May 2009 for which Robert Perry was the principal author. Where information from the MineFill or Snowden reports is included in this report it is specifically credited. Information from the Walus and Bitterroot internal Pinnacle reports is generally credited by this reference and may or may not be specifically referenced.

Information provided by Pinnacle includes:

- Assumptions, conditions, and qualifications as set forth in the report;
- Land status (Bitterroot Group analysis by staff attorney Jose Pinedo);
- Drill hole records;
- Property history details;
- Sampling protocol details;
- Geological and mineralization setting;
- Data, reports, and opinions from prior owners and third-party entities; and
- Gold and other assays from original assay records and reports.

2.6 Units

Unless explicitly stated, all units presented in this report are in the Metric System (i.e. tonnes, kilometers, centimeters, and troy ounces). All monetary values are in United States (US) dollars unless otherwise stated.

Common units of measure and conversion factors used in this report include:

Linear Measure:

1 inch = 2.54 centimeters

1 foot = 0.3048 meter

1 yard = 0.9144 meter

1 mile = 1.6 kilometers

Area Measure:

1 acre = 0.4047 hectare

1 square mile = 640 acres = 259 hectares

Capacity Measure (liquid):

1 US gallon = 4 quarts = 3.785 liter

1 cubic meter per hour = 4.403 US gpm

Weight:

1 short ton	= 2000 pounds	= 0.907 tonne
1 pound	= 16 oz	= 0.454 kg
1 oz (troy)	= 31.103486 g	

Analytical Values:

	percent	**grams per metric tonne**	**troy ounces per short ton**
1%	1%	10,000	291.667
1 gm/tonne	0.0001%	1.0	0.0291667
1 oz troy/short ton	0.003429%	34.2857	1
10 ppb			0.00029
100 ppm			2.917

Frequently used acronyms and abbreviations:

AA	=	atomic absorption spectrometry
Ag	=	silver
As	=	acid soluble
AsCu	=	acid soluble copper
Au	=	gold
°C	=	degrees Centigrade
CIC	=	Carbon-in-column
CIMM	=	Canadian Institute of Mining, Metallurgical, and Petroleum
CIP	=	Carbon-in-pulp

CN	=	cyanide
CNCu	=	cyanide soluble copper
°F	=	degrees Fahrenheit
FA	=	Fire Assay
ft	=	foot or feet
g	=	gram(s)
g/kWh	=	grams per kilowatt hour
g/t	=	grams per tonne
h	=	hour
ICP	=	Inductively Coupled Plasma Atomic Emission Spectroscopy
km	=	kilometer
kV	=	kilovolts
kWh	=	Kilowatt hour
kWh/t	=	Kilowatt hours per tonne
l	=	liter
m	=	meter(s)
ml	=	milliliter
m2	=	square meter(s)
m2/t/d	=	square meters per tonne per day
m3	=	cubic meter(s)
m3/h	=	cubic meter(s) per hour
mm	=	millimeter
MW	=	megawatts
NSR	=	net smelter return
Ag oz/t	=	troy ounces silver per short ton (oz/ton)
Au oz/t	=	troy ounces gold per short ton (oz/ton)
ppm	=	parts per million
ppb	=	parts per billion
RC	=	reverse circulation drilling method
T	=	total
TCu	=	total copper
ton	=	short ton(s)
tonne	=	metric tonne
t/m^3	=	tonne per cubic meter
tpd	=	tonnes per day
tph	=	tonnes per hour
µm	=	micron(s)
%	=	percent
tpy	=	tons (or tonnes) per year

tpm = tons (or tonnes) per month
tpd = tons (or tonnes) per day

actinium = Ac	aluminum = Al	americium = Am	antimony = Sb	argon = Ar
arsenic = As	astatine = At	barium = Ba	berkelium = Bk	beryllium = Be
bismuth = Bi	bohrium = Bh	boron = B	bromine = Br	cadmium = Cd
calcium = Ca	californium = Cf	carbon = C	cerium = Ce	cesium = Cs
chlorine = Cl	chromium = Cr	cobalt = Co	copper = Cu	curium = Cm
dubnium = Db	dysprosium = Dy	einsteinum = Es	erbium = Er	europium = Eu
fermium = Fm	fluorine = F	francium = Fr	gadolinium = Gd	gallium = Ga
germanium = Ge	gold = Au	hafnium = Hf	hahnium = Hn	helium = He
holmium = Ho	hydrogen = H	indium = In	iodine = I	iridium = Ir
iron = Fe	juliotium = Jl	krypton = Kr	lanthanum = La	lawrencium = Lr
lead = Pb	lithium = Li	lutetium = Lu	magnesium = Mg	manganese = Mn
meltnerium = Mt	mendelevium = Md	mercury = Hg	molybdenum = Mo	neodymium = Nd

3.0 RELIANCE ON OTHER EXPERTS

The Silver Coin Gold Project, having previously been an operating mine for several years, has been the subject of numerous written reports. Many of these reports and other documents were prepared by mining consulting firms on behalf of the operators of the mine/property at the time. Tt has used a number of the references in the preparation of the mineral resource estimate detailed herein. The reports referenced have each been reviewed for materiality and accuracy, as they pertain to Pinnacle's plan for development of the property. Specific experts, both internal to Tt and external, that had an important role in the preparation of this report include:

Dr. Stephen A. Krajewski

Dr. Krajewski graduated with Geography (B.S., 1964), Geology (M.S., 1971) and Earth Science (Ed.D., 1977) degrees from The Pennsylvania State University. He is a member of the American Institute of Professional Geologists (Member Number 4739), a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME), member of the American Association of Petroleum Geologists, and a member of the Rocky Mountain Association of Geologists.

Dr. Krajewski has utilized computers to map and model mineral deposits since 1983. His geologic career has included 42 years of domestic and international experience in the employ of major and junior mining industry companies, major and minor oil and gas companies, environmental consulting companies, a state geological survey, and universities.

Dr. Rex C. Bryan

Dr. Bryan graduated with a Mineral Economics Ph.D. from the Colorado School of Mines, Golden, Colorado, in 1980. He graduated in 1976 from Brown University, in Providence, Rhode Island, with a MSc. Geology. He also graduated from Michigan State University with a MBA (1973) and a BS in Engineering (1971). Dr. Bryan is a member of SME.

Dr. Bryan has worked as a geostatistical reserve analyst and mineral industry consultant for a total of 26 years since graduating from the Colorado School of Mines. He is an expert witness to industry and for the U.S. Department of Justice on ore-grade control, reserves, and mine contamination issues. He is currently a consultant to the industry in mine valuation, ore reserve estimation, and environmental compliance.

Mr. Robert Perry

Mr. Perry graduated from the University of Colorado with a B.A. in Geology in 1973 and with a M.S. in Geology in 1976, also from the University of Colorado. Mr. Perry is a Certified Professional Geologist (CPG) with the American Institute of Professional Geologists (CPG #11074)

Mr. Perry has worked as an economic geologist for more than 30 years in the U.S., Canada, South America, Central Asia and Mexico. He discovered the Beartrack Mine in Idaho and uranium mines in western Colorado. He has held senior management positions with both public and private companies and is currently working as a consultant to several companies.

Mr. Robert Perry is the Qualified Person responsible for most of the sections of this report to ensure that they meet all of the necessary reporting criteria as set out in Canadian Instrument

NI43-101 guidelines. Mr. John W. Rozelle, P.G. is the Qualified Person responsible for a portion of SECTION 1.0 and all of SECTIONS 16.0, 17.0, 18.0 and 24.0 of this report.

4.0 PROPERTY LOCATION AND DESCRIPTION

4.1 Location

The Silver Coin property is located about 24 kilometers north of Stewart, British Columbia centered on UTM coordinates 436,000mE, 6,218,000mN (Zone 9, NAD83) or about 130 degrees 02 minutes longitude west and 56 degrees 06 minutes latitude north on NTS map sheets104B010 and 104B020. Of the 26 claims that make up the property, 22 claims are jointly owned by Pinnacle (70%) and Mountain Boy Minerals (30%) as detailed in the TABLE 4-1. The remaining 4 lesser explored INDI claims are jointly owned by Nanika Resources Inc. (45%), Pinnacle (28.05%) and Mountain Boy (26.95%).

4.2 Area of the Property, Mineral Tenure, Title

The property consists of 26 contiguous claims, including one Crown Grant Claim (FIGURE 4-2) which totals 2,244.5 Ha. However, because of overlapping boundaries, these claims cover a net area of 1,255 Ha.

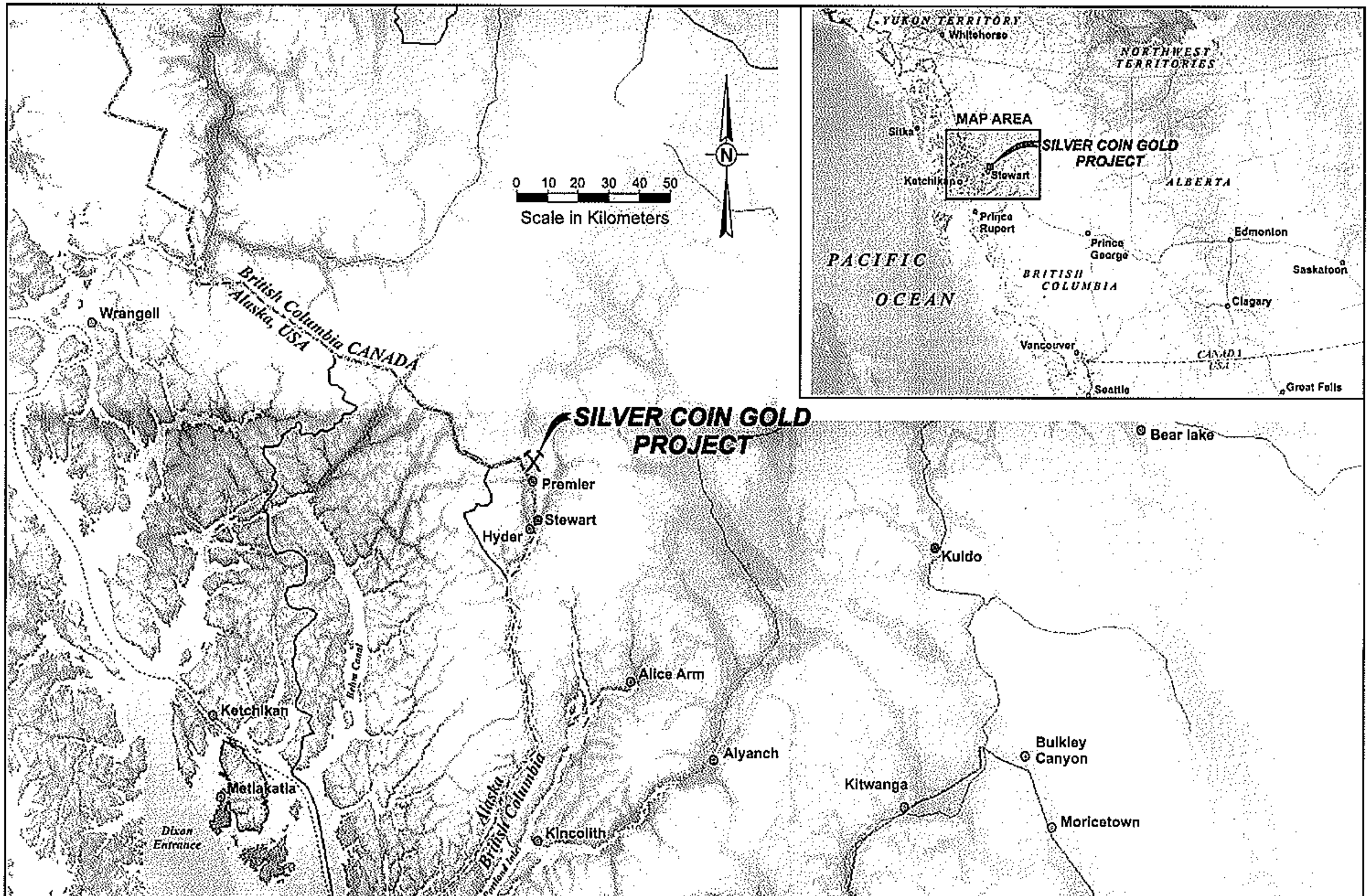
The principal resource at Silver Coin is called the Main Breccia Zone and almost the entire resource lies on two claims; the Kansas claim and the Big Missouri claim. Based on the various agreements governing the property, Pinnacle owns 70% of Kansas and Big Missouri claims, with Mountain Boy Minerals owning the remaining 30%. Pinnacle has an option which expires in 2014 to acquire an additional 10% of the claims.

Tt has accepted the work completed by the Bitterroot Group and their staff attorney, Jose Pinedo, to be accurate regarding the status of the current land holdings.

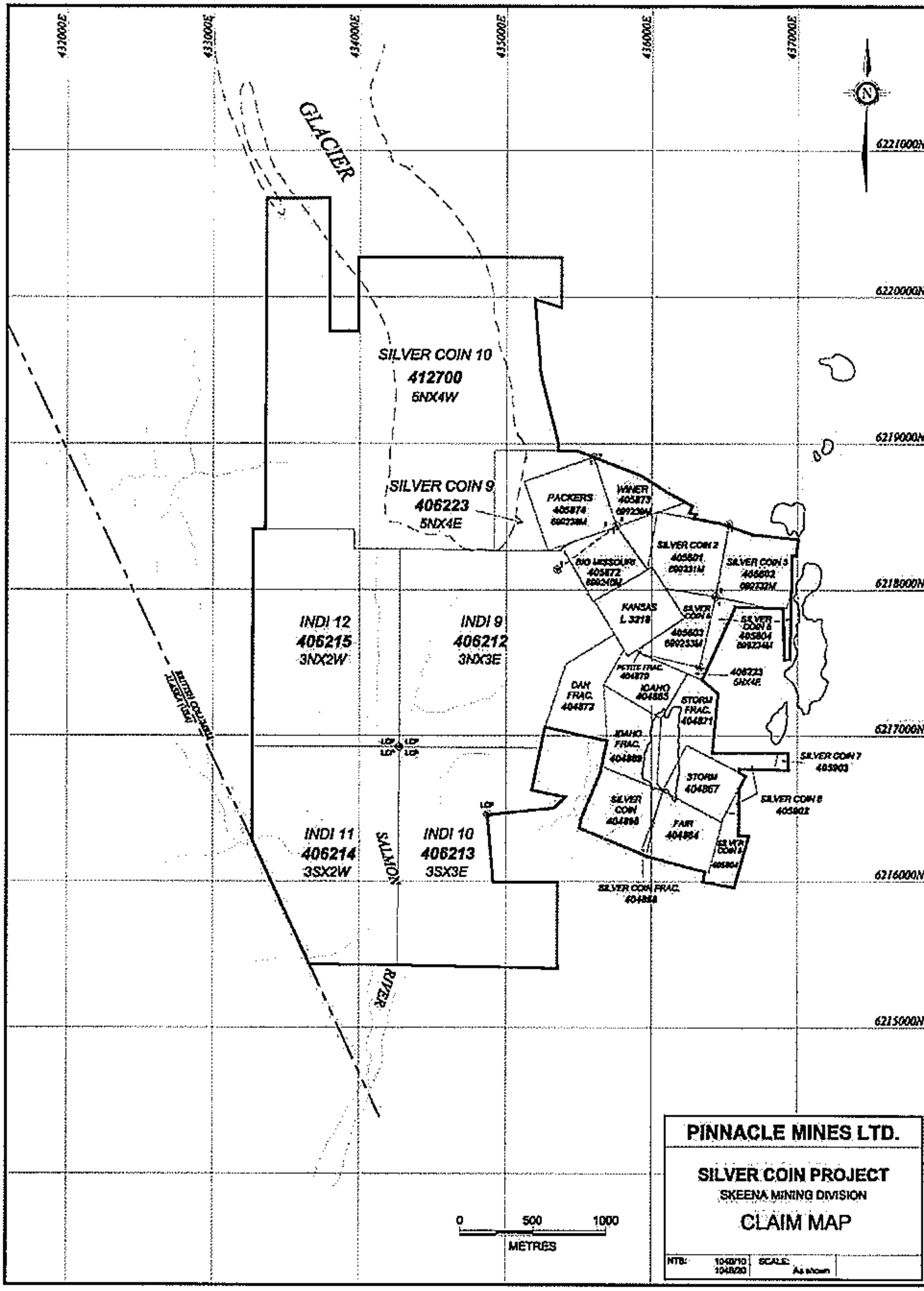
Review of Ownership Documents

Tt has had access to the following information and agreements which support Pinnacle's ownership of the Silver Coin Gold Project. To the best of our knowledge, the applicable agreements are in good standing, and the representations and warranties given by the parties in each of them are still in effect and remain valid. Pinnacle has represented that the Silver Coin Project is not subject to any other royalties, back-in rights, payments, agreements, or encumbrances, aside from those described herein:

- Joint Venture Agreement between Mountain Boy Minerals Ltd ("MBM") and Pinnacle Mines Ltd ("Pinnacle") dated December 31, 2005, effective as of June 1st 2006. ("MBM-Pinnacle JV").
- Joint Venture Agreement between Mountain Boy Minerals Ltd ("MBM") and New Cantech Ventures, Inc. ("Cantech") dated January 1, 2005. ("MBM-Cantech JV"). New Cantech Ventures, Inc is now Nanika Resources Inc. (NR)
- Option and Joint Venture Agreement between Tenajon Resources Corporation ("Tenajon") and Pinnacle Mines Ltd ("Pinnacle") dated May 12, 2005. ("Tenajon-Pinnacle JV").
- Letter Agreement between Tenajon Resources Corporation ("Tenajon") and Pinnacle Mines Ltd ("Pinnacle") dated April 15, 2008 ("Tenajon-Pinnacle LA"); and, the Share Purchase Agreement between Tenajon Resources Corporation ("Tenajon") and Pinnacle Mines Ltd ("Pinnacle") dated April 15, 2008 ("Tenajon-Pinnacle SPA").



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig4-1.cdr	**Figure 4-1** **General Location Map of the Silver Coin Gold Project**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/30/2009	



Issued by:	Drawing Provided by/Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig4-2.cdr	**Figure 4-2** **Silver Coin Gold Project** **Claim Boundary Map**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

TABLE 4-1: CLAIMS COMPRISING THE SILVER COIN PROPERTY PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009						
Claim Name	**Crown Granted**	**Tenure Number**	**Units**	**Area (ha)**	**Owner**	**Expiry Data**
Kansas	Crown granted	3218 C.G.	1	19.55	Pinnacle 70%, MBM30%	01/07/2010
Storm Fraction	Reverted Crown granted	404871	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Dan Fraction	Reverted Crown granted	404872	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Storm	Reverted Crown granted	404867	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Silver Coin	Reverted Crown granted	404866	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Idaho	Reverted Crown granted	404865	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Fair	Reverted Crown granted	404864	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Silver Coin Fraction	Reverted Crown granted	404868	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Idaho Fraction	Reverted Crown granted	404869	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Petite Fraction	Reverted Crown granted	404870	1	25	Pinnacle 70%, MBM 30%	21/07/2017
Silver Coin 2	2-post units	405601	1	25	Pinnacle 70%, MBM 30%	3/10/2017
Silver Coin 3	2-post units	405602	1	25	Pinnacle 70%, MBM 30%	3/10/2017
Silver Coin 4	2-post units	405603	1	25	Pinnacle 70%, MBM 30%	4/10/2017
Silver Coin 5	2-post units	405604	1	25	Pinnacle 70%, MBM 30%	4/10/2017
Silver Coin 6	2-post units	405902	1	25	Pinnacle 70%, MBM 30%	8/10/2017
Silver Coin 7	2-post units	405903	1	25	Pinnacle 70%, MBM 30%	8/10/2017
Silver Coin 8	2-post units	405904	1	25	Pinnacle 70%, MBM 30%	9/10/2017
Big Missouri	2-post units	405872	1	25	Pinnacle 70%, MBM 30%	11/10/2017
Winer	2-post units	405873	1	25	Pinnacle 70%, MBM 30%	10/10/2017
Packers	2-post units	405874	1	25	Pinnacle 70%, MBM 30%	10/10/2017
Silver Coin 9	Modified grid	406223	20	500	Pinnacle 70%, MBM 30%	28/10/2017
Silver Coin 10	Modified grid	412700	20	500	Pinnacle 70%, MBM 30%	29/07/2017
INDI 9	4 post claim	406212	9	225	Pinnacle 28.05%, MBM 26.95%, Nanika Resources Inc. 45%	27/10/2017
INDI 10	4 post claim	406215	9	225	Pinnacle 28.05%, MBM 26.95%, Nanika Resources Inc. 45%	27/10/2017
INDI 11	4 post claim	406214	9	150	Pinnacle 28.05%, MBM 26.95%, Nanika Resources Inc. 45%	27/10/2017
INDI 12	4 post claim	406212	9	150	Pinnacle 28.05%, MBM 26.95%, Nanika Resources Inc. 45%	27/10/2017

- Arrangement Agreement between Tenajon Resources Corporation ("Tenajon") and Pinnacle Mines Ltd ("Pinnacle") dated September 4, 2008. ("Tenajon-Pinnacle AA").
- Purchase Agreement between Pinnacle Mines Ltd. ("Pinnacle") and Mountain Boy Minerals Ltd. ("MBM") dated July 6, 2009 ("Pinnacle-MBM PA").
- The official claim map from the BC government website for information on the Silver Coin Claims.

Summary of Relevant Claim Transactions

- Pinnacle owns 70% of the Silver Coin Claims, all nine of the Reverted Crown granted claims, all ten of the 2-posted units, and 40 of the units in the Silver Coin 9 and Silver Coin 10 modified grid claims. The other 30% is owned by MBM, all in accordance with the MBM-Pinnacle JV.
- Pinnacle owns 28.05% of the INDI 9 to INDI 12 claims also known as the "Dauntless project". MBM owns 26.95% and Nanika Resources (formerly New Cantech Ventures) owns 45%, all in accordance with the MBM-Cantech JV and the MBM-Pinnacle JV. Based on the terms of these two agreements, MBM earned 55% of these claims from Cantech and Cantech kept 45%. Pinnacle now owns 51% of the 55% that MBM owns, or 28.05%.
- Pinnacle and Tenajon, signed the Tenajon-Pinnacle JV by which Pinnacle could earn up to 60% of the Kansas Claim. Pinnacle fulfilled those conditions and earned such percentage. In June 2006, this claim became part of the MBM-Pinnacle JV, so MBM earned 49% of the 60% owned by Pinnacle, or 29.4%. Later, in 2008, Pinnacle bought out Tenajon's interest in the Tenajon-Pinnacle JV purchasing the balance of 40% with Pinnacle shares (the Tenajon-Pinnacle LA; the Tenajon-Pinnacle SPA and the Tenajon-Pinnacle AA). The result of these transactions was that Pinnacle owned 70.6% of the Kansas claim and MBM owned 29.4%.
- Pinnacle and MBM, signed the Pinnacle-MBM PA by which Pinnacle paid cash for an additional 19% of all the claims (except the INDI claims) and transferred 0.6% of the Kansas claim to MBM which resulted in Pinnacle owning 70% and MBM owning 30% of all the Silver Coin Claims (except the INDI claims, which still remains 28.05%; 26.95% and 45% Pinnacle-MBM-Nanika respectively).

4.3 Environmental Liability and Permitting

The Silver Coin Gold Project is an advanced stage exploration project. It occurs in a moderately active mining district dating back approximately 100 years. Prospecting and small scale exploration dates back to the early 1900's. Since the early 1980's, (prior to MBM and Pinnacle's involvement) several companies drilled approximately 714 exploration holes on the Silver Coin property. Westmin mined approximately 100,000 tonnes of ore from an underground operation in 1991 and there are small mine waste dumps remaining as a result of this underground mining. There was larger scale historic mining to the north at Big Missouri and several kilometers south at Silbak Premier. However, there are no apparent large-scale environmental issues on the property held by the Pinnacle Mines-MBM JV ("Pinnacle").

The project is located in a scenic area near mountain streams, lakes, and the headwaters of the Salmon River and the Salmon Glacier. It is also located near the international border with the United States. There is precedent for successfully operated modern mines in the area, as Silbak Premier operated an open pit gold mine and mill complex with few significant environmental issues. Westmin was actually one of the first mine operators to develop an

Environmental Management Plan (EMP) for the mine and to conduct proper environmental audits. A water treatment plant was constructed to manage runoff, and Boliden maintains year-round environmental monitoring at the site associated with the Silbak mine site.

The most significant disturbance on the site is the portal area located above the Granduc Road where the steep hillside has been notched to provide a level area. The portal itself has been closed but the pad area and nearby drill roads remain. Due to the extensive historical drilling at the site there are numerous drill access roads that may require regrading and closure if the project does not proceed to development. The environmental firm Cambria Gordon Ltd. visited the site in 2007 and again in the fall of 2009. Its comments are provided below.

In the fall of 2009, Pinnacle retained Cambria Gordon Ltd. to conduct preliminary environmental baseline studies for the proposed Silver Coin Project. The environmental baseline study program included the following components:

- Fisheries assessment (presence/absence of fish) and preliminary limnology of No-Name Lake.
- Determine potential sampling locations for a preliminary baseline water quality monitoring program and conduct measures of physical water quality parameters.
- Stream-flow measurements and preliminary hydrological information of year-round surface water flows on the property.
- Overview assessment of rare and/or endangered wildlife and vegetation species/ecological communities, whose distribution overlaps with the property footprint.

The following information is taken directly from Cambria Gordon's "Executive Summary":

"The key objective of this document is to report the preliminary environmental baseline information gathered in desktop scoping exercises and in the field program, in order to support future baseline studies and Project design considerations.

To determine the presence/absence of fish in No-Name Lake, two gillnets (floating and sinking) were set overnight for 24 hours and 15 minnow traps were set for a total effort of 340 hours. No fish were caught using both sampling methods. A bathymetric survey was completed (along an E-line transect across the length of No-Name Lake) to determine water depths, which ranged from 12.1 m to 31.1 m. Limnological data collected from No-Name Lake provided dissolved oxygen levels that were on the lower end of the threshold in terms of supporting fish in the water column. The lake appears to be of low productivity, as aquatic invertebrates were not observed along the shorelines or captured in traps and water samples were colourless (an indicator of low productivity).

No-Name Lake is a candidate for Non-Fish Bearing Status (NFBS) classification (granted by the BC Ministry of Environment) based on: 1) the sampling effort conducted with no fish captured, 2) barriers present (between No-Name Lake and known fish habitat >5 km downstream) which prevent the upstream migration of fish, 3) the assessed low productivity of the lake and 4) the biophysical setting of the lake - high elevation (820 m) and downstream waters that are steep with numerous cascades and falls (Cascade River).

A total of 12 sampling locations were identified as part of the preliminary baseline water quality monitoring program. All 12 sites were located east of the Granduc Road. At each station, water temperature, dissolved oxygen, pH, and conductivity were recorded. Water temperatures ranged from 4.3 to 10.7 0C, dissolved oxygen ranged from 7.1 to 10.0 mg/L, pH ranged from 7.7 to 9.2 and conductivity ranged from 12.0 to 298.3 µs/cm.

At the time of the field survey, two surface watercourse locations (Site A, Site C) contained adequate depth/flow to conduct water velocity measurements such as depth, width, velocity and total flow (m^3/s). Site A was located just downstream from No-Name Lake and had a total flow of 0.08 m^3/s. Site C was located further downstream and had a total flow of 0.18 m^3/s. Increased flows at Site C can be explained by additional inflow from a few small tributaries.

The desktop review of rare and/or endangered species and key habitats was performed using a list of protected rare and/or endangered species and ecological communities that are potentially present in the Project area. Four mammal species, 2 bird species, 11 plant species, and 4 ecological communities were identified as having distributions that overlap with the Project area. A field program was carried out on September 23rd – 25th, 2009 to collect preliminary baseline information in relation to vegetation, wildlife, and ecological communities. The Project area was broken down into three study areas and aerial photo interpretation was used to identify distinct vegetation and wildlife habitat types. Radius plot and strip transect surveys were utilized to collect information on vegetation and wildlife habitat types and to assess the occurrence of vegetation, wildlife and/or wildlife habitat features. None of the listed plant species and ecological communities were observed in the representative plots sampled. The study identified potential habitat for mammals (carnivores, rodents, ungulates) and birds (passerine, raptors, waterfowl). Mapped mountain goat wintering range habitat is present within the project property boundaries on the west side of Granduc Road. No unique and/or critical habitats associated with rare and/or endangered wildlife species were identified in the representative plots sampled."

4.3.1 Consideration of the use of Cyanide for Mineral Processing

Cambria Gordon also reviewed the history of Cyanide use in the region and the regulatory issues that Pinnacle may encounter in proposing a process involving cyanide recovery of gold at Silver Coin. The following information is taken directly from their conclusions.

"Considerations in the use of cyanide for gold extraction (including transportation, storage and handling, permitting and waste discharges) were obtained from a literature review of current permitting and management practices of cyanide extraction processes for mines in BC and Alaska.

The use of cyanide for the extraction of gold and/or silver is one of the most commonly used extraction processes; however, due to the potentially toxic nature, cyanide use can also be controversial. Currently, neither B.C. nor Alaska prohibits the use of cyanide for gold extraction.

Legislation exists, provincially in BC, in the State of Alaska, and federally in Canada and the US, that could limit the preferred operational use of cyanide. Limiting factors would need to be considered as part of the design, and would include, but not be limited to transportation providers, storage facilities, and management programs, and effluent discharge quality.

The project would be required to demonstrate a need for the preferred cyanide extraction process, in comparison to alternative processes that provide a reduced risk to environmental and social components. Economic arguments, on their own, are typically not acceptable if the project remains economically viable using alternate processing techniques.

Based on our literature review, cyanide remains a regulated gold extraction process in BC and Alaska. As part of the regulatory process of a new project, the project would have to demonstrate that alternative techniques for extraction, as compared to cyanide use, do not provide a reduced risk to the environmental and social components, are not feasible, or are not economically viable.

As a result of the above, cost considerations during the pre-feasibility design stage should consider the economic differences between design, permitting (effects assessment), operations, and decommissioning of a cyanide extraction process and a non-cyanide extraction process."

4.3.2 Existing Environmental Liabilities

Cambria Gordon had visited the Silver Coin site in 2007 in the company of J. Pardoe, the then and current Provincial officer in charge of permitting for the Silver Coin project. This earlier visit combined with itsr recent site activity is the basis for its assessment of the existing environmental and reclamation liability at the site. As above, the following text is taken directly from its report.

"During the 2007 reclamation and 2009 preliminary environmental baseline site visits, no facilities, machinery, or non-organic debris were observed on site.

The Silver Coin property is traversed by established roads. The Granduc Road, traversing the west facing slope of the property is a public access road maintained by the District of Stewart, typically in summer only, for mining industry activities and tourism.

The west face of the property and the landscape surrounding the property is very steep terrain, with observable downslope movement of material in the form of individual boulders and small landslides.

Select slope areas within the west face of the property disturbed by exploration activities have been loosened or oversteepend when compared to the natural landscape.

Due to the known mineralization of the general area, a possibility exists that rock on the property or drainage from or on the property has potential to leach metals or cause acid rock drainage (ML/ARD). This potential also exists for the naturally exposed, undisturbed rock in the surrounding areas.

Due to the exploration techniques used, appearing to be primarily drilling from pads with minimal trenching, the broken rock on site is limited and typically not more than a single layer thick on the natural underlying land or supporting benches.

Therefore, as a result of the limited quantity of rock and the naturally surrounding exposed rock, it is reasonable to assume the MEMPR would not require rock testing for ML/ARD. MEMPR may, however, require confirmation that water quality emitting from the buried adits on the property are neutral and do not contain levels of elevated metals. We have assumed the water quality would be acceptable for discharge."

The Cambria Gordon summary of cyanide issues and reclamation liabilities is included in its report. Cambria Gordon estimates that the current cost to reclaim the site is approximately US$66,000. In May 2009, Janice Girling of the Ministry's Smithers office confirmed that the annual reporting on the property is current and the US$35,000 reclamation security bond is intact. However, Jill Pardoe, Chief Inspector of Mines out of Smithers, notes that the property is due for an assessment of securities since she was unable to inspect the site in 2008. Thus, Pinnacle may be required to increase the reclamation security bond.

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

5.1 Accessibility and Infrastructure

Excellent paved roads connect Stewart with Smithers and Terrace, which are major supply centers in this part of British Columbia. A 25 km stretch of good gravel road (Granduc Road) links Stewart with the property. A section of this road from Stewart to Premier Mine (11 km) is maintained year-round. Heavy snowfalls limit road access beyond the Premier Mine November to May unless snow clearing is done on the non-maintained portion of this road. A short spur road off the Granduc Road, which crosses the property, provides access to the claims. An alternative access to the property is via a 4x4 road from the Granduc Road near the Premier Mine. This road continues along Silver Lake eventually connecting with the access road which joins the Granduc Road on the top of the Big Missouri Ridge.

Stewart features a year-round seaport with full loading facilities. For many years this port has been used to ship ore and ore concentrates from Red Cliff, Granduc, Snip, Eskay Creek and other mines. Currently, ore from the Huckleberry Mine is shipped through this port.

5.2 Climate and Physiography

Climate in the area can be severe. Heavy snowfalls in the winter and rain and fog in the summer are typical of the Stewart area. Snowfall up to 30m has been experienced at the higher elevations, which can remain in the gullies until July. Because of the mountainous terrain and weather conditions, field work is generally restricted to between May and November. However, once development starts, year-round core drilling and development work can proceed as has been done on many properties in the general area.

5.3 Physiography and Topography

The area of the Silver Coin property encompasses steep mountain slopes typical of the Coast Range region of British Columbia. Thick glacial moraine material is restricted mostly to lower elevations and valley floors with good rock exposure along ridge tops and creek beds.

The western part of the property; namely the INDI claims and Silver Coin 10 claim cover a section of the main Salmon River Valley which include lower portion of Salmon Glacier. From the Salmon River Valley the claims extend east over to the Big Missouri Ridge and then to Cascade River and Silver and Hog Lakes. The southeast portion of the property around No Name Lake features gently rolling topography. Pinnacle considers this area to be a potential site for a mill and associated infrastructure should the project be developed. Elevations on the property range from 500m in the Salmon River Valley to 1000m on the top of the Big Missouri ridge.

The deep, broad valley of the Salmon River is bordered by steep and extensively bluffed slopes, generally covered by glacial moraine and locally with thick alder and willow underbrush below the Granduc Road. Sparse stands of hemlock and minor spruce are present above the Granduc Road to the top of Big Missouri Ridge. Along the south side of the Big Missouri claim, an avalanche chute locally called "Slippery Jim" is covered with talus and landslide rubble and heavy alder brush. Along the ridges, small tarns, less than 100 meters in length occupy depressions.

6.0 HISTORY

6.1 Property History

This chapter discusses exploration and mining on the Silver Coin property from 1904 to the present.

The present Silver Coin property includes the historical Silver Butte (SB), Terminus and Silver Coin properties. The former Silver Butte property included the present Winer, Big Missouri and Kansas claims. The Terminus property was covered by Silver Coin 3 and 4 claims. The Silver Coin property included Silver Coin, Idaho, Idaho Fraction and Dan Fraction claims.

The bulk of the Silver Coin historical work was conducted on the former Silver Butte property by Esso Minerals Canada, Tenajon Resources and Westmin Resources in the period from 1979 to 1995. During that time extensive trenching, sampling, and drilling was followed by underground development and mining. Pinnacle has obtained most, but not all data from this work. FIGURE 6-1 represents the drillhole locations and drill traces for the known historic drilling, both surface and underground, for the Silver Coin Gold Project.

6.2 Early Years (1904 – 1939)

Although very little data is available from work done on the property in the early years (1904-1939), the following summarizes important activities.

Terminus Claim

According to the B.C. Ministry of Mines, mineralization was found on the Terminus Claim in 1911. In 1916, a Crown granted claim was established over the showings. During the 1930's a short adit was driven on some massive galena veins. Work on the property continued intermittently from 1911 but with little documentation.

Silver Coin Claims

The Silver Coin group of claims was located in 1904 along the Big Missouri Ridge. The property was purchased in the early 1930's by the Noble family, who held it until 2003. In the early 1930's a short adit was completed on the Dan showing. A number of pits were excavated on the property over mineralized showings, two of which were the Silver Coin and Idaho.

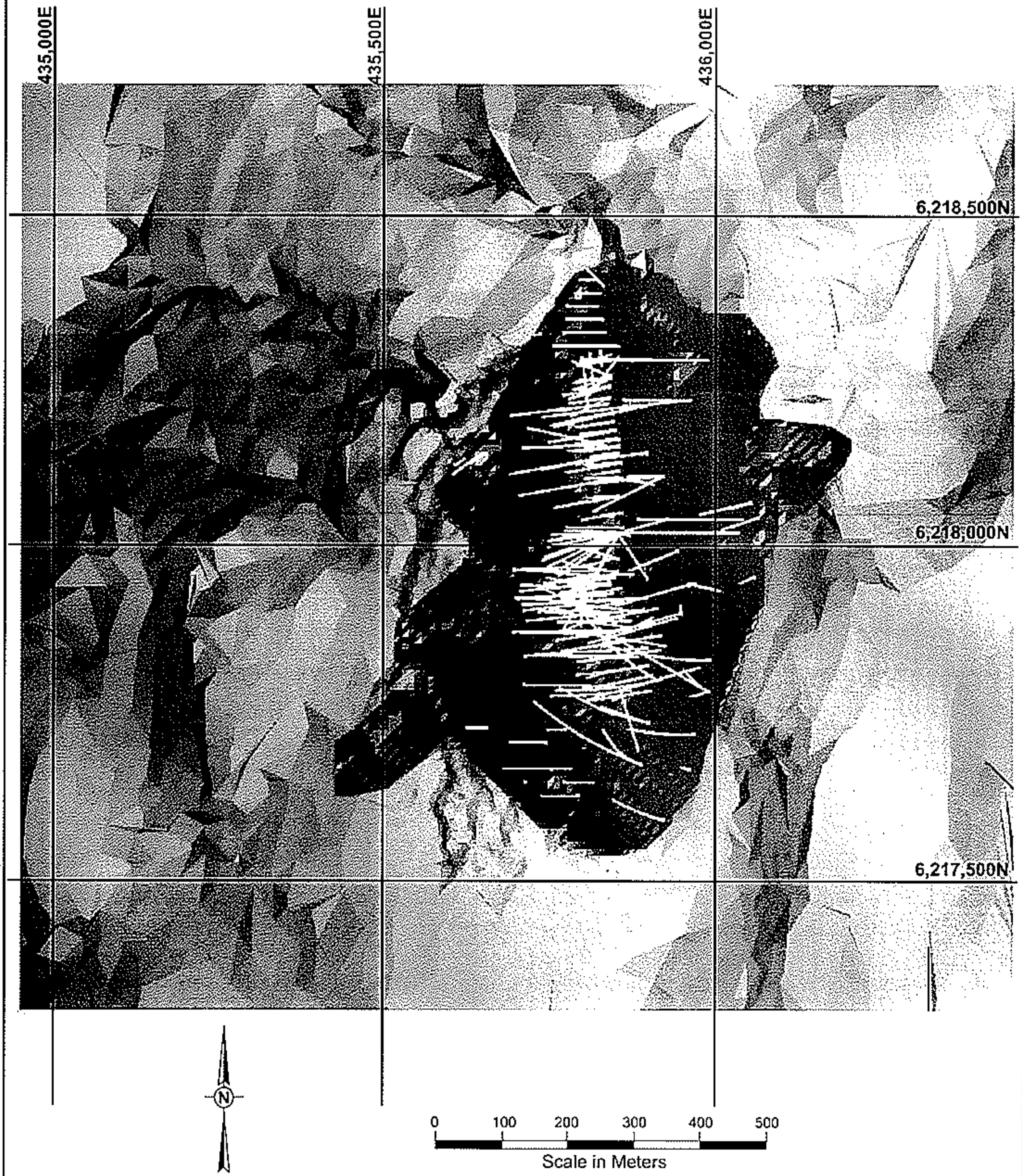
Silver Butte Property

1904 The Big Missouri claim was staked over a large mineral showing (most likely a present BM showing) on steep bluffs overlooking the Salmon River.

1911 An 18.3m crosscut was driven towards a large surface showing on the Big Missouri claim.

1914 A sample taken across a 13.72 m cut returned 3.42 g Au/t and 205.68 g Ag/t.

1915 The crosscut tunnel was extended 6.09m.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig6-1.cdr	Figure 6-1 Historic Drillhole Location Map
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

1916 A composite sample taken from 120 boulders of a large slide located on the Big Missouri claim gave an average grade of 4.45 g Au/t and 16 g Ag/t.

1930 Buena Vista Mining completed limited trenching on the Big Missouri claim.

1939 Buena Vista Mining conducted a surface sampling program on the Big Missouri claim. A series of surface samples near the west corner of the Big Missouri claim returned values averaging 14.39 g Au/t and 11.65 g Ag/t across a width of 16m.

Subsequent to this period, little or no work was completed until the late 1960s.

6.3 Recent Work (1967-2003)

6.3.1 Exploration (1967 through 2003)

Terminus Claim

In the early 1980s, the Terminus claim was purchased by Tournigan Mining, which subsequently sold it to Westmin Resources Ltd in 1983-84. Three vertical drillholes totaling 100 meters were completed in the early 1980's. Subsequent, soil sampling and airborne surveys including K-count radiometric surveys were completed over the Terminus claim as part of a larger exploration program on the Big Missouri property held by Westmin. The radiometric survey indicated that sericite alteration extended across the Terminus claim, south to No Name Lake. In addition, soil sampling indicated anomalous silver values south of the present workings. The claim was dropped in 2004 by Westmin and restaked the same year by Mountain Boy Minerals as the Silver Coin 3 and 4 claims.

Silver Coin Claims

1967 Prospecting by Granduc Mines located the area of the Dan showing. The caved adit was cleared and sampling and trenching on the showing was completed.

1981 E.W. Grove prepared a geological report on the property based on his visit to it in 1967.

Silver Butte

1969 Lockwood Survey Corporation conducted an airborne EM and magnetometer survey of the Salmon River Valley.

1971 El Paso Mining and Milling Company conducted a soil geochemical survey over the area of the Winer claim.

1975 Canex Placer Limited prospected the property area.

1978 Consolidated Silver Butte Mines Ltd. prospected and trenched the property. Two previously undiscovered mineralized outcrops were found.

1979 Consolidated Silver Butte Mines Ltd. conducted a widespread IP geophysical survey over the property.

1980 In the fall of 1980, Esso Minerals Canada Limited entered into an agreement to explore the Silver Butte property and completed a soil survey in that year over portions of the Big Missouri, Packers Fraction and Winer claims. A 400 by 500-meters soil grid was sampled along east-west lines located 100 meters apart. The samples were taken at 25m intervals except in the area overlying the geophysics anomaly where samples were taken at 10 meters intervals. The samples returned from 5 to 2600 ppb Au (287 ppb average), 1.1 to 27.2 ppm Ag (4.6 ppm average), 13 to 4320 ppm Pb (254 ppm average), and 27 to 2380 ppm Zn (284 ppm average)

1981 During the fall of 1981, Esso continued surface exploration consisting of geological mapping and sampling.

1982 Esso drilled 22 diamond drill holes totaling 1375m and excavated 17 trenches (the total length of the trenches is unknown). The soil survey area was extended and combined with other Esso soil surveys in the Salmon River valley. The combined survey contained approximately 1720 samples. Lloyd Wilson, an Esso Minerals geophysicist ran a test of induced polarization survey over the Winer claim. A total of 2km of lines were surveyed. A chargeability anomaly was measured over the heavy mineralization in the Face Cut #2 trench area (Facecut/35 Zone) and near diamond drill holes SB-15 and 16.

1983 A total of 1680m of diamond drilling in 14 holes and 210 meters of trenching in five trenches was completed. L. Wilson conducted an induced polarization survey over the Anomaly Creek – North Gully fault block. The1982 anomalies, near the Granduc Road (near drillholes SB-15 and 16) were confirmed in the 1983 survey. However, the anomalies decrease rapidly with depth. Down hole resistivity was tested in several holes from the 1982 drill program; namely holes SB 15,16,20,21 and 22. These drillholes showed a poor resistivity contrast down the hole. The possibility of a successful charged potential survey over the Facecut/35 Zone was considered small. The GENIE system was used to conduct an electromagnetic survey over the grid area. No anomalous responses were found.

1985 Esso purchased the Kansas Crown granted claim. Subsequently Tenajon Resources (formerly Tenajon Silver) entered into an option agreement with Esso whereby Tenajon could earn a 50% interest by spending $1,200,000.00 over a four-year period.

1986 Tenajon drilled four surface diamond drill holes totaling 996 meters.

1987 Tenajon conducted a surface diamond drill program totaling 3809.9m in 23 holes.

1988 The 1988 work program extended from January to early November. Work consisted of underground drifting and diamond drilling as well as surface work consisting of road building, diamond drilling, geological mapping and surveying. A total of 3063.8 meters of underground drilling was completed in 36 holes. Road construction included 2.9 kilometers of road building. A total of 4443 meters of surface drilling were completed in 23 drill holes.

1989 Tenajon conducted a drilling program which included 2826.5 meters in 15 surface holes and 1510.4 meters in 17 underground holes plus extensions in two of the 1988 underground holes.

1990 Tenajon completed 2544.9 meters of drilling in 16 surface holes and 899.4 meters in 16 underground holes. The same year Westmin Resources entered into an option agreement with Tenajon and subsequently completed

1833.7 meters of surface drilling in 13 holes and 643.3 meters in four underground holes as well as extensions of three previous holes.

1991 The Facecut-35 Zone was mined.

1992 No work was carried out on the property.

1993 Work included a major underground development followed by a program of underground drilling which totaled 1967 meters of AQ size core in 85 holes.

1994 Westmin continued a major program of underground development followed by 3507 meters of drilling in 62 underground holes from the new drift.

1995 Westmin initiated various ore reserve studies on the Kansas and West Kansas ore zones.

1996 Due to the closure of the Premier Gold Mine in April 1996, all activity ceased on the Silver Butte property.

2003 In October 2003, Uniterre Resources Ltd, which was the registered owner of the Big Missouri, Winer and Packers reverted Crown grants allowed them to expire. Subsequently, Mountain Boy Minerals staked these claims.

6.4 Exploration Drilling 2004-2008

Introduction

The drilling done on the Silver Coin Property in the period from 2004 to 2008 by Pinnacle and Mountain Boy Minerals totals 50,305 meters in 324 holes. The drilling was done exclusively from the surface and was concentrated on the Main Breccia Zone with much less drilling on Terminus, West No Name Lake, and the Road Zones. The purpose of drilling on the Main Breccia Zone was to expand the known mineralized zones as well as infill drilling. Most of the drilling was done on 20m centers.

A discussion of the Pinnacle-Mountain Boy drilling in the time period 2004-2008 is included in SECTION 11.0 of this report.

6.4.1 Underground Development and Bulk Sampling

Between 1987 and 1994, the previous operators of the property completed approximately 1,220 meters of drifting on three levels, 103.2 meters of crosscutting on one level and 130 meters of Alimak rising. Of this, 883 meters of drifting and 17 meters of sub-drifting on the Facecut Zone were completed on the 810 level, 250 meters of drifting on the 895 level with the remaining 70 meters of drifting on the 917 level. The two crosscuts were from the 810 level to the Facecut and 35 Zones.

In 1986 Tenajon collared and drove an adit 20m in overburden before abandoning it. In 1987 Tenajon collared an adit and completed 90m of drifting. During 1988 the drift was extended 773 meters on the 810 level with 63.5 meters of crosscut on the Facecut Zone, 39.7 meters of crosscut on the 35 Zone and 17 meters of sub-drift on the Facecut Zone.

The 1993 exploration program included a 19 meters extension of the 810 level, construction of an Alimak chamber, a 130-m long Alimak raise at 50 degrees to reach the target area, 63 meters of sublevel drift and crosscut at the 895m elevation and 70m of sublevel drift and crosscut on the 917m elevation. Development muck from the upper part of the Alimak raise, and initial rounds of the sublevels taken from the Alimak deck, comprised the first bulk sample of 1,107 dry tonnes. The second bulk sample consisted of 1,540 dry tonnes of development muck from the combined sublevels. In 1994, a major program of underground development, included 168m of development drifting on the 895 sublevel at the south end of the drift developed in 1993. Development muck totaling 1,481 tonnes from the sublevel was stockpiled at the portal and then milled at the Premier Gold mill later in the year. Assay grade of this bulk sample is unknown. Location of underground workings is shown on FIGURE 6-2.

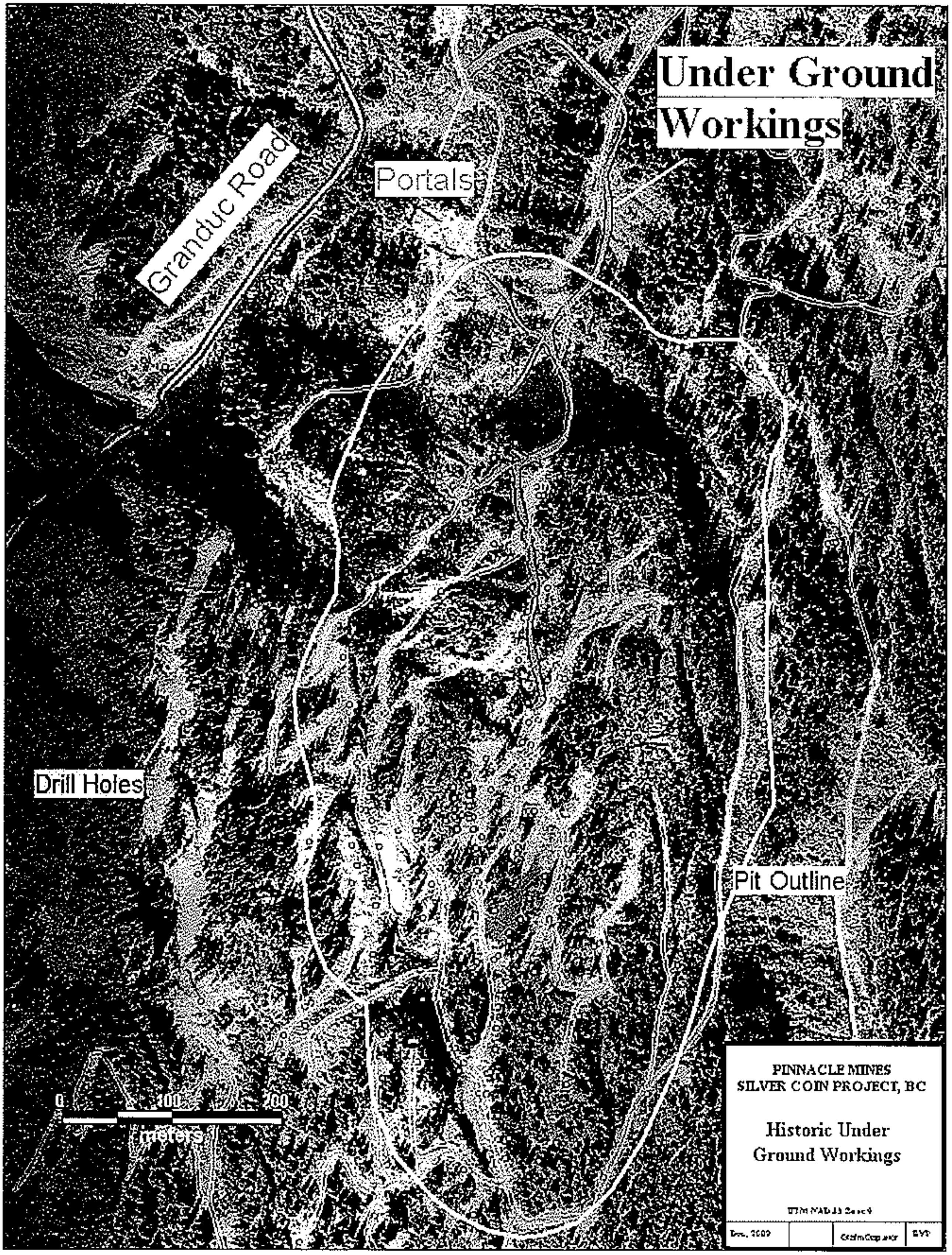
6.5 Historical Production

During the 1930s, a short adit was driven on massive galena veins on the Terminus Zone, in the area of present Silver Coin-2 claim. Work continued intermittently with little documentation. Also in the early 1930s, a short adit was driven on the Dan Zone in the area of the Dan Fraction claim. Several small open pits were excavated on the property, including pits on the Silver Coin and Idaho Zones.

In 1911, a crosscut was driven for 18 m towards a large surface outcrop of mineralisation on the Big Missouri claim (BM Zone) and in 1915 the cross cut was extended a further 6 m.

In 1991, Westmin Resources mined the Facecut-35 Zone producing 102,539 tonnes at an average grade of 8.9 g Au/t and 55.50 g Ag/t. Mining was primarily by sublevel retreat with a minor amount of benching. Base metal rich – low gold sections of the Facecut/35 Zone were not mined. No base metal values were recovered as the ore was processed using a cyanide leach process at the Premier Mine mill five km south of Silver Coin. Recoveries averaged 92.9% for gold and 45.7% for silver.

Westmin estimated (Lhotka P. 1991 – draft report) that 111,000 tonnes of material grading 0.61 g Au/t, 29 g Ag/t and 3.46% Zn were directed to the tailings pond. Sampling in 2004 by Mountain Boy Minerals and Pinnacle indicated that the mine tailings from the Facecut-35 Zone averaged 0.72 g Au/t, 31.2 g Ag/t, 0.388 % Cu, 0.48 % Pb and 3.61 % Zn in two samples.



Issued by:

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Drawing Provided by/Prepared for:	File Name:
Pinnacle Mines Ltd.	Fig6-2.cdr
Project: Silver Coin Gold Project	Project Number: 114-311007
Project Location: Stewart, British Columbia	Date of Issue: 01/07/2009

Figure 6-2
Location of Historic Underground Workings

7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The Silver Coin property is centered on Big Missouri Ridge within the western boundary of the Triassic to Jurassic Bowser Basin about 24 kilometers east of the Coast Crystalline Complex. Much of the property is underlain by Triassic-Jurassic basin-filling sedimentary and volcanic rocks of the Stuhini Group, Hazelton Group and Bowser Lake Group. These rocks have been metamorphosed to greenschist facies and have been intruded by plutons of both Mesozoic and Cenozoic age.

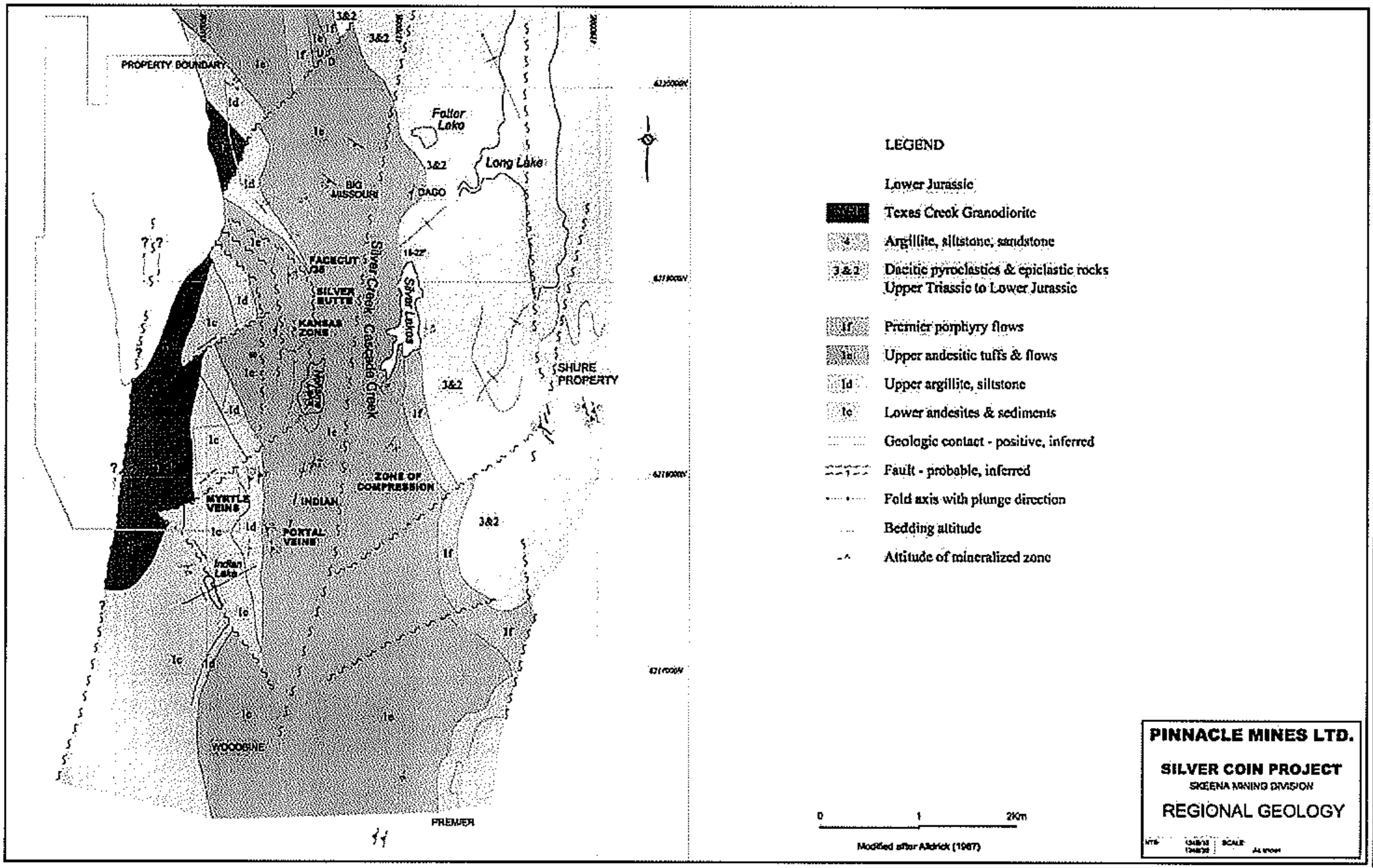
Much of the information presented here on the regional geologic setting and property geology has been excerpted or modified from public domain and proprietary reports by Alldrick (1988 and 1993), Greig (1994a and 1994b), Grove (1971, 1981, and 1986), Stone and Godden (2007), Walus (2009) and Mazur (2006). Entire sections of these reports have been included with little modification in the present report. The reader is referred to the original reports and to the Appendices for additional details. The report by Alldrick (1993) on the geology (FIGURE 7-1) and metallogeny of the Stewart mining camp is especially thorough and is an excellent resource for the geologic history and mineralization in the area.

C.F. Greig (1994a and 1994b) mapped the Stewart area for the Geological Survey of Canada and assigned rocks in portions of the district to the Stuhini Group of Triassic age. The Stuhini Group rocks either underlie or are in fault contact with the rocks of the Jurassic Hazelton Group. These Triassic rocks consist of dark-gray laminated to thick-bedded silty mudstone and fine- to medium-grained and some coarse-grained sandstone. Locally, the Stuhini Group also includes thick-bedded heterolithic pebble to cobble conglomerate, thick-bedded sedimentary breccia, and massive tuffaceous mudstone. Regionally the Stuhini includes pyroxene basalts, basaltic andesites and feldspar-porphyritic volcaniclastic rocks (Alldrick, 1993).

Extensive exposures of Hazelton Group rocks in the western portion of the Bowser Basin have been named the Stewart Complex (Grove, 1986). This complex forms a north-northwesterly trending belt extending from Alice Arm to the Iskut River. The Unuk River Formation is the lowest member of the Hazelton Group. This unit consists of at least 4,500 meters of Lower Jurassic marine and non-marine volcaniclastics. These volcanic rocks consist of monotonous green andesitic rocks including ash and crystal tuff, lapilli tuff, pyroclastic breccias and lava flows. Regionally, feldspar-porphyritic andesite flows and tuffs are recognized at the top of the formation and two siltstones form important stratigraphic markers within the formation.

The upper unit of the Unuk River Formation is termed the Premier Porphyry Member and is texturally similar to dikes of Premier Porphyry which cut the underlying strata and the Texas Creek batholith (Alldrick, 1993). The Premier Porphyry Member regionally includes tuffs and flows with variable phenocrysts species, notably hornblende, plagioclase and K-feldspar. Minor sandstone regolith and vent breccias are locally present. Alldrick (1993) states that the Unuk River Formation is the host for all of the major gold deposits of the Stewart mining camp and that the deposits around the Silback Premier and Big Missouri mines occur stratigraphically below the Premier Porphyry Member. The Unuk River Formation is interpreted to represent a predominantly subaerial composite andesitic stratovolcano.

In the area of the Silver Coin property, the Unuk River Formation is overlain at steep discordant angles by the lithologically similar Betty Creek Formation which is middle Lower Jurassic in age. The Betty Creek Formation represents a second cycle of trough filling consisting of a sequence



Issued by:	Drawing Prepared for/Provided by:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig7-1.cdr	Figure 7-1 Geologic Map of the Silver Coin Property and Surrounding Area
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

of distinctively colored red to green epiclastic rocks with interbedded tuffs and flows which range in composition from andesite to dacite. The thickness of the Betty Creek is quite variable regionally from 4 to 1,200 meters.

The Unuk River and Betty Creek Formations are in turn unconformably overlain by a thin felsic tuff horizon of upper Lower Jurassic age (approximately 185-190Ma) termed the Mt. Dilworth Formation. This formation is a 20 to 120m thick sequence dominated by variably welded dacite tuffs. Hard, resistant exposures of the Mt. Dilworth Formation are commonly pyritic and form gossanous cliffs. This formation is an important stratigraphic marker in the Stewart area. Alldrick (1993) described five members of the Mount Dilworth Formation including the Lower Dust Tuff Member, the Middle Welded Tuff Member, the Upper Lapilli Tuff Member, the Pryritic Tuff Member, and the Black Tuff Member. The Pyritic Tuff member has been interpreted to represent pyrite impregnation around fumarolic centers and brine pools.

The entire sequence just described is unconformably overlain by non-marine sediments and minor volcanics of the Middle Jurassic Salmon River Formation. This formation includes a thick package (at least 300 meters) of complexly folded, banded, predominantly dark-colored siltstone, greywacke, sandstone with intercalated calcarenites, minor limestone, argillite, conglomerate, littoral deposits, volcanic sediments and minor flows. The basal unit of the Salmon River Formation is a pyritic limestone.

The Upper Jurassic Bowser Lake Group overlies the Hazelton Group rocks described above. The Bowser Lake Group is exposed in the westernmost portion of the Bowser Basin and is also found as remnants on mountain tops in the Stewart area immediately to the west. These rocks consist of dark grey to black silty mudstone and thick beds of massive, dark-green to dark-grey, fine- to medium-grained arkosic sandstone. Chert-pebble conglomerates are characteristic of the Bowser Lake Group in the type locality northeast of the Silver Coin area (Alldrick, 1993).

D. Alldrick (1988) has interpreted several volcanic centers of Lower Jurassic age in the area north of Stewart, B.C. Volcanic centers within the Unuk River Formation are located in the Big Missouri-Silbak Premier area and in the Brucejack Lake area. Volcanic centers within the Lower Jurassic Betty Creek Formation are present in the Mitchell Glacier and Knipple Glacier areas. Alldrick (1993) also identified a stratovolcano at Mount Dilworth, five kilometers north of the Silver Coin property. Alldrick mapped flows of the Premier Porphyry Member, in the Silver Coin area. This member marks the top of the Unuk River Formation and intrusive phases of the Premier Porphyry include dikes that cut all the underlying rocks including the Early Jurassic-age Texas Creek Batholith. Alldrick's work suggests that all gold deposits in and around the Silbak Premier and Big Missouri mines occur in rocks that are stratigraphically below the Premier Porphyry Member.

Various intrusives occur in areas underlain by Early Jurassic and Tertiary rocks. The granodiorite bodies of the Coast Plutonic Complex largely engulf the Mesozoic volcanic rocks on the west. To the east, there are numerous smaller intrusions which range in composition from monzonite to granite including highly felsic varieties. Some of these likely represent late phases of the Coast Plutonic Complex of middle Cretaceous age; others are probably genetically related to the Jurassic volcanic rocks that were deposited in the western portions of the Bowser Basin.

The granodioritic Texas Creek Plutonic Suite (TCPS) in the Stewart area is Jurassic in age (Alldrick, 1993) with isotopic dates ranging from 211 to 186 Ma. This suite typically is coarse grained with abundant hornblende and locally very coarse K-feldspar phenocrysts. The TCPS includes the foliated Premier Porphyry dikes which are thought to be the intrusive equivalents of the Premier Porphyry Member of the Unuk Formation. The dikes are closely related to all of the major ore zones at the Silbak Premier mine; are altered to chlorite, sericite and carbonate; are andesitic in composition; and have sericite-chlorite-quartz pressure shadows adjacent to euhedral pyrite indicating post-pyrite deformation under greenschist facies metamorphic conditions.

Other intrusives are Tertiary in age with a spike in activity from 45 to 55 million years (Armstrong, 1988). This Eocene suite, termed the Hyder Granodiorite Suite (HGS), is characterized by lack of alteration, medium grain size, equigranular texture, presence of biotite, and accessory sphene. The Hyder Suite rocks regionally host major molybdenum deposits such as the Quartz Hill deposit in southeast Alaska and minor deposits of silver, lead, gold, zinc, and tungsten. Tertiary HGS dike swarms are common and range in composition from granodiorite and aplite through lamprophyre. Two of these swarms represent approximately 1.5 kilometers of northeast-southwest extension. Alldrick (1993) states that the dikes cut regional folds but are offset by most of the major and minor faults in the Stewart area.

Early deformation in the Silver Coin area is related to Triassic-Jurassic subduction and docking of several terranes. The various terranes comprising the Canadian Cordillera were probably assembled by late Jurassic time.

By the middle Cretaceous an Andean type magmatic arc had developed along the continental margin above an east-dipping subduction zone (Alldrick, 1993). Transpression from 90 to 70 Ma gave rise to right lateral-strike slip faults such as the Tintina Fault with hundreds of kilometers of displacement. An Eocene volcanic arc developed in the Coast Plutonic Complex from 60 to 40 Ma. Localized plutonism and volcanism developed from 40 to 20 Ma with generally small stocks and dikes. This intrusive activity was controlled by north to northeast striking extensional normal faults. East-dipping subduction and sporadic basaltic volcanism resumed from 20 Ma to the present.

Doubly plunging, northwesterly-trending synclinal folds with steep axial surfaces have developed in the Salmon River and underlying Betty Creek Formations in the Silver Coin area. These folds are locally disrupted by small west-directed thrust faults which strike parallel to the major fold axes. Steeply dipping strike-slip faults trend at high angles to the trend of the fold axes. Alldrick (1993) noted the strong regional contrast in fold geometries between the Hazelton Group, which is characterized by open cylindrical folds, and the overlying Salmon River Formation, which occupies synclinal (basinal) cores and shows tight disharmonic folds.

Five sets of major faults in the Stewart area were defined by Alldrick (1993). These include: "north striking sub-vertical shears, northerly striking west-dipping shears, southeast to northeast-striking 'cross structures' that cut the northerly structural grain, decollement surfaces or bedding plane slips that are present near the base of the Salmon River Formation, and mylonite zones." He also proposed that the regional faults were originally "ductile contractional reverse faults and were reactivated as brittle fractures during later extensional episodes".

Mylonite zones have developed in the Texas Creek batholith and these parallel similar mylonites in the country rock. Mylonites are present in the banded sulfide zone at the Silbak Premier mine and a southeast-striking set of these deforms Jurassic ore and localizes Tertiary ore at the Riverside mine. Alldrick (1993) describes foliation envelopes that have developed along both ductile and brittle faults with early foliations cut by those related to later faults. Flattened clasts defining a foliation are common in tuffs indicating ductile deformation along probable east-verging reverse faults (Alldrick, 1993). These early reverse faults were later reactivated during Tertiary intrusive activity: doming and extension resulting in west-dipping normal faults with relict ductile fabrics.

Alldrick (1993) summarized the radiometric dates that have been obtained in the Stewart area. These data indicate late Triassic to Early Jurassic (211 to 160 Ma) volcanism, emplacement of subvolcanic plutons and dikes, and deposition of turbidites; deformation and regional greenschist metamorphism with a thermal peak at approximately 110 Ma; and emplacement of stocks and dikes of the Coast Plutonic Complex between 55 and 20 Ma.

Alldrick (1993) describes the results for lead isotope studies on minerals from mines in the Stewart area. The data suggest that all of the deposits are Phanerozoic and probably post-Paleozoic and they represent two distinct mineralizing events. The two events were regional in scale with strong geographic overlap and are separated by a regional metamorphic event of Cretaceous age. The two mineralizing events correspond to a Jurassic gold-silver-lead-zinc-copper mineralizing event genetically related to the calc-alkaline Hazelton Group volcanic rocks and an epigenetic silver-lead-zinc event and molybdenum/tungsten occurrences related to Eocene granodioritic plutons of the Hyder Plutonic Complex. The two intrusive and mineralization events and the metamorphic event are supported by lead isotope results and by U-Pb and K-Ar dates.

Alldrick (1993) presented a detailed description of many of the deposits in the Stewart area. Deposit categories include gold-pyrrhotite shear-hosted veins formed adjacent to subvolcanic intrusions, silver-gold-base metal veins and breccias emplaced in faults and hydrothermal breccia zones at shallow levels in the volcanic pile, and generally barren fumarolic pyrite hosted by dacite of the Mount Dilworth Formation. Alteration adjacent to the veins and breccias passes from silicification outward into sericitic alteration, carbonate alteration, and finally to chloritic alteration farthest from the veins. The veins are zoned, generally with higher gold values and higher sulfide contents in the deeper portions.

7.2 Property Geology

Introduction

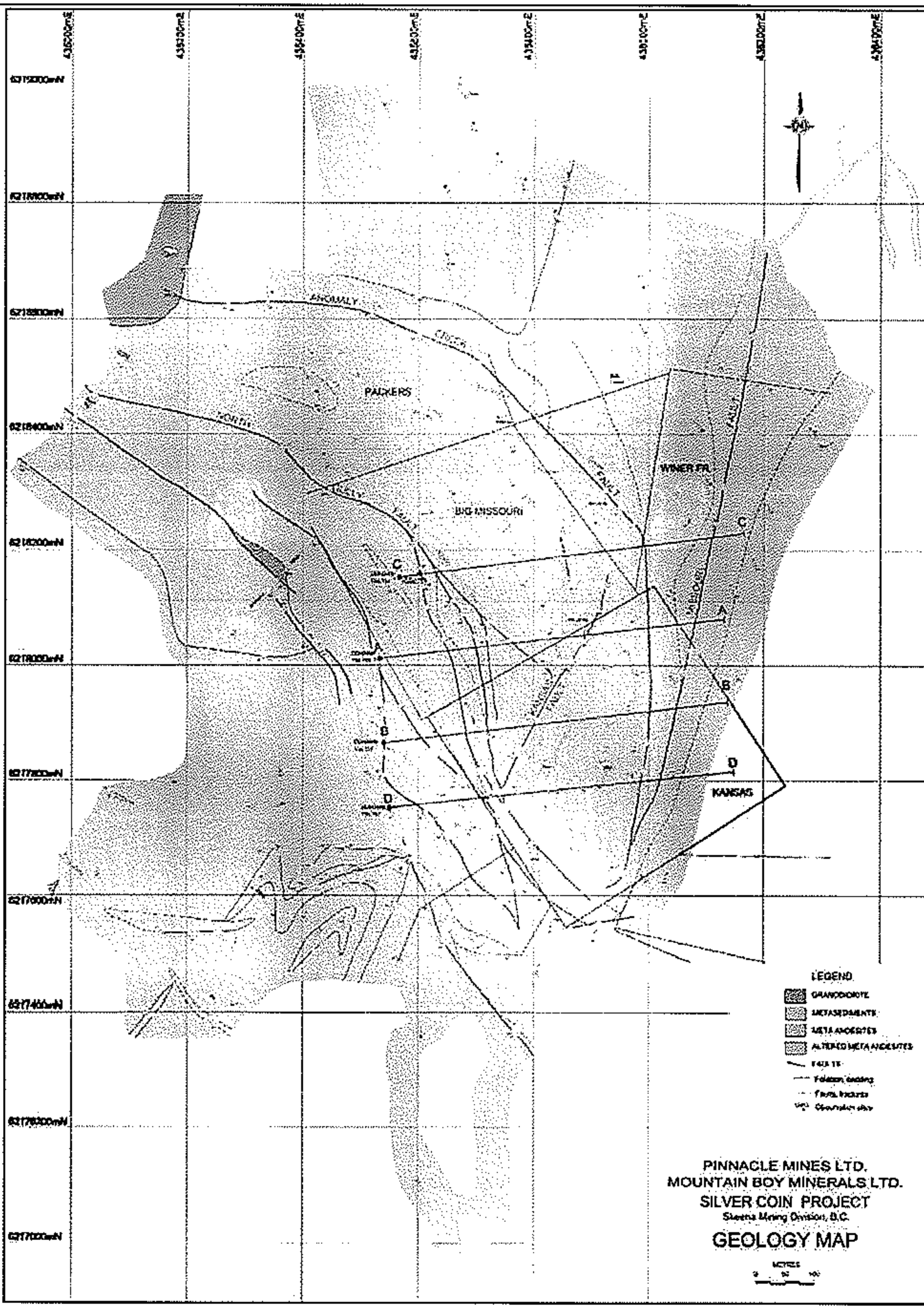
The geology of the Silver Coin deposit is complex and remains a matter of debate, in spite of at least four separate geologic mapping campaigns. The principal difficulties stem from the massive nature of most of the stratigraphic units, the lack of reliable stratigraphic and structural marker horizons, and subtly different rock types that have been subjected to various and multiple stages of alteration, metamorphism, deformation, and mineralization. Available geologic information was developed by several generations of operators over a period of many years resulting in a lack of continuity between the various geologic data sets. The property was mapped by Britten (1988), Alldrick (1993) and later by Mazur (2006) (FIGURE 7-2). Geological maps and interpretations produced by these authors show significant differences in geological interpretation.

The biggest obstacle in interpreting the geology of the property has been recognition of the primary lithologies in the andesitic rocks. A report on the property by Westmin (Lhotka et al, 1994) states: "*Recognition of primary lithologies is difficult in the drift due to alteration and recrystallization. Frequently, the primary geologic unit mapped in the drift does not match that logged at the collar of the drill holes drilled from the drift.*"

Lithology and Geologic History

North-south-striking faulting has divided the Silver Coin property into three different geologic areas:

- an area to the east of the claim group that is bounded by the Cascade Creek fault zone;
- an area located between the Cascade Creek fault zone and the next north-south oriented fault (located about one km to the west) that is dominated by andesitic volcanic rocks with minor trachyte; and



Issued by:

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Drawing Provided by/Prepared for:	File Name:
Pinnacle Mines Ltd.	Fig7-2.cdr
Project: Silver Coin Gold Project	Project Number: 114-311007
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 7-2
Geology of the Silver Coin Property

the central portion of the claim block where northwest-trending faults have created a graben that hosts mineralized zones.

The sequence of predominantly andesitic volcanic and volcaniclastic rocks which constitutes the fault blocks described below was subsequently cut by numerous intrusive bodies of subvolcanic, porphyritic andesite and less numerous bodies of aphanitic dacite.

Along with other rocks from the Stewart area, the volcano-sedimentary rocks of the Silver Coin property underwent a period of regional lower greenschist facies metamorphism characterized by the presence of sericite, chlorite, carbonate and pyrite. In surface exposures, rocks that underwent regional metamorphism tend to have green color - in contrast to altered rocks that tend to be light-grey and yellow. Despite this, distinguishing between mineral assemblages formed during regional metamorphism and altered rocks is difficult, not least because the two assemblages often occur together. It is probably for these reasons that previous authors working on the Silver Coin property did not differentiate between regional metamorphic and alteration mineral assemblages.

To the south of the graben, Texas Creek granodiorite and andesitic pyroclastic rocks crop out on the former Silver Coin Crown Granted claims (Stone and Godden, 2007). Foliated andesite is the most common rock type, with only a few outcrops of sheared limey argillite. The main features in the Silver Coin project area are lineaments striking northwest and northeast, which strongly influence the topography over most parts of the former Silver Coin property. The lineaments are interpreted as zones of intense fracturing, probably with shearing on the N20°W set and possibly on the N25°E set.

The eastern portion of the Silver Coin property, immediately to the west of the Cascade Creek fault, contains a silicified and mineralized cataclasite zone that is up to 75 meters wide, hosted within andesitic volcanic rocks, carrying three to five percent disseminated euhedral pyrite. The mineralized zones occur along a regional deformation zone extending from the former Big Missouri mine through the Silver Coin 3 and 4 claims and south towards No Name Lake. This regional structure consists of fractured country rock that is intricately laced with unevenly spaced quartz-calcite veinlets and stringers, with or without sulfides. The western portion of the claim block is underlain by Texas Creek granodiorite that intrudes the volcano-sedimentary rocks to the east.

The last major geologic event in the area of the Silver Coin property was emplacement of the Jurassic granodioritic Texas Creek batholith (Alldrick, 1993) which underlies most of Silver Coin 9 and 10 claims as well as the Indi claims. Apophyses derived from this batholith intruded the metamorphosed Jurassic-Triassic volcano-sedimentary rocks along the Anomaly Creek fault system. One porphyritic phase of this intrusive sequence has been routinely referred to in drill logs by Premier Mines and on the Silver Coin property as the Premier Porphyry. Alldrick (1993) mapped flows in the Salmon River Valley as Premier Porphyry and these are thought to be extrusive equivalents of intrusive phases of the Premier Porphyry. Recognition of Premier Porphyry is important because this rock is interpreted to represent the source rock for mineralization in the nearby Premier Mine and possibly at Silver Coin.

A petrographic study revealed the widespread presence of trachyte within the Perseverance zone. Kruchkowski (2005, 2006) is of the opinion that trachyte intruded the area along the North Gully and Anomaly Creek faults and that it was a source of mineralization for the zones located in the graben area.

Structure

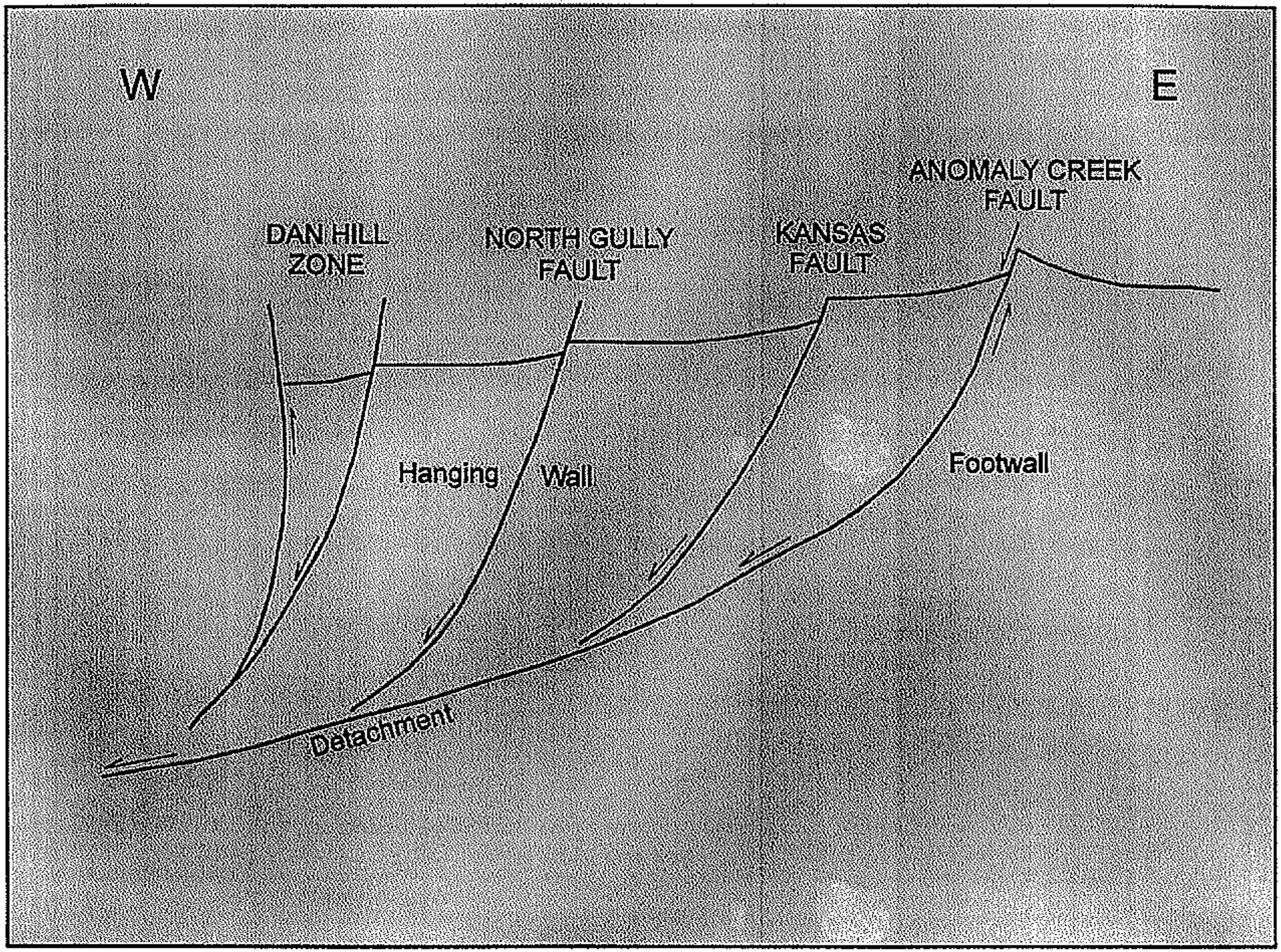
The structure of the Silver Coin property was studied in detail by Melnyk and Britten (1989), Alldrick (1993), and more recently by Mazur (2006). Doubly plunging, northwest-trending folds of the Salmon River and Betty Creek Formations dominate the structural setting of the Silver Coin area. The folds are locally disrupted by faults. These later structures include: small thrusts with trends parallel to the major fold axes, cross-axis steep wrench faults which locally drag beds, selective tectonization of tuff units, and major northwest faults.

According to Mazur (2006), the dominant structural feature of the Silver Coin property is the Anomaly Creek Fault, which he interpreted to have acted as a master detachment fault. This interpretation is outlined in the Appendices of this report. The Mazur report is relatively comprehensive and detailed and incorporates the work of the earlier authors. The structural interpretations of the earlier workers can be found in the references cited at the end of this report. Strongly deformed, altered and mineralized Jurassic-Triassic rocks between the Anomaly Creek fault and the subsidiary North Gully fault have been termed the "Main Breccia Zone". This zone is at least one Km long and 200-300 meters wide and hosts the bulk of gold mineralization on the Silver Coin property. This master (detachment) fault and related subsidiary listric faults in the hanging wall have progressively dropped the hanging wall to the southwest as shown in the schematic cross section (FIGURE 7-3)

The mineralized zones of the Kansas and Big Missouri claims are part of a major mineral trend that strikes north-south and hosts the Big Missouri and Indian mines. In the area of the Perseverance, Kansas, Facecut and 35 mineralized zones, the (major) structure is joined by three large, sub-parallel and northwest striking faults that have moderate dips to the west (the Anomaly Creek, Gully and North Gully faults).

The Anomaly Creek fault has been interpreted as a right-lateral, oblique-slip structure of unknown displacement. The Gully fault has been interpreted as a reverse fault, the displacement of which is probably not large (the alteration zones on both sides of the fault do not appear to be significantly offset). The nature of movement on the North Gully fault is not well understood since little work has been done across the areas in which the structure is developed. Reverse movement for this fault was implied (but not proven) by Melnyk and Britten (1989).

There are two prominent sets of foliations at Silver Coin. One set strikes east-southeast to east-northeast and is steeply dipping. A second, more widespread set trends north-south and dips moderately to the west.



Issued by:	Drawing Prepared for/Provided by:	File Name:
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig7-3.cdr
	Project: Silver Coin Gold Project	Project Number: 114-311007
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 7-3
Schematic Cross Section Through the Silver Coin Mineralized Zones

8.0 DEPOSIT TYPES

Integration of Regional and Property Geologic Work

Much excellent geologic work has been done both regionally and in the immediate area of the Silver Coin property and some of this work is cited elsewhere in this report. As might be expected, significant differences in interpretation exist between the various workers who have studied the area. The following discussion highlights some of differences where their resolution has implications for exploration in the Silver Coin area.

Mazur (2006) cited evidence suggesting that the alteration and mineralization at Silver Coin post-dates and overprints the regional greenschist metamorphism. He also suggested that mineralization was syngenetic with Jurassic plutons and dikes and that the dikes and the related mineralization are syntectonic.

Mazur further suggested that the Main Breccia Zone and related mineralized zones were controlled by a master listric normal fault, the Anomaly Gulch fault, and genetically related faults in the hanging wall of the master fault. These faults were proposed to have developed along a major, northwest-trending right-lateral strike slip zone. The extensional model of Mazur (2006) appears to explain the graben (or half graben) in the area of the Main Breccia but does not appear to address some of the contractional faulting described by earlier workers.

In contrast, Alldrick (1993) and Melnyk and Britten (1989) suggest early reverse faulting was followed by extensional faulting. Alldrick (1993) cites as evidence brittle faults that have relict ductile deformation features in the immediate wall rocks. Reactivation of Jurassic faults would seem likely due to Cretaceous age subduction and thrust faulting followed by Eocene extension. Alldrick also concluded that the gold-silver-base metal mineralization in the area pre-dated the regional greenschist metamorphism.

The complex geometry of many of the mineralized zones and extensive multiphase breccia development would suggest a complex history of deformation along the controlling faults, probably involving early reverse faults with later oblique and normal displacement. The strongly arcuate, concave to the southwest pattern of the 20 mineralized zones on the property suggest the possibility that the mineralization was emplaced along a ring fracture or cone sheet system. Although the andesitic volcanic center proposed by Alldrick (1993) is north of the Silver Coin property, the emplacement of slices of the Texas Creek granodiorite in the western part of the property along similar concave faults suggests that these curved faults controlled emplacement of both the mineralization and the intrusives that are thought to be the source of the mineralization. The arcuate faults are concave toward the main mass of the Jurassic Texas Creek granodiorite to the west-southwest. Deflation of a volcanic/intrusive center above the Texas Creek granodiorite southwest of the property might account for development of the half graben bounded by the mineralized arcuate faults on the property. These structures were probably modified and offset during Jurassic subduction; Cretaceous contractional deformation, subduction and transpression; and later extension during the Tertiary.

Grove (1986) found that deformation and foliation development was initiated prior to emplacement of the extensive Jurassic plutons but that the foliation is best developed near the intrusives. Large cataclasite zones are cut by the plutons. Grove also described development of banding and low temperature recrystallization in the Unuk River and Cascade Creek cataclasite zones in the Silver Coin area. This may explain why Mazur (2006) describes the mineralization as post-metamorphic whereas Alldrick (1993) describes the mineralization as pre-metamorphic.

Alldrick's (1993) suggestion that a marine facies of the strongly pyritic Mount Dilworth Formation and immediately overlying sediments might be prospective for Kuroko type massive sulfides holds additional promise in the Silver Coin area.

The bulk of the evidence at the Silver Coin property supports a Jurassic age for the mineralization, synchronous with development of one or more andesitic volcano complexes and emplacement of shallow plutons and dikes related to the Texas Creek granodiorite and the Premier Porphyry. There also appears to be good evidence for a graben, bounded on the east by the Anomaly Creek fault. Mineralization was emplaced along faults and it is likely that some of these structures have been the locus of later movement that has brecciated, remobilized and displaced the mineralization.

Proposed Genetic Model

Pinnacle's current working theory involves two mineralizing events with at least two (and probably several) periods of faulting. In simplest terms, the initial mineralizing event is believed to be Jurassic-age Kuroko type base metal mineralization. Alldrick (1993) cites substantial supporting evidence for this proposal from lead isotope geochronology on galena samples coming from and near the Silver Coin property. The Facecut/35 mineralization mined in the 1980's was essentially a massive sulfide deposit, likely a preserved sulfide body enriched in gold by a later mineralizing event. A Tertiary mineralizing event, also supported by lead age dating, is reported by Alldrick (1993) for some nearby deposits; although there are no samples of this age from Silver Coin. Pinnacle speculates that this later event may have introduced gold and remobilized the earlier mineralization.

Alldrick discusses regional extensional and compressional tectonic events and Pinnacle believes that some of the same structures were active in both of these tectonic periods. The principal faults at Silver Coin are north-striking shallow west-dipping structures that were probably active at least twice. The photo in FIGURE 14-4 shows a mylonite zone in the middle of a brittle-fractured fault, which we interpret to show an earlier ductile deformation event that was later reactivated under brittle conditions.

The morphology of the mineralization at Silver Coin is a north-trending sub-horizontal crudely cylindrical body (FIGURE 8-1). In a compressional or extensional environment with stacked and upward-curving faults, a north-trending, sub-horizontal dilatational environment may have developed between the faults. This would have provided a favorable environment for bulk disseminated gold mineralization. Oblique "ladder" type veins, in this case sub-vertical could have provide local higher grade mineralization as exploited via the historical underground workings. This would also explain the different drill results evidenced by underground versus surface drilling.

The proposed structural mechanism that explains the "tubular" shape of the mineralized body may limit prospects for exploration success from deeper drilling. However, additional mineralization may occur elsewhere in the thrust environment within several of the other stacked faults mapped in the deposit area further to the east or west.

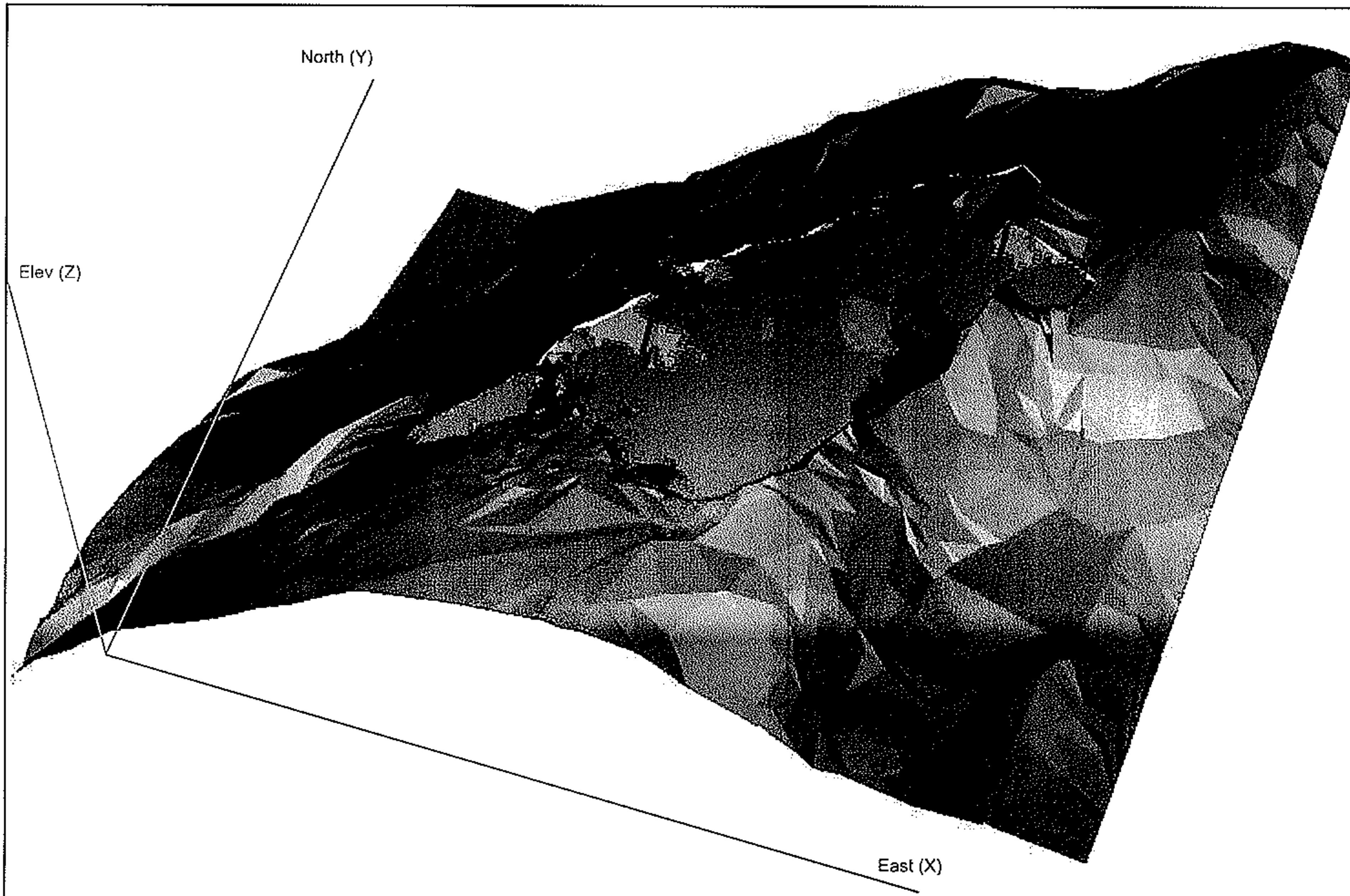
Proposed Structural Model

Much of the existing thinking on the geology of the deposit is based on work done by S. Mazur in 2006. His mapping and interpretation was heavily influenced by an overarching geologic interpretation that the structural geology is dominated by west-dipping detachment faulting and the associated listric normal faulting typical of such an environment. Mazur's assumptions are evident in one of his cross sections, included below as FIGURE 8-2. Mazur's recognition of extensional normal faulting at Silver Coin is a significant contribution to the understanding of the deposit. However, Pinnacle believes that Mazur's interpretation may have been carried beyond what the data support and the following discussions offer some different interpretations of the geology and structure.

Pinnacle concluded that Mazur's fault interpretation and perhaps his broader structural model, does not adequately honor the faults mapped in the drill core. For example, the sharp inflections shown on Mazur's interpreted faults are unsupported by drill data.

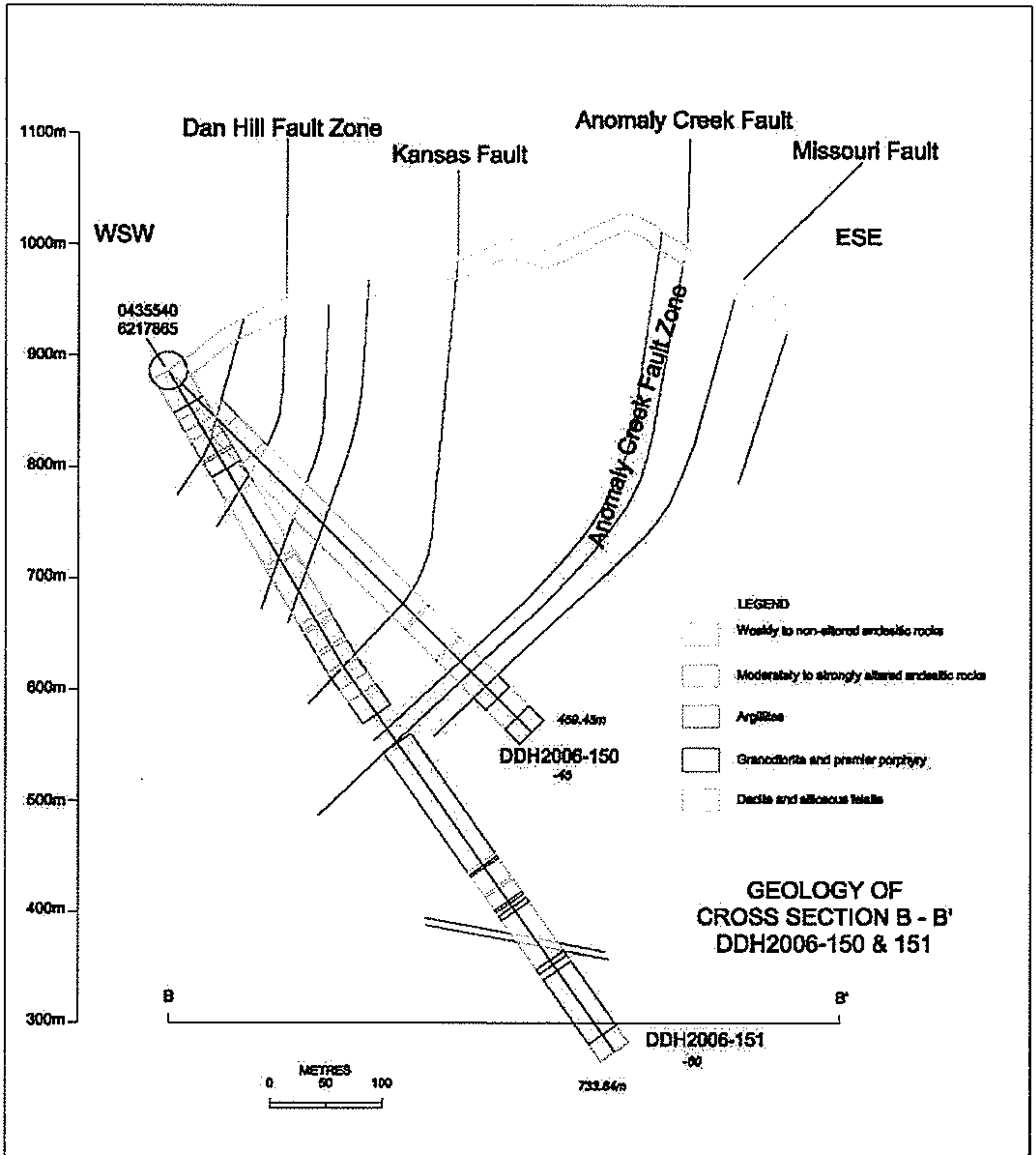
After generating full new sets of vertical cross sections and level plans at 20m spacing, Pinnacle undertook a reinterpretation of the faulting based more rigorously on the faults mapped in drill core. This effort started with a section by section interpretation of what appears to be a through-going north-striking west-dipping controlling structure ("Ore Controlling Fault (FIGURE 8-3). This fault is apparent in the majority of holes that traverse the zone and more importantly, it seems to be a lower boundary to gold mineralization. This fault was mapped section to section and then projected onto the level plans to generate a 3D surface. Pinnacle then generated a plausible series of nearly parallel stacked faults that step westward to the limit of the drill data. These were similarly mapped onto level plans and digitized to produce a series of 3D fault surfaces usable in MicroMine® software (FIGURE 8-1).

There is some potential for bias in this exercise in that the majority of the surface drill holes are east-directed and the predominant theory is that the structures are west dipping. Never-the-less, the data seem to support this interpretation.

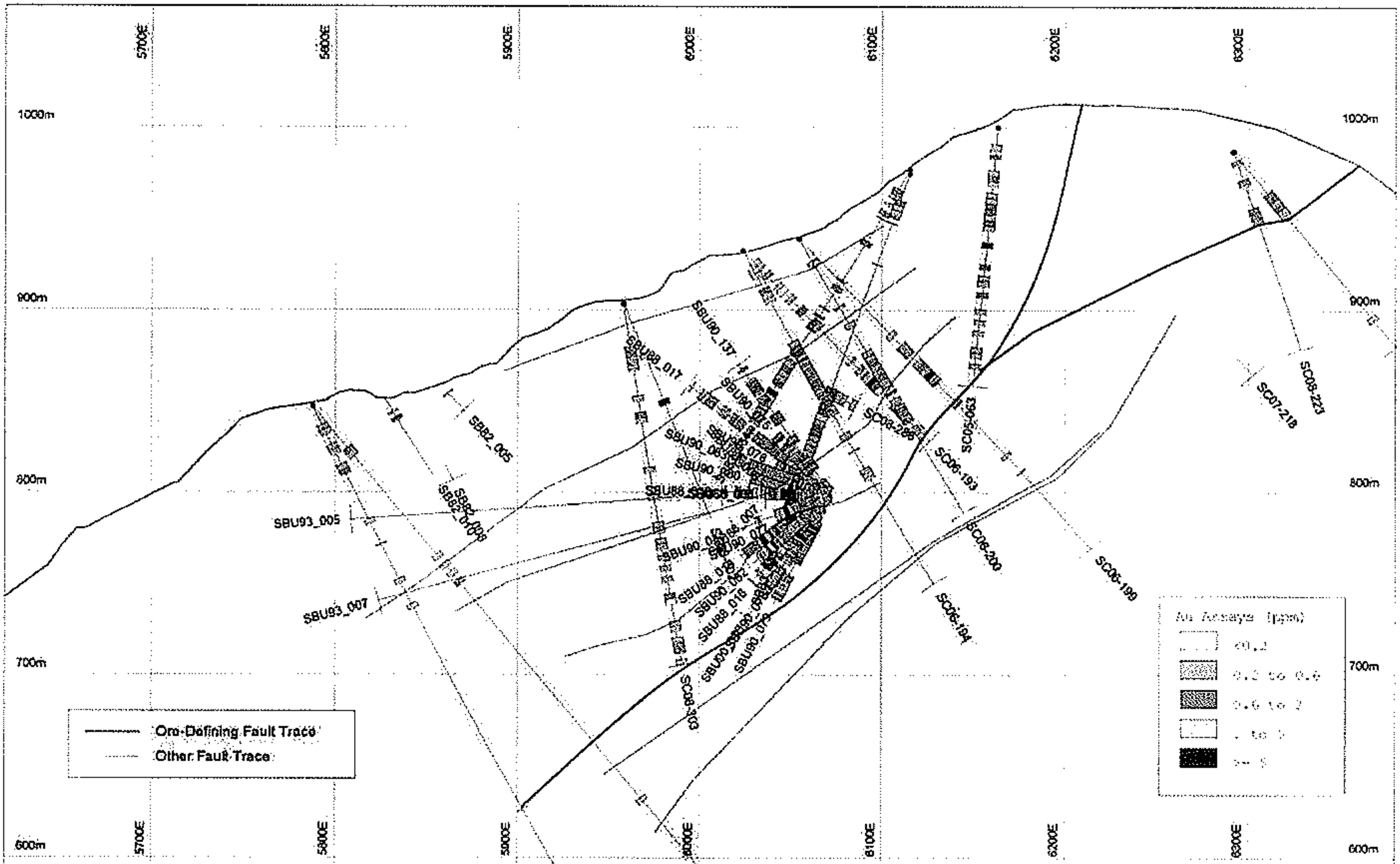


Issued by:	Drawing Prepared for:	File Name:
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig8-1.cdr
	Project: Silver Coin Gold Project	Project Number: 114-311007
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 8-1
Shape of Gold Mineralization above the Internal Cutoff (g Au >=.33) in relation to the Updated Cone Pit



Issued by:	Drawing Provided by/Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig8-2.cdr	**Figure 8-2** **Mazur Section with Interpreted Faulting**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	



Pinnacle Mines Ltd Vertical E-W Section Looking North	Cross Section 7640N
SILVER COIN PROJECT	

Issued by: TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Drawing Prepared for/Provided by:	File Name:
Pinnacle Mines Ltd.	Fig8-3.cdr
Project: Silver Coin Gold Project	Project Number: 114-311007
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 8-3
Section Showing the Controlling Fault In Bold Red (Looking North)

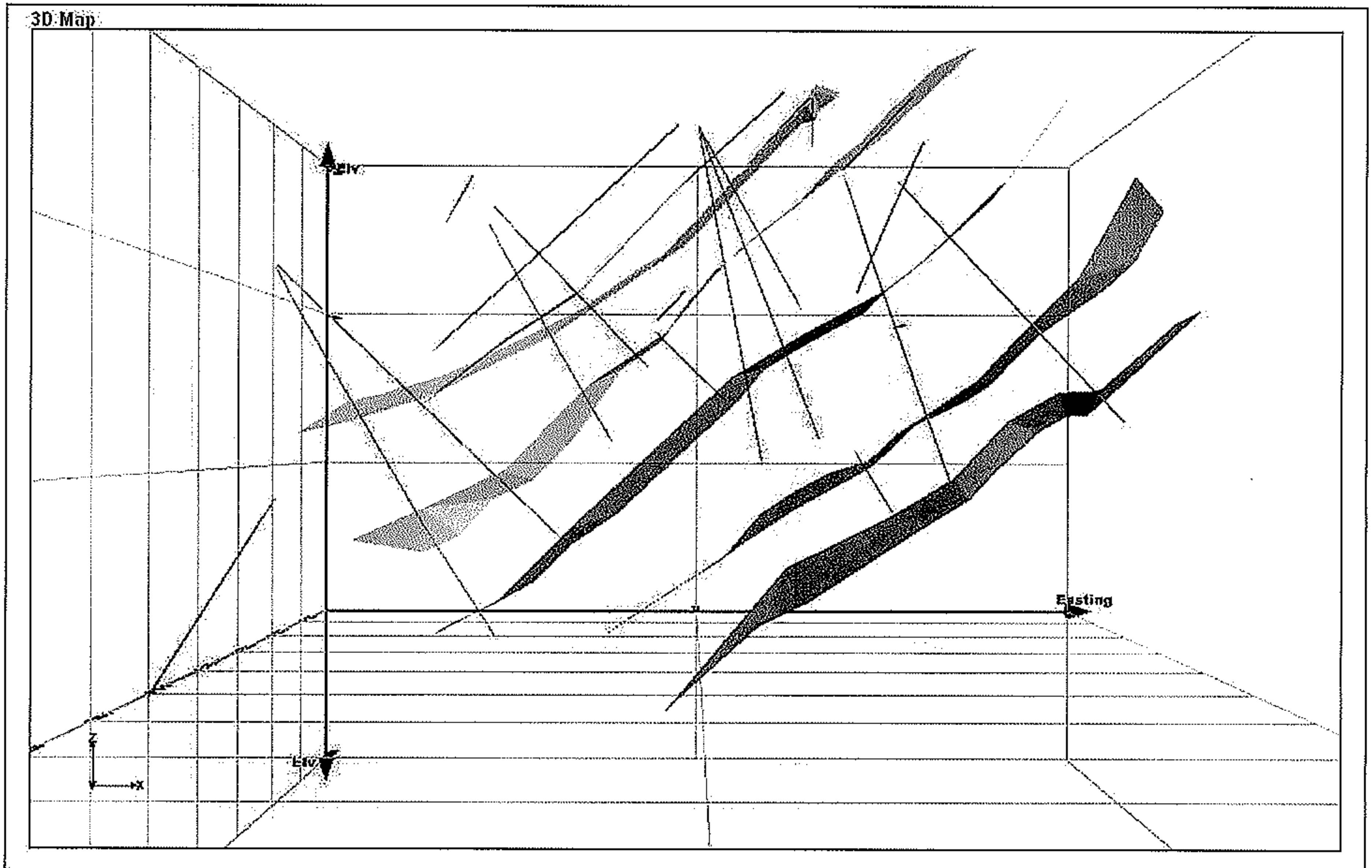
Several interesting insights are apparent from this effort. First, the eastern-basal fault noted above forms a predictable base to gold mineralization as illustrated on FIGURE 8-3 in bold red. Secondly, these faults seem to be parallel to each other in the dip direction rather than behaving in a listric fashion and converging with depth (FIGURE 8-4). Third, the overlying subsidiary faults appear to terminate against the controlling fault going north

FIGURE 8-5 illustrates the relationship between the existing drillholes and the controlling fault that defines the bottom of the mineralization at the Silver Coin deposit. This controlling fault forms the major defining feature in the three-dimensional geologic model and resource estimate that are detailed in SECTION 17.0 of this report.

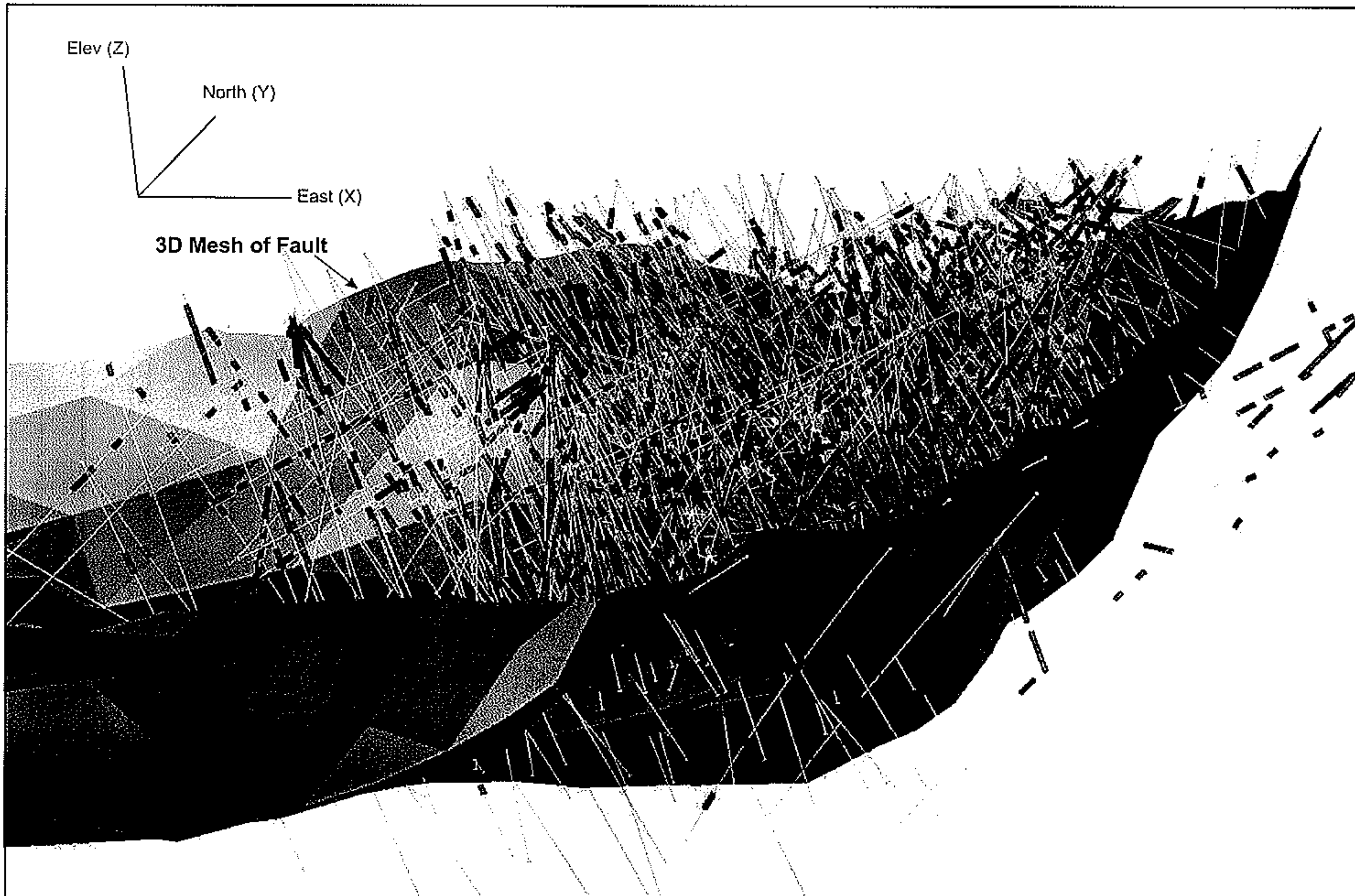
Digital Geologic Model

Pinnacle has worked to develop both lithological and structural digital models of the deposit to enhance the accuracy of the resource estimate. The structural model is dominated by a curve-shaped surface that the Company believes is a reactivated fault. Originally, the fault was most likely a reverse fault, formed in a compressional environment. Later, during uplift and intrusive-caused doming, the fault was reactivated with normal displacement. Whatever the case, the fault is believed to have been a conduit for mineralization. FIGURE 8-6 illustrates this fault surface and the lack of significant mineralization occurring immediately below it.

Synthesizing a digital lithologic model is challenging; lithology described by the numerous different geologists in separate drill campaigns over the exploration history of the deposit is simply not consistent. At Silver Coin the subtly different lithologies have all been strongly altered and visual logging is simply not adequate to distinguish the original lithologies of the package of andesitic volcanic rocks that hosts the mineralization. Pinnacle has postponed further effort on the lithology model until geologists can study the altered rocks at surface and bring this understanding to the job of re-logging some or most of the drill core.



Issued by:	Drawing Prepared for/Provided by:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig8-4.cdr	**Figure 8-4** **Interpretation of Silver Cone Faulting** **In Section 72400N (Looking North)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	



Issued by:	Drawing Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Pinnacle Mines Ltd.***	Fig8-5.cdr	**Figure 8-5**
	Project: Silver Coin Gold Project	Project Number: 114-311007	**Relationship of the Silver Coin**
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	**Major Controlling Fault and the Drillholes**

9.0 MINERALIZATION

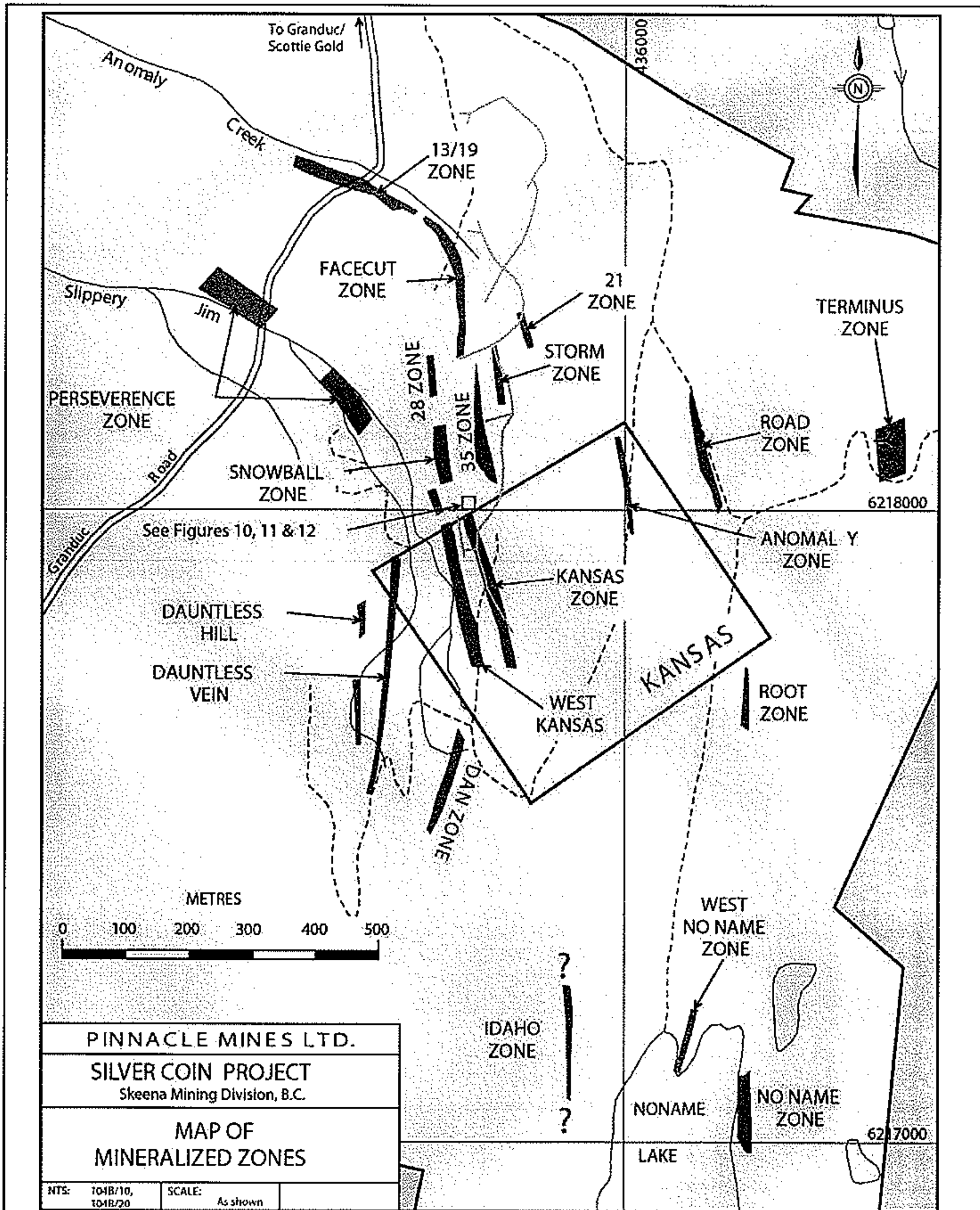
Silver Coin mineralization has been characterized by Alldrick (1993) and others as epithermal gold-silver and sulfide-bearing breccias and veins. The mineralization is quite similar to that mined at the nearby Silbak Premier Mine, where 2 million ounces of gold and 43 million ounces of silver were produced along with 55 million pounds of lead, 18 million pounds of zinc, and 4 million pounds of copper (BC EMPR Production Statistics). The Silbak Premier mineral system was enriched in base metals, dominated by lead and zinc. Both deposits have implied relationships to sub-volcanic and/or magmatic sources of heat and fluids. The global silver to gold production from Silbak Premier was approximately 20 to 1. Accounting for differential recoveries of the metals is difficult, but given that all modern production was optimized for gold recovery, production figures probably bias the silver to gold ratio low. Silver Coin saw no significant historic production, but the current overall mineral resource has a silver to gold ratio of about 5 to 1. Even the high-grade sulfide mineralization mined at the Facecut-35 Zone in 1991 had only a 6-to-1 silver to gold ratio. Hence, it appears from a cursory review, that Silver Coin is a relatively higher gold and lower base metal system than Silbak Premier. One possible implication of this is that the Silver Coin mineralization occurs higher in the system than the Silbak Premier deposits.

In the Stewart area, gold is spatially and temporally associated with early Jurassic quartz-rich alkaline to calc-alkaline intrusions and volcanic centers. Alldrick (1993) and others have described two main styles of mineralization in the district: high sulfide (>20% sulfide) base metal-rich silver-gold and low sulfide (<5% sulfide) silver-gold mineralization. Alldrick's study suggests, and petrography at Silver Coin reportedly confirms, that the lower sulfide mineralization is earlier than the higher sulfide type. The style of mineralization and geochemical fingerprint observed today may reflect either or both geologic time overlap and/or physical zonation.

Mineralization across the Silver Coin property is contained within the 20 different zones identified on FIGURE 9-1. Stone and Godden (2007) have defined ten types of mineralization. Gold is typically associated with silicification and locally with base metal mineralization. Sulfides include pyrite, sphalerite, galena, chalcopyrite, and rarely tetrahedrite. In the area of the Indi 9 claim, the Extreme copper showing is located in brecciated granodiorite. Detailed descriptions of the mineralized zones, types of mineralization, types of alteration, structural controls, and textural features are found in the appendices and references cited at the end of this report. Studies by Stone and Godden (2007), Kruchkowski (2007), and Walus (2009) are particularly relevant in this regard. The bulk of the gold mineralization present on the Silver Coin property is of low sulfidation epithermal character. This category is strongly indicated by the presence of electrum, crude banding of some sulfide veins, presence of chalcedonic quartz, and very widespread silicification. The detailed characteristics of this type of mineralization are described in a document titled "Appendix E Deposit Type/Mineral Deposit Profiles", available on the website of the British Columbia Ministry of Energy, Mines and Petroleum Resources and referenced below.

The most significant mineralization is the Main Breccia Zone that has been traced over a strike length of 2.5 kilometers, a vertical distance of 700 meters and widths varying from 20 to 100 meters. Mineralization is structurally controlled, generally with strong potassic and phyllic wall rock alteration. Secondary enrichment is not a significant component.

[1]http://www.empr.gov.bc.ca/Mining/Geoscience/MineralDepositProfiles/Pages/default.aspx



Issued by:	Drawing Provided by/Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Pinnacle Mines Ltd.***	Fig9-1.cdr	**Figure 9-1** **Mineralized Zones on the Silver Coin Gold Property**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

The mineralized zone consists of fractured andesite tuff with quartz veinlets containing sphalerite, galena, pyrite, locally chalcopyrite and sporadically distributed fine native gold, and silicified tuff and intensely brecciated and silicified stockwork zones. The Main Breccia zone is defined by gold values greater than 0.2 g/t compared to a background value of less than 0.1 g/t Au (Stone and Godden, 2007).

10.0 EXPLORATION

Exploration-related activities on the Silver Coin Gold Project site have been accomplished by a number of industry standard techniques. Some of these include: surface and underground core drilling, surface trenching, underground adits, surface geochemistry, and surface geophysics. The following discussion briefly presents some of these studies.

Geochemistry

The Silver Coin Project has not benefitted from rigorous surface or drill core geochemical programs. The drilling database includes analyses for Au, Ag, Cu, Pb, and Zn; approximately two thirds of the assays in the drill hole database have been assayed for base metals (23,000 out of 34,000). Multi-element data exist for two batches of 95 and 34 samples taken from drill pulps. The soil geochemistry is incomplete and only available on paper maps.

Litho geochemistry on Drill Samples

There is a weak general correlation between gold and silver; that is to say, that there are many samples that are elevated in both gold and silver (FIGURE 10-1). However, there are also distinct populations in which gold and silver vary independently. The very high silver values in the drill data seem to indicate a population that is significantly depleted in gold relative to silver: there is a large population of extreme gold values without corresponding silver enrichment.

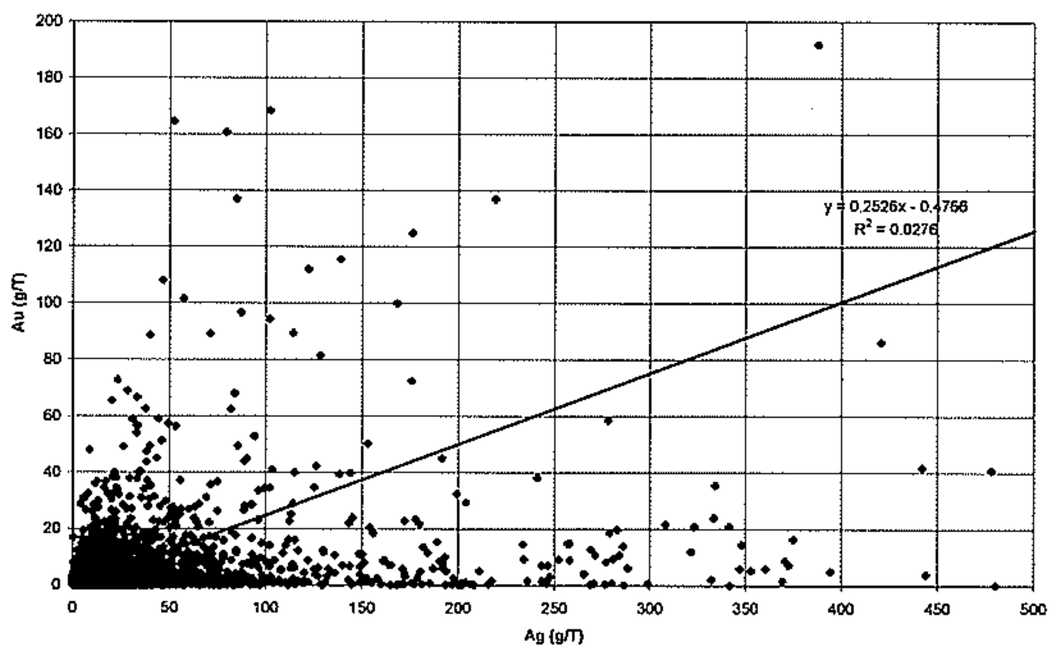


FIGURE 10-1: AG VS. AU IN SILVER COIN DRILL DATA (N = ~28,000).

Two small batches of samples were submitted to International Plasma Labs (IPL) for multi-element ICP-ES determinations after multi-acid digestion. There appear to be quality control issues with some components of these data that is being discussed below. These analyses included total sulfur determinations for the opportunity to compare gold and silver with sulfur content. Silver is moderately to strongly correlated with sulfur, while gold demonstrates only a weak correlation with sulfur. This is best demonstrated in the figures below (Figure 10-2 10-2 and FIGURE 10-3) from the small batch of samples analyzed by IPL during summer of 2008.

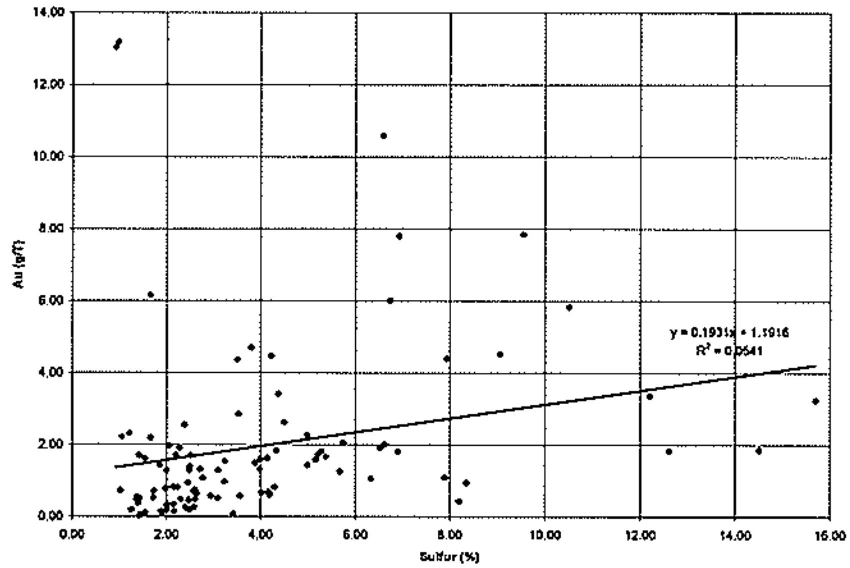


FIGURE 10-2: S VS AU IN SILVER COIN DRILL DATA.

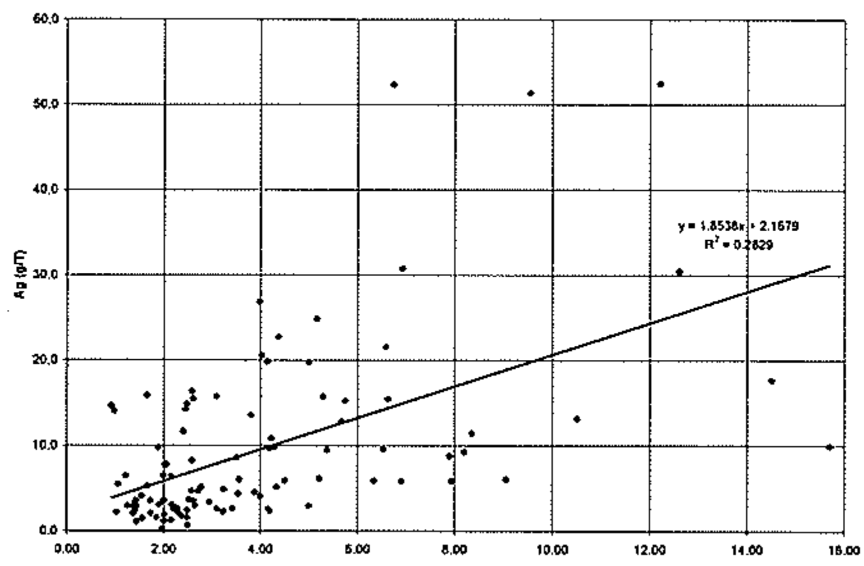


FIGURE10-3: S VS. AG IN SILVER COIN DRILL DATA.

Gold does not correlate strongly with zinc and copper. While there are many samples that exhibit high gold and high base metals, there are also strong populations that reflect either gold or base metal dominant mineralization. Silver Coin lies in a region of high gold content, and the base metal distribution may provide evidence of intensity or zonation within the system.

The chart of 23,000 samples that have zinc and gold in common (FIGURE 10-4) seems to confirm that Silver Coin is a zinc-gold deposit. A large percentage of high grade gold values (> 5 g/t) are correspondingly enriched in zinc. However, it is also true that few of the extremely high grade gold samples (> 20 g/t) contain very high grade zinc (> 10%). Hence, there are separate populations of very high grade gold and very high grade zinc (the groups of samples that follow the X and Y axes). It is not surprising to see gold well represented in high base metal samples because the sulfide deposits tend to be enriched in gold as well; but the highest gold is likely not be related to the highest sulfide samples.

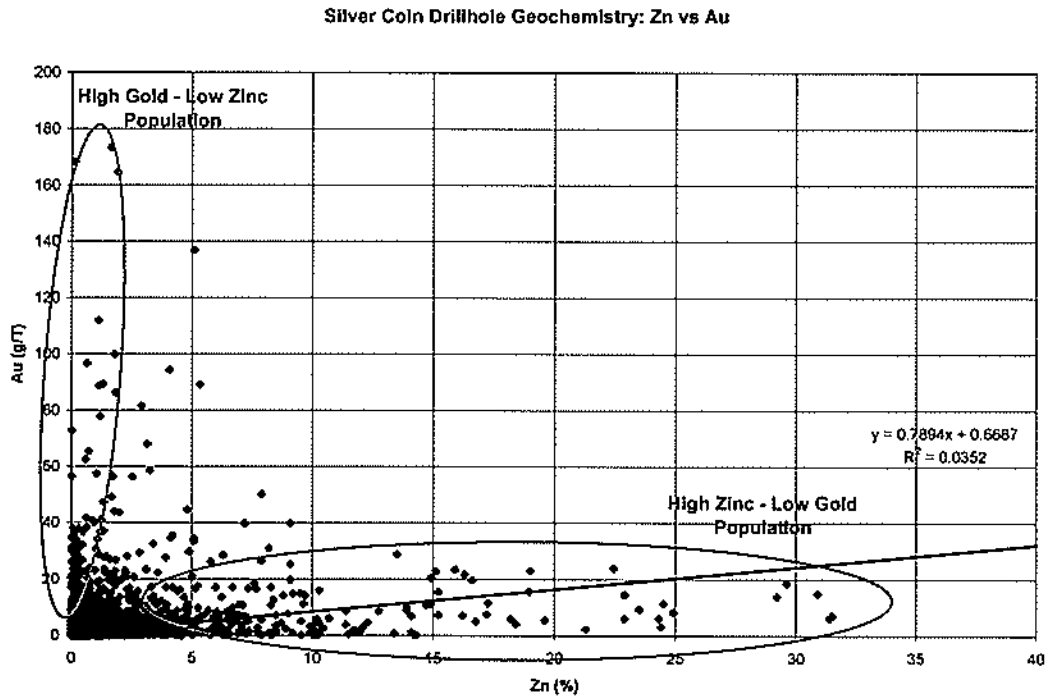


FIGURE 10-4: ZN VS AU IN SILVER COIN DRILL DATA (N = ~23,000)

Relationships such as these serve to highlight the considerable variation within the deposit.

A similar relationship exists within the copper data (FIGURE 10-5). Copper is generally less enriched with gold than zinc at Silver Coin. There is also a population that is high in gold and relatively low in copper. An inverse relationship also exists: that is, high in copper but relatively poor in gold; all of the very high copper values (> 3%) are low in gold.

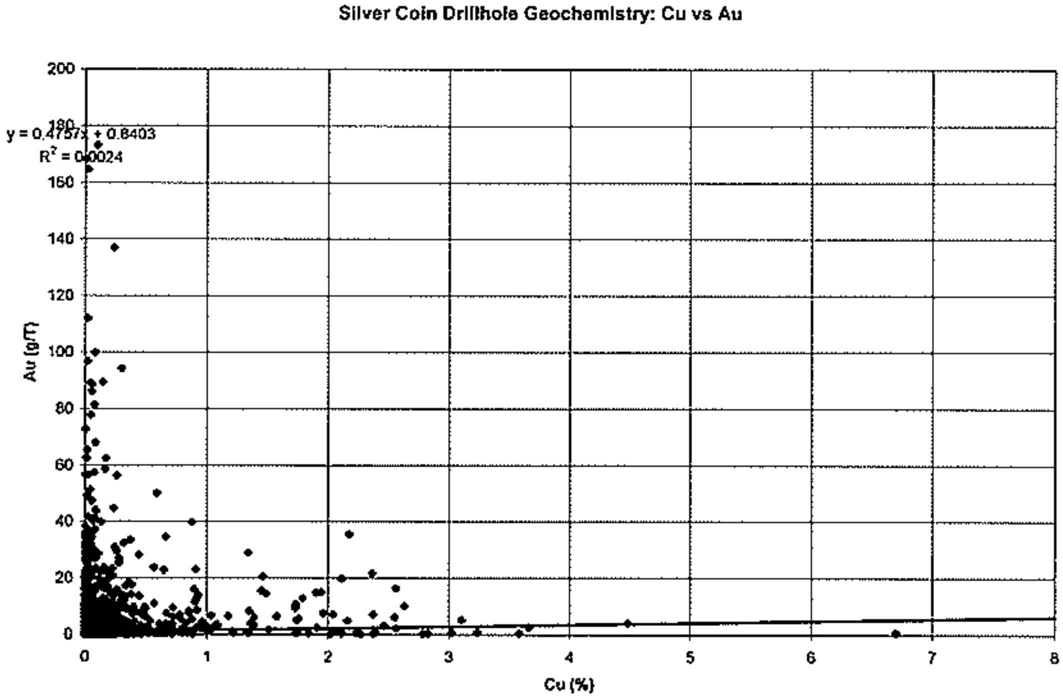


FIGURE10-5: CU VS AU IN SILVER COIN DRILL DATA (N = ~23,000)

The first batch of multi-element analyses was reported in August 2008 on IPL certificate# 08H3745. This batch of 95 samples ranged from 0.04 to 13.19 g Au/t, so it disproportionately represents the gold-rich parts of the deposit.

The samples were analyzed by a multi-acid digestion (including hydrofluoric acid) and multi-element scan by ICP-AES. This technique breaks down the major rock forming minerals so as to constitute a near-total digestion for many elements. It is more aggressive than the traditional aqua regia digestion that is often used to focus on trace elements bound in sulfide minerals or to save money. The disadvantage of the multi-acid digest is that it can volatilize some key elements. It is usually not possible to recover quantitative mercury by this method. Arsenic and antimony can be accurately reported if special care is applied.

In the case of certificate# 08H3745, the arsenic and antimony data are not meaningful; virtually all of these samples report below detection limits and are suspected to be inaccurate.

The data from Spring of 2009 on IPL certificate 09C0861 are reported from the same analytical package and they yield apparently useful results for As and Sb. The average values of As and Sb for these 34 samples are 44 and 12 ppm, respectively. The lab also provided single element assay methods for As and Sb, but these are reported in percentages with a reporting limit that is much too high for useful geochemistry. Nevertheless, these limited data indicate that Silver Coin is enriched in As, Sb, Hg, Bi, and Cd in addition to Ag and base metals as discussed above. This suite of anomalous elements affirms the perception of the Silver Coin deposit as an epithermal gold-silver deposit.

The presence of bismuth up to 12 ppm may be significant because that constitutes two orders of magnitude enrichment above background levels. The 2 ppm reporting limit for the method is insufficient to interpret the data with confidence, but if this holds up with additional analyses, such bismuth concentrations are likely indications of a magmatic influence on the rocks. This means that higher temperature fluids, such as those associated with a porphyry or an intrusion related system, influenced the Silver Coin mineralization.

Among this suite of epithermal elements, only Hg and Sb show a weak positive correlation with gold. Zinc, on the other hand, is strongly correlated with mercury (FIGURE 10-6). This relationship is noteworthy because it is not commonly observed in epithermal gold deposits. Normally the mercury, having high volatility, moves through the system to lower temperatures, such that it can be driven off entirely from the deeper parts of the system. Zinc and base metals normally increase with depth as temperatures warm approaching the source of the hydrothermal fluids: the highest zinc should not correspond with the highest mercury.

Epithermal gold systems can be zoned over more than a vertical Km, in which case gold and mercury would be well separated from the high copper, lead, and zinc. The systems may also be compressed over a short vertical range. In that case, the same geochemistry and affinities are at work, but the various zones of the system will clearly overlap. Eskay Creek is reported to be such a system – having many of the components of a high sulfidation epithermal gold system but also exhibiting an intimate association of gold and silver with massive sulfide deposits. Geologists from the BC Geological Survey reported that the zonation appeared collapsed at Eskay because the massive sulfide was venting on the sea floor in close proximity to a causal magma. There is some evidence for such a collapsed alteration system at Silver Coin, evidenced in part by the juxtaposition of low and high temperature components of the system.

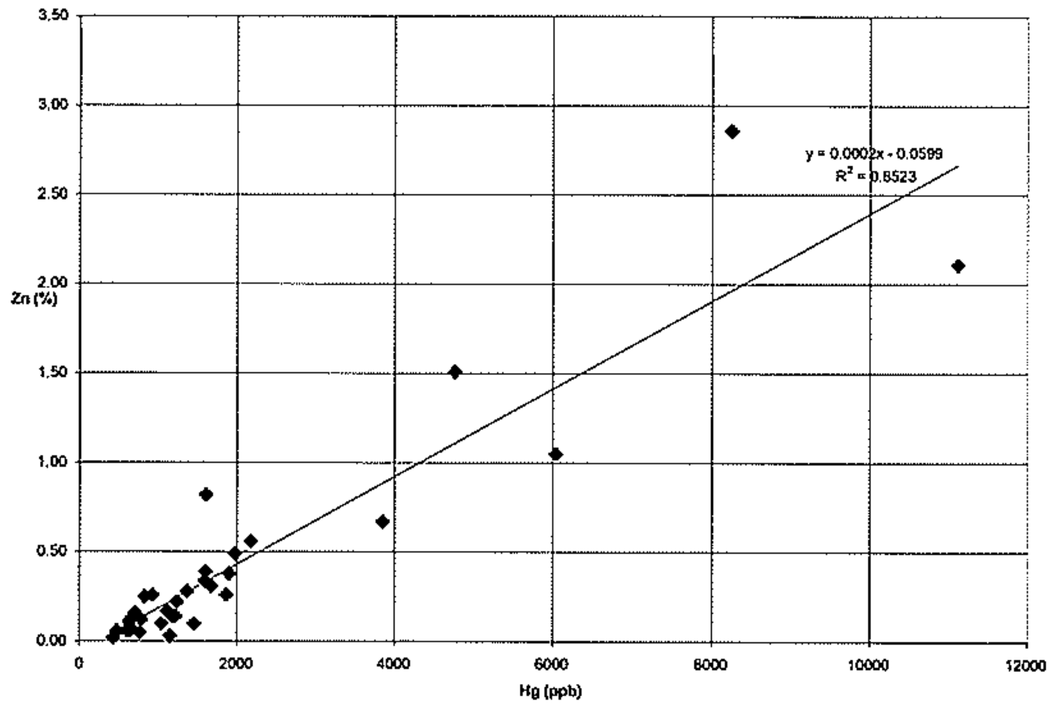


FIGURE 10-6: HG VS ZN IN SILVER COIN DRILL DATA

The elements having the strongest correlation with gold are titanium and magnesium. These elements have a moderate to strong negative correlation with gold concentrations (FIGURE 10-7). This indicates that rocks high in magnesium or titanium (along with vanadium) are unlikely to be highly enriched in gold. This is useful because it suggests that mafic volcanic rocks (those high in iron, magnesium, vanadium, titanium, and certain other elements) are less likely to host ore-grade gold than felsic volcanic rocks. It is quite difficult to distinguish the original composition of the host rocks at Silver Coin. Most of the volcanic rocks of the area are broadly described as andesitic (or intermediate) in composition. But if this preliminary conclusion holds up after further testing, it may be possible to focus on the subtle variations in volcanic composition as indications of favorable host rocks.

An expanded litho-geochemical study is needed, coordinated with petrographic studies, to determine if these observations are valid.

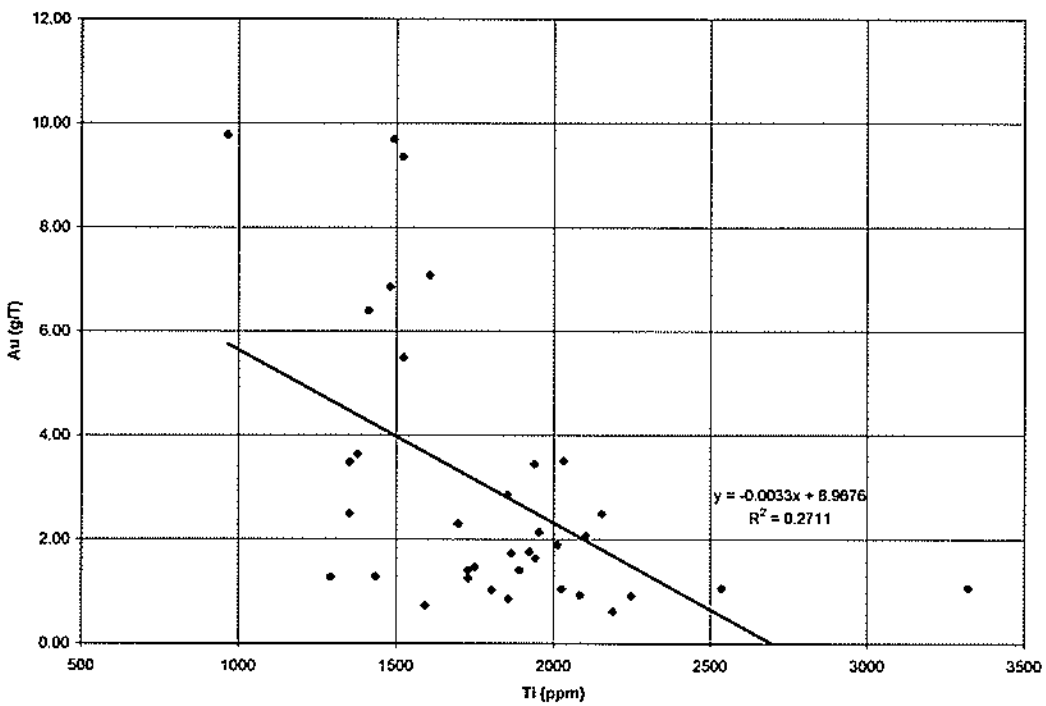


FIGURE 10-7: TI VS AU IN SILVER COIN DRILL DATA

Some of the observations reported herein suggest two mineralizing events:

- A base metal-rich higher temperature system that is enriched in silver but much less so in gold
- A gold-dominant epithermal system of lower temperature, perhaps demonstrating its own vertical gradients in chemistry to base metal rich roots

Massive sulfide deposits of various origins tend to have a signature consistent with the first type. Syngenetic exhalative sulfides often exhibit a close association between very high silver and lead-zinc sulfides. This relationship is also common in sulfide deposits associated with zoned porphyry or intrusive related systems. This duality of mineralizing events may be significant at

Silver Coin because the district boasts several styles of important deposits, including the world class porphyry deposit at Kerr Sulphurets, the hybrid gold-VMS deposits at Eskay Creek, and the Granduc VMS deposit.

Different styles of mineralization may develop at different times but physically overlap: there is evidence in the district of early Jurassic VMS style mineralization and middle Jurassic porphyry related mineralization that may have overlapped. It is possible that a zinc-dominant VMS system formed at Silver Coin in early Jurassic time, only to be crosscut and over-printed by epithermal veins and breccias, possibly related to a magmatic-driven system at depth. The close association of mercury and zinc could be explained in this way. Pre-existing massive sulfide mineralization may have provided a de-sulfidation mechanism to precipitate metals from later epithermal fluids.

Soil Geochemistry

Despite at least five episodes of soil geochemical surveys, the property has still not been systematically covered. Esso conducted the most complete soil surveys near the north side of the current land package, collecting approximately 1270 samples between 1980 and 1982. Recent surveys by Pinnacle and Mountain Boy Minerals were reported to be small and focused on the south side of the property.

Although the original Esso report references 11 maps, only five were available, including contoured maps for Au, Ag, Cu, Pb and Zn anomalies. Unfortunately, the original assays were not mapped or reported. According to the report, the soil was sampled on a grid basis, generally at a 50m by 25m spacing.

Soil sampling methodology included taking samples from the B horizon at an approximate depth of 20 to 30 cm using a mattock. The samples were stored in paper bags, dried, and then sent for analysis. Samples taken prior to 1989 were analyzed for Ag, Cu, Pb and Zn with only some being tested for Au. In 1989, all of the samples were tested using a 31 element I.C.P. method.

These five maps were geo-referenced and uploaded into the GIS database (FIGURES 10-8 and 10-9). The original data from those surveys has not been located, but they reportedly included values up to 2600ppb Au, 27ppm Ag, 4300ppm Pb, and 2400ppm Zn. The high gold in soils is particularly poignant when superimposed on the proposed pit outline resulting from years of drilling by Pinnacle and its predecessors. As the drilling from 2007 and 2008 showed, the mineralization on the north end of the deposit comes to the surface and that is apparent in this 27-year-old soil anomaly.

The most significant aspect of this incomplete database may be the large zinc anomaly to the northwest, offset relative to the highest gold values. This could be a zinc halo outside the main gold zone, or it could represent continuation of the mineralized system to the northwest. Just as in the litho-geochemistry, the gold and zinc populations are clearly independent in the soil data as well.

The Ag soil geochemistry map outlines several zones of interest with the most pronounced being centered around the surface exposure of the 35 Zone. Below Granduc Road, several anomalous zones occur in areas of mineralized float. To the northeast there are several sporadic anomalous zones which appear to be related to weak zones of mineralization observed in outcrop.

Issued by:	Drawing Prepared for/Provided by:	File Name:
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig10-8.cdr
	Project: Silver Coin Gold Project	Project Number: 114-311007
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 10-8
Contoured Gold (PPB) in Soils at the Silver Coin Gold Property

Issued by:	Drawing Prepared for/Provided by:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig10-9.cdr	Figure 10-9 Contoured Zinc (PPB) in Soils at the Silver Coin Gold Property
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

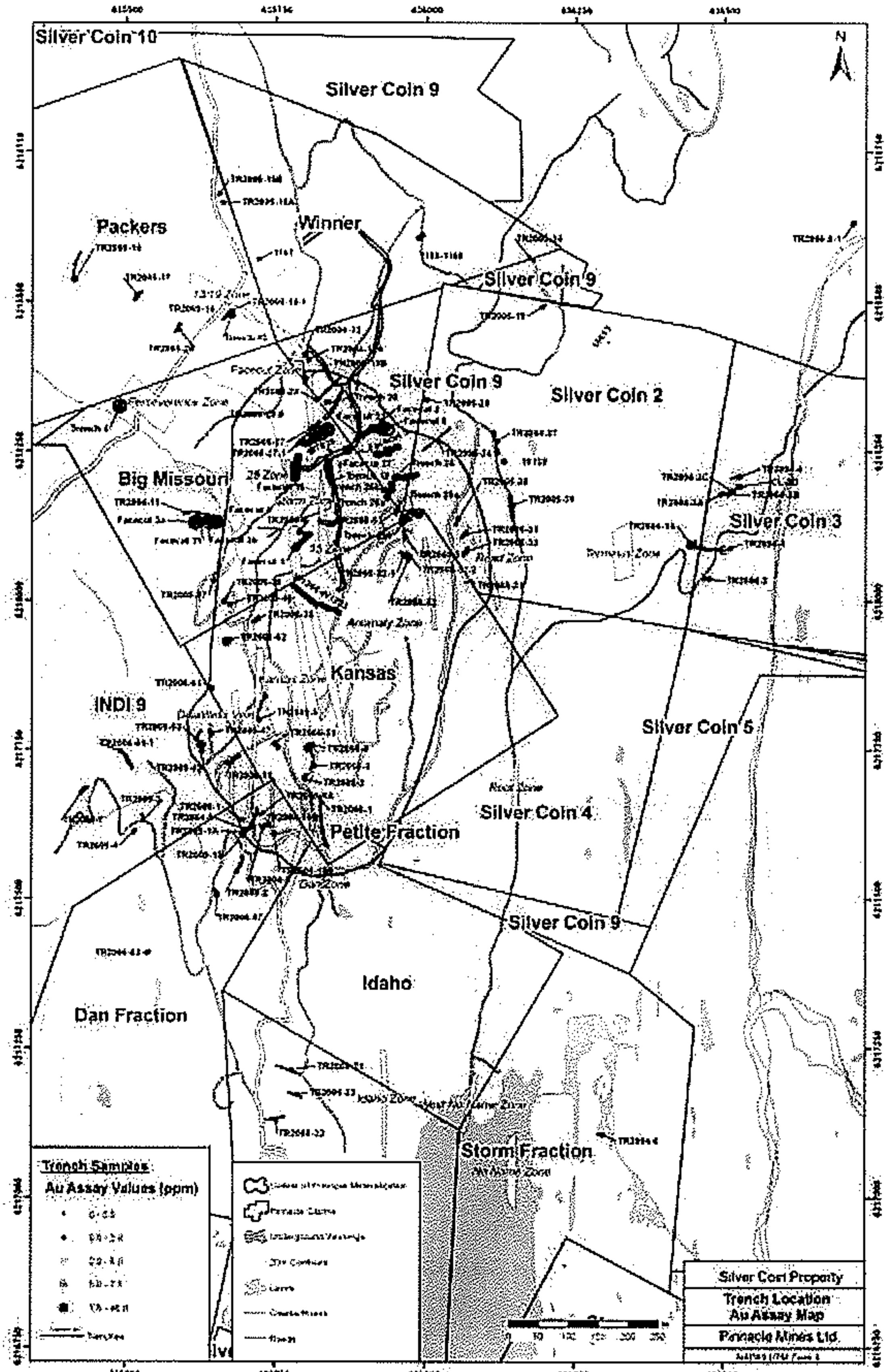
It is also significant that isolated gold anomalies occur 500m north-northeast of the pit outline. Since neither the supporting geochemical data nor the field notes are available, it is difficult to interpret such isolated anomalies. Given the steep slopes on the north end of the property, those anomalies could be affected by colluvium. The anomalies themselves may be transported, or they could be diluted or partially obscured by creeping soils. At least one of those anomalies appears to be untested by drilling. Even though some of the mapping suggests a westerly bend in the mineral system, there is a strong prevailing structural corridor continuing to the north. That combined with the presence of significant mineralization at the Big Missouri Mine to the north should warrant careful consideration of a possible northerly continuation of the Silver Coin system.

The northern end of the Silver Coin deposit exhibits very strong near surface mineralization. As the topography descends to the north, geochemistry is an obvious tool to lead the targeting outside the existing drill pattern. Based on the success from the northern holes in 2008, step out drilling will be simple enough initially. However, there are valid targets to the northwest and to the north. These geochemical data and the structural model for this mineralization suggest that a northern continuation is possible. The erosional exposure to the north makes it particularly amenable to geochemistry, utilizing both reconnaissance scale methods such as stream sediments and targeting-scale tools such as soils.

The structural corridor that hosts Silbak Premier, Silver Coin, and Big Missouri continues northward towards the giant Kerr Sulphurets deposits. It is quite possible that district scale zonation will become apparent when the geochemistry is viewed through a large enough lens. The combined gold resources at the Kerr-Sulphurets-Mitchell and Snowfield projects now exceed 60 million ounces.

Trench and Rock Chip Sampling

Old maps available indicate the rock chip sample locations, Cu, Pb and Zn assay results for the rock chip samples, and trench locations with Au results. A composite figure of these maps is shown below (FIGURE 10-10).



Issued by:

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Drawing Provided by/Prepared for:	File Name:
Pinnacle Mines Ltd.	Fig10-10.cdr
Project: Silver Coin Gold Project	Project Number: 114-311007
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 10-10
Historical Trenching with Gold Values

Trench W7026 had the best Au assay results with 6 samples greater than 10 ppm. Facecut 3a and Facecut 6 also had samples that were greater than 10 ppm.

Geophysics

Several I.P. (induced polarization) surveys were completed in 1979 by Consolidated Silver Butte Minerals and in 1982 and 1983 by Esso Minerals Canada. Esso's geophysicists reportedly determined that the surveys were "inconclusive" and the survey was abandoned at an early stage. The geophysical data has not yet been digitized and integrated into the digital database.

The Silver Coin property is lacking in systematic geophysical work. There are areas of high sulfide content in the Silver Coin mineralization, and other deposits in the district are also associated with high sulfide content. Electrical geophysical surveys are recommended. Given the importance of structural controls on the gold mineralization in this district, it would also be prudent to consider magnetic and gravity surveys as part of a district strategy

11.0 DRILLING

This section summarizes the drilling that Pinnacle-MBM has completed on the property in the time frame 2004-2008. No drilling occurred in 2009. FIGURE 11-1 details the location of the Pinnacle drilling from this period.

2004 Drilling

From June 10 to October 19, 2004 a total 3,133m of BTW core in 38 holes was drilled using a J.K Smit 300 drill owned by Mountain Boy Minerals. These holes tested the West No Name Lake, Terminus, Kansas/West Kansas and part of Perseverance Zone called the BM Zone. The best results came from holes 2004-29 to 38 drilled on the Kansas/West Kansas Zone. Hole SC04-34 yielded an average of 11.29 g Au/t, 33.65 g Ag/t 0.066% Cu, 1.11% Pb, and 4.65% Zn over 21.33m and hole SC04-37 returned 5.15 g Au/t, 41.99 g Ag/t, 0.12% Cu, 1.73% Pb, and 2.58% Zn over 24.4m.

2005 Drilling

In 2005, a total of 8,041.61m of NQ, BQ and AQ size core was drilled in 67 separate holes. The bulk of the drilling was done on the Main Breccia Zone with minor drilling testing the area of DDH-87-16, the 21 Zone, and a granodiorite dike with a quartz stockwork carrying stringers of sulfides located on the INDI 12 claim. The best drill results in 2005 included 44.4m of 5.95 g Au/t, 24.87 g Ag/t, 0.06 % Cu, 0.29 % Pb and 2.11 % Zn in DDH SC05-44, 118.65m of 5.39 g Au/t, 32.76 g/ Ag/t, 0.014 % Cu, 0.16 % Pb and 0.34 % Zn in DDH 2005-52 and 9.15m of 47.37 g Au/t, 69.56 g Ag/t, 0.08 % Cu, 1.41 % Pb and 1.74 % Zn in DDH SC05-65.

2006 Drilling

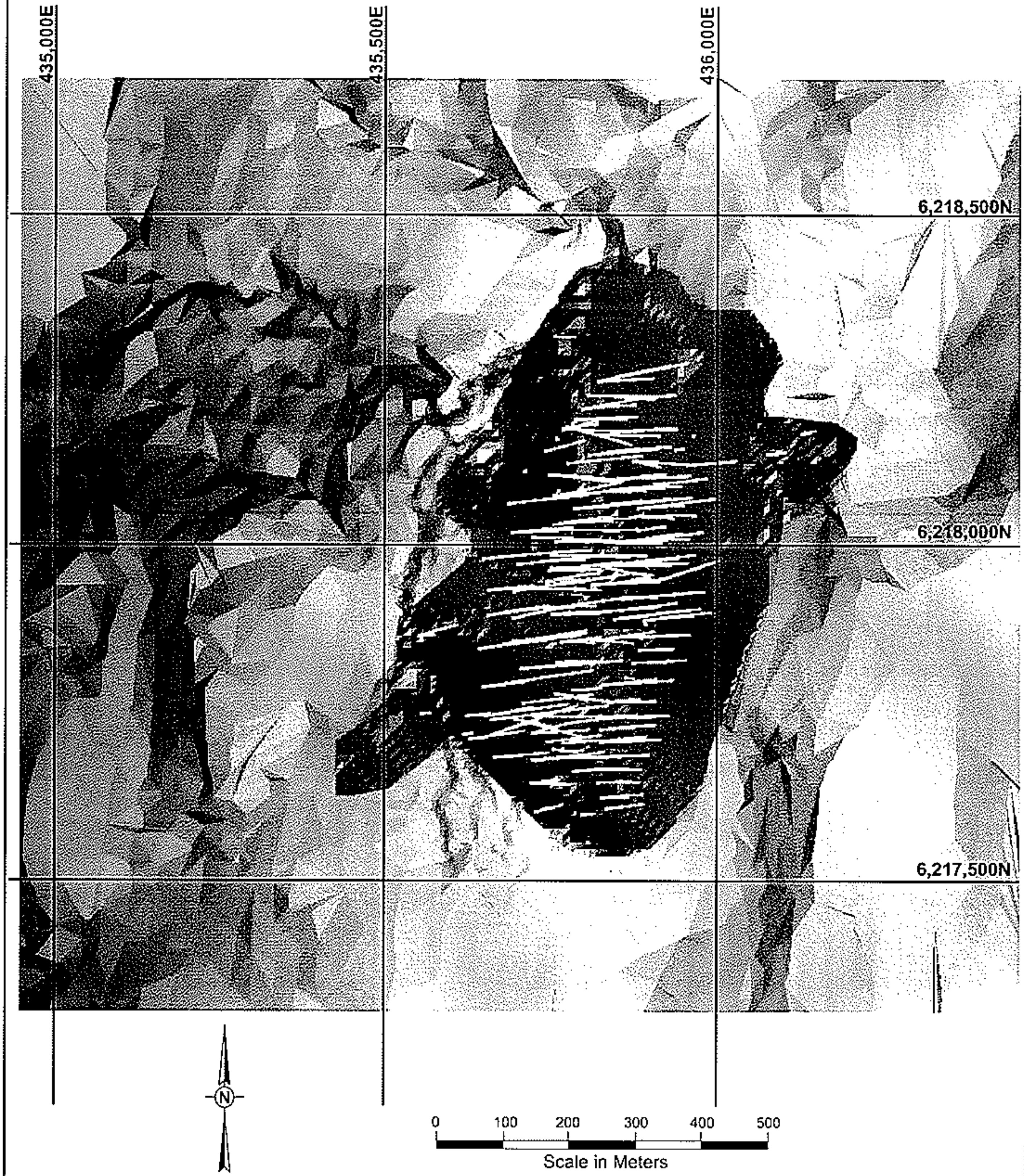
In 2006 Pinnacle drilled 24,151m of NQ and BQ core in 115 holes. The majority of holes were drilled on the Main Breccia Zone. Eight deep (300-750m) reconnaissance holes (holes 2006-150 through157) were drilled below the Main Breccia Zone. The best 2006 results include 2.64 g Au/t and 424 g Ag/t over 9.14m in hole SC06-124; 3.64 g Au/t over 45.73m in hole SC06-130; and 9.05 g Au/t over 18.29m in hole SC06-178.

2007 Drilling

The short 2007 drilling program, totaling 2,764m of BQ core in 16 holes, was conducted exclusively on the Big Missouri claim. The program was designed partly as infill drilling and partly to extend the known mineralization of the Kansas/West Kansas Zone to the west and east. The most significant results came from hole SC07-212 which returned 1.74 g Au/t over 66.75m.

2008 Drilling

In 2008 Pinnacle drilled 88 holes totaling 12,216m of NQ and BQ size core. Seventy-eight of the holes were drilled on the Main Breccia Zone. Five holes tested the Road Zone and another five the Northeast Zone. The 2008 program was very successful. Several holes extended the known mineralization of the Kansas/West Kansas Zone for another 80m to the south. The best results from these holes include 12.66 g Au/t, 24.46 g Ag/t, 0.3% Pb, and 0.77% Zn over 12.19m in hole SC08-232; 6.26 g Au/t, 11.27 g Ag/t, 0.20% Pb, 0.50% Zn over 42.37m in hole SC08-233; and 5.40 g Au/t, 12.0 g Ag/t, 0.32% Pb, 0.60% Zn over 27.43m in hole SC08-264. Several other holes encountered new high grade gold and base metals mineralization believed to be an extension of 21 Zone (discovered in 1982) to the north. The best results from these holes included 3.74 g Au/t, 21.40 g Ag/t, 0.67% Pb, 0.99% Zn over 64m in hole SC08-298 and 4.45 g Au/t, 29.3 g Ag/t, 0.2% Pb, and 0.48% Zn over 54.87m in hole SC08-306. Other holes, planned as infill drilling on the Main Breccia Zone also



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig11-1.cdr	Figure 11-1 Pinnacle Drillhole Locations
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

yielded many good results. The best assays came from hole SC08-282 which yielded 8.86 g Au/t, 70.81 g Ag/t, 0.14% Pb, and 0.35% Zn over 45.72m. Ten holes, which tested the Road and the Northeast Zones returned weak results.

12.0 SAMPLING METHOD AND APPROACH

12.1 Sample Method and Details

Extensive surface sampling has been done by numerous operators on the Silver Coin property. Prior to 1980 little is known about the sampling method. From 1980-1994, recognized companies such as Esso, Tenajon and Westmin worked on the property and while only limited detail is available about their work, some evidence in the form of standard field notes and maps, lends support to the assumption that the work was done to industry standards. Starting in 2004 all work was done by Mountain Boy Minerals and Pinnacle. The two companies have collected rock-chip, channel, trench and soil samples. Much of the sampling was done or supervised by Alex Walus, the Pinnacle project geologist during that period. Walus (2009) says the following;

"Soil and rock sampling conducted on the property by Pinnacle Mines and Mountain Boy Minerals was done according to standard, proven methods. Soil samples were collected from the B horizon and placed in Kraft paper bag. Samples were collected every 25 meters, distance between the soil lines were either 25 or 50 metres. Rock samples from trenches were collected using a rock hammer and chisel to obtain a continuous chip line across the strike of the mineralization. Sample intervals were dependent on intensity of mineralization and/or lithology. Most intervals were 2.0 meters in sample length. A large portion of the soil and rock samples from this period were collected A. Walus."

12.2 Core Drilling Sampling Method

The following discussion applies to drilling from 2004 through 2008. Sampling details for earlier drilling are not available.

Drill core was placed in wooden core boxes at the drill site and then each core box was labeled and securely closed for transport to the logging and core storage facility in Stewart. Core boxes were stored in Pinnacle's secure logging area in covered core racks in Stewart. When the geologist was ready the core was logged onto paper logs and then sawed in half for assaying. The boxes of sawed half-core were returned to the core rack for long term storage. Subsequently, the paper logs were transcribed into electronic spreadsheets where basic rock descriptions are recorded and data amenable to digital representation and plotting was entered, such as depth down the hole, rock codes and assays.

12.3 Data Collection

All drilling on the Silver Coin property has been diamond core drilling. At various times surface holes included BTW, NQ and small amount of AQ core. Underground drilling was exclusively AQ core.

Except for occasional narrow fault zones, core recovery from the great majority of holes drilled on the Main Breccia Zone was very good. Not all holes from earlier drilling contain recovery percentage data. Core recovery in several holes drilled to the north-west of the Facecut-35 Zone was very poor, and many holes were lost due to the bad ground in earlier Pinnacle-Mountain Boy drilling. From 1982 to 1987, the companies only sampled sections of the core with good visual mineralization. In 1989 Tenajon sampled and assayed more intervals from some of these earlier holes. From 1988 to 2008 geologists logged, sawed and sampled every hole from top to bottom.

Geologists logged the core onto paper logs according to standard industry practice. The logs were initially stored at Pinnacle's field office in Stewart and subsequent moved to Pinnacle's Vancouver office where they currently reside. After logging, all paper logs were entered into electronic spreadsheets for permanent storage and to facilitate computerized plotting of the data.

After the core was logged geologists marked sample intervals with sequentially numbered assay tags and the core was divided in half using either core splitters (some earlier drilling) or sawed (all post-2004 drilling). Half of the split core was sent to the lab for assaying and the other half was kept on site for future reference. Stone and Goddard, (2007) note the following:

"The positions of the markers are visually estimated, not measured. Greater accuracy may be obtained by utilizing a measuring tape to identify intervals, but this would greatly increase the time required to log each run. After each set of six core boxes has been logged, the geologist checks the first and last assay tags and that paper logs are correct, not least to avoid any discrepancies."

The core from 1993 and 1994 drilling was not split as the entire core was sent to the lab for assaying. From 1982 to 1991 the core was split using a core splitter and from 2004 to 2008 the core was cut with a diamond saw. The previous property operators (1982-1994) in most cases used one- and two-meter intervals to sample both mineralized and non-mineralized sections of the core. From 2004 to 2008 Pinnacle and Mountain Boy Minerals' geologists used 1.5 and 3.0 meters intervals to sample the core. Pinnacle is not aware of any factors that could materially affect the accuracy and reliability of the results. The rocks on the property are fresh with little or no secondary minerals on the surfaces that would enhance metal values.

Pinnacle geologist Alex Walus either personally sampled or supervised sampling of most of the holes drilled between 2004 and 2008. Walus (2009) states, "*the samples were representative and of high quality, collected according to standard industry practices.*"

12.4 Drilling, Sampling, and Recovery Factors

In the Main Breccia Zone the rock is generally competent and core recovery is excellent. As noted above, all holes drilled since 2004 were sawed and sampled from top to bottom although this was not uniformly the case for earlier drilling (pre-2004). Not all holes were logged for percent recovery (pre-2004).

12.5 Sample Quality

The quality of the core in the Main Breccia Zone, with the exception of faulted intervals is excellent. This allows the mineralized intervals, which are typically somewhat silicified, to be accurately sampled.

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

Tt and Mr. Robert Perry have reviewed all of the Pinnacle's sample preparation, handling, analyses, and security procedures. It is Tt's opinion that the current practices meet NI 43-101 and CIMM defined requirements.

13.1 Core Sample Preparation and Security

In the period from 2004 to 2008, no aspect of sample preparation was conducted by an employee, officer, director or associate of the Issuer. As for the period from 1982 to 1994, Tt and Mr. Perry are not aware whether any aspect of sample preparation was conducted by an employee, officer, director or associate of the Issuer.

Prior to 2004, the operators used Min-En (presently Assayers Canada), ALS Chemex, Vangeochem and the Westmin Lab at the Premier Gold mill for assaying. Although the first three are believed to be reputable commercial labs with internal sample preparation standards and independent staffs; however, it is not possible to say with certainty that no aspect of the sample preparation was conducted independent of any employee, officer, director or associate of the operator.

For all drilling done in the period 2004-2006, all samples were prepared and analysed by Assayers Canada of Vancouver, British Columbia (ISO/IEC 17025 accredited). As well as their ISO accreditation, Assayers Canada has been accredited by Standards Council of Canada as a proficiency testing provider for specific mineral analysis parameters by successful participation in proficiency tests.

The 2004-2008 samples were prepared according to the following procedure; crushed with a jaw crusher and then passed through a secondary crusher so that 60% of the sample passes #10 mesh. The sample was mixed, and a 250 g sub-sample split is taken using a riffle splitter. The sub-sample was then pulverised in a ring pulveriser until 90% of the sample passed 150 mesh. Both the crusher and the pulveriser were cleaned with pressurised air to prevent contamination.

13.2 Sample Analysis

For silver and base metals, a 1.0g sample was digested by four acid digestion and analysed by atomic absorption spectrometer. Assays were reported to a detection limit of 0.1 g/t for silver and 0.01% for base metals.

13.3 Precious Metal Assay Analysis

Assayers Canada gold assays were done by fire assay with atomic absorption finish using 30 gram samples. Assays were reported to a detection limit of 0.01g Au/t.

13.4 Quality Control

Assayers Canada automatically employed standards and blanks in their normal assay procedure. Starting in 2006, Pinnacle began introducing duplicate samples and developed a database of 1,258 duplicate results in their overall program of 9,983 samples. Minefill included a chart (their Figure 15.3) of duplicate vs original samples for the 2006 drill program showing excellent correlation with an R^2 of 0.9955.

Bitterroot reviewed the complete drillhole database and associated quality control data available in Pinnacle's possession. In its report, Bitterroot said the following:

"The largest components of that quality control data are the Pinnacle compilations of analytical control data, replicates, and duplicates. The various worksheets included documentation of an umpire assay program, wherein the company sent selected pulps and duplicate core samples out to an independent lab for comparison and confirmation of the primary lab data. Snowden also conducted a small core resampling program in 2008 to verify mineralization and assess total error in the sampling, preparation, and assay process.

There is ample evidence in these data of a quality control program in place at the Silver Coin project since at least 2005. The company included analytical control samples at several concentration levels, including analytical blanks. The company also used laboratories that employ internal quality assurance and control programs. In addition, the company documents a program of re-analyses to provide checks on the primary lab. They also went back to systematically re-sample drill core so that total variability of field sampling procedures and lab procedures can be assessed."

Minefill (Stone and Godden 2007) made an effort to validate and verify preexisting exploration data and any quality control data associated with that. In that report,iIn addition to the duplicate sample program in 2006, Bitterroot noted that in the current database duplicate assays exist in the data from 2004 through 2008, suggesting that perhaps the 2004 and 2005 duplicates were done retroactively in response to the Minefill recommendations. Starting in 2007, Pinnacle began a program of check assays and has developed a database of comparative assays between Assayers Canada (the primary lab), and ALS Chemex Labs.

Minefill and Snowden (2008) did extensive verification comparing original assay certificates with Pinnacle's computer database. They found robust records with good correlation back to 1993. The 1990 data was substantially not verifiable to their standards and most of it was omitted from the database.

Pinnacle has documented its duplicate-assay and analytical control program and demonstrated that there is no evidence of major systematic errors or bias in that data.

14.0 DATA VERIFICATION

Mr. Robert Perry conducted a site visit to the Silver Coin Gold Project area on April 20 through 21, 2009, and again in August 2009. During this time Pinnacle staff discussed the history of the project, all available data, answered questions posed by Mr. Perry, and presented the current geologic interpretation of the Silver Coin deposit. This section details the results of Mr. Perry's verification of existing data for the Silver Coin Gold Project.

14.1 Topography

Topographic control on the Silver Coin Gold Project site has developed through several stages. Historic exploration and development work was completed based on an orthogonal mine grid established with a north-south baseline at a bearing of approximately 360°. Drillholes, pit development, mine facilities, and historical mapping data were recorded in this system. Using recent survey work on the property. Pinnacle has developed a coordinate transformation formula that converts the mine grid coordinates to NAD 83 Zone 9 UTM coordinates to satisfy government reporting requirements and facilitate on-site activities using portable GPS receivers.

14.2 Assessment of Selected Silver Coin Drill Core

Mr. Alex Walus and Mr. Robert Perry traveled to Stewart on April 20, 2009 and spent about 1.5 days reviewing Silver Coin core. The goal was to develop a sense of the visual nature of the mineralized rock and assess the quality and consistency of the core logging. The majority of the core is no longer available, but the core since 2005 is stored in covered outdoor racks at the Pinnacle office in Stewart. There is a well-lit indoor facility for logging core.

At the time of the visit up to six feet of snow remained in the area surrounding the core storage facility (FIGURE 14-1) making review of some of the holes impractical. Fortunately, most of the holes of interest were reasonably accessible and nine important holes were pulled from the racks and reviewed.

These holes (FIGURE 14-2) were chosen to sample core across most of the deposit in an effort to review the evidence for the major Anomaly Creek fault zone, and to compare and contrast examples of barren, slightly mineralized, and well-mineralized intervals. The time and available core were adequate to achieve these goals.

The two most significant geological challenges at Silver Coin are the difficulty of identifying basic rock types and predicting zones of gold mineralization based on visual appearance of the core. Both of these challenges are real and have and will continue to make core logging and geologic interpretation difficult.

In addition to pervasive greenschist metamorphism, the rocks at Silver Coin have undergone two or more episodes of alteration including silicification and argillic alteration associated with one or more of the mineralizing events. This alteration obscures the original lithology and has contributed to a substantial level of disagreement between logging by different geologists in different years. These differences show up in efforts to map the underground geology and in construction of sections and level plans. Significant effort has already been made to reinterpret and standardize the core logs. To date, this has been most effective in identifying faults, structural fabrics, silicification and brecciation. Unfortunately, it has

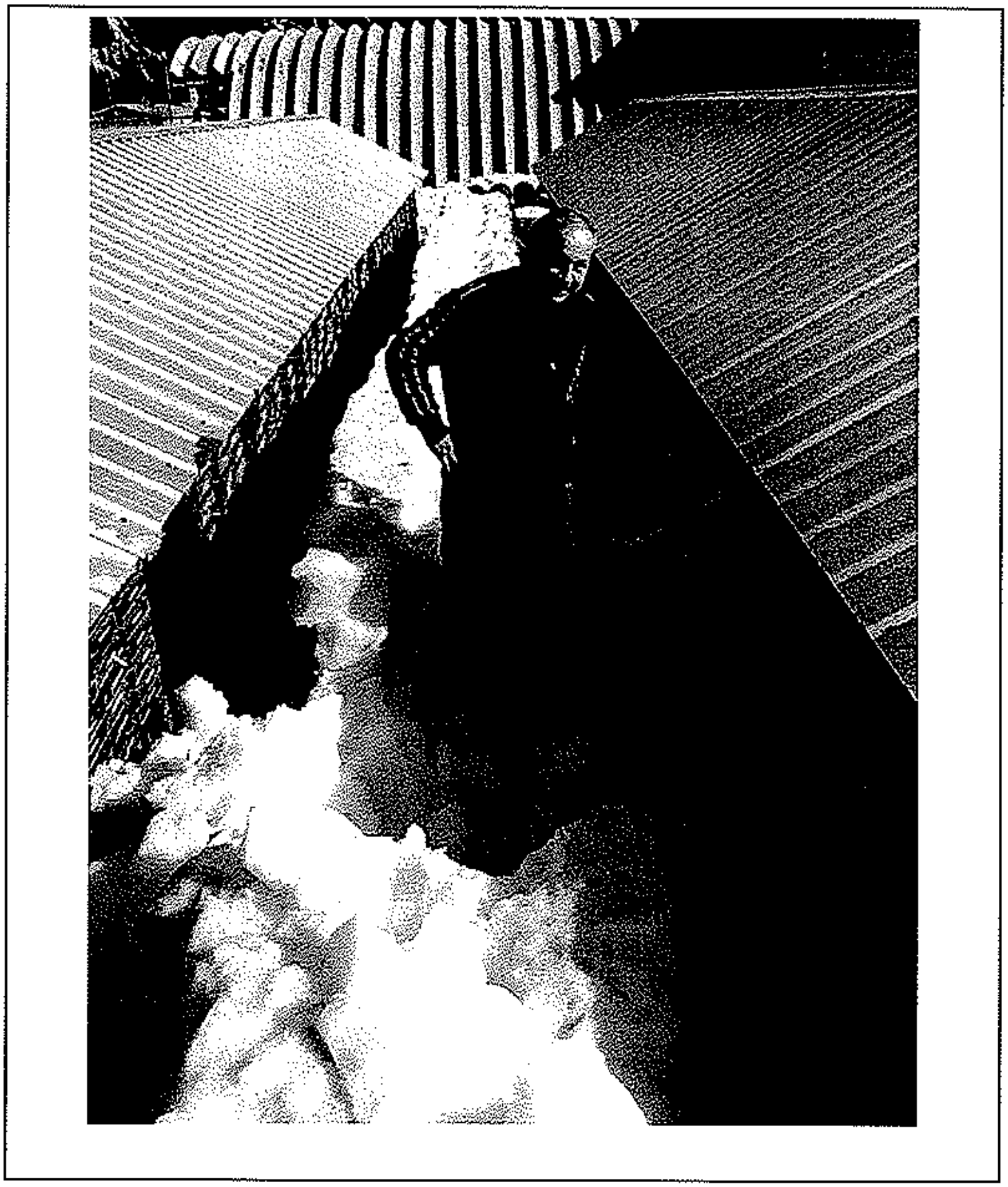
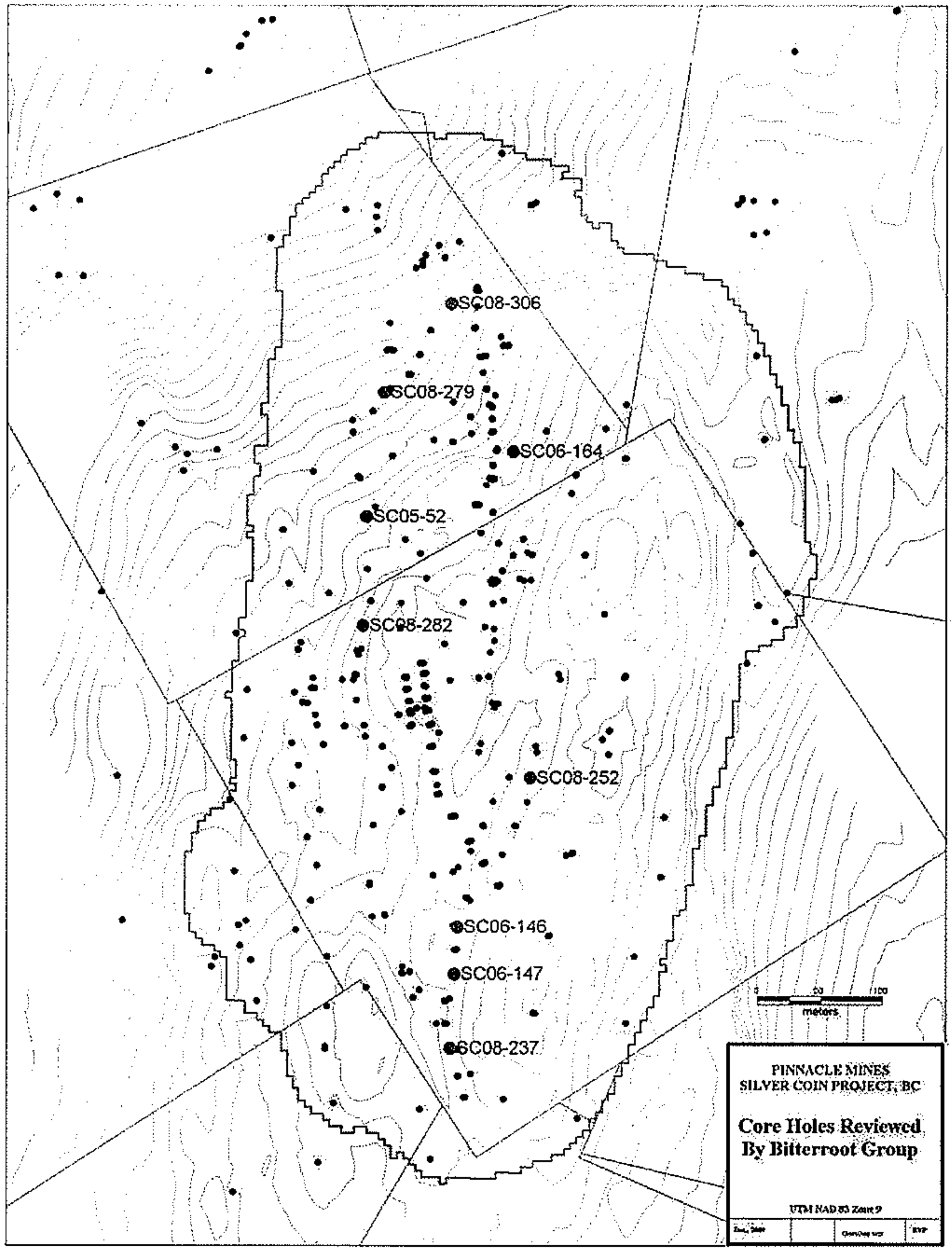


FIGURE 14-1: PHOTOGRAPH OF MR. ALEX WALUS IN STEWART APRIL 21, 2009 AT THE CORE STORAGE AREA



Issued by:	Drawing Provided by / Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig14-2.cdr	**Figure 14-2** **Location of Reviewed Drillholes at the Silver Coin Gold Property**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/30/2009	

been less effective in developing a reliable and consistent picture of the lithology from hole to hole. Sections show improbable disagreement in rock types between adjacent holes logged by different geologists. In some cases, careful review of the core descriptions makes it possible to reasonably change the rock type one geologist chose and eliminate the problem.

Although not necessarily critical to describing or sampling core, the lack of correlation between typical visual indicators of mineralization and gold grade is a frustrating aspect of logging core at Silver Coin. While alteration and veining are apparent and widespread, there is little to see in the core that reliably indicates mineralization and no way to visually distinguish between low- and high-grade mineralization (FIGURE 14-3).

Holes Reviewed

While in Stewart, Mr. Perry was able to review selected intervals of the following drill core:

SC05-52 Interval 509'-529': This hole was chosen to see the high-grade interval from 524'-529' which, unfortunately, was completely missing from the box. The interval 509'-518' assays 0.83 g Au/t and is a strongly silicified light gray-green fine grained andesite that grades into a zone of 50% brecciated quartz veins with trace to 1% fine grained pyrite and a trace of chlorite in the white quartz. From 514'-519' the grade increases to 5 g Au/t and the rock is predominantly a brecciated quartz vein with a trace of pyrite and galena. Later cross-cutting 1-2mm quartz veinlets host druzy quartz. Larger euhedral quartz crystals can be seen in hexagonal section in the white quartz matrix. From 519'-524' the grade drops to 1 g Au/t and is visually similar to the interval above with the notable exception that pyrite increases to 5% with traces of galena.

SC06-146 Interval 386'-439': This hole is in the southern part of the deposit and hosts a zone of crystal tuff with an interval of 10-13 g Au/t within average grade gold. The rock is a gray-green sparsely quartz-veined crystal tuff. An interval of 3 g Au/t in the upper part of the interval looks less favorable than the underlying rock that assays less than 1 g Au/t and shows stronger veining and some deformation. The interval 414'-424' assays 12.9 and 10.5 g Au/t and does exhibit stronger quartz veining and breccia with 1-5% pyrite. The underlying intervals host trace to 1% tan sphalerite and galena and assays in the 2-3 g Au/t range.

SC06-147 Interval 424'-446': This hole hosts a strong fault zone in the interval 434'-460'. The un-faulted host rock is tan to green andesite breccia with 15% quartz/calcite veinlets. The veinlets contain 1-3% pyrite and sparse 1mm galena veinlets. Toward the underlying fault, the brecciation, quartz veining and galena mineralization become stronger. The majority of this fault is simply core rubble fragments although some clayey fault gouge is preserved. The interval 445'-446' is strongly banded with rounded breccias fragments, suggesting that it is a mylonite zone (FIGURE 14-4). Overall, this section of core suggests that this fault zone may have been active both before and after mineralization, similar to Alldrick's (1993) observation of deformed wall rock adjacent to brittle-fractured faults. The increasing quartz veining and galena toward the more broken core suggests pre- or syn-mineral faulting. The mylonite zone is contained in an interval of almost no gold and may have been less permeable. The clayey gouge and unhealed core indicates strong post-alteration/mineralization movement in this zone.

SC06-164 Interval 370'- 397': This is another interval of strong faulting. The rocks are somewhat deformed but apparently, later re-broken. There appear to be zones of banded sulfide that have been deformed and the host rock, while very altered, looks more felsic.

SC08-237 Interval 162'- 212': This hole is located at the far southern end of the deposit and hosted very good gold mineralization (177'-327', 150' (45.7m) @ 3.78 g Au/t) representing an interesting extension of potential. Unfortunately, core with the best gold intervals wa buried in snow and not accessible. The host rock is brecciated fine- grained andesite with sparse quartz/calcite

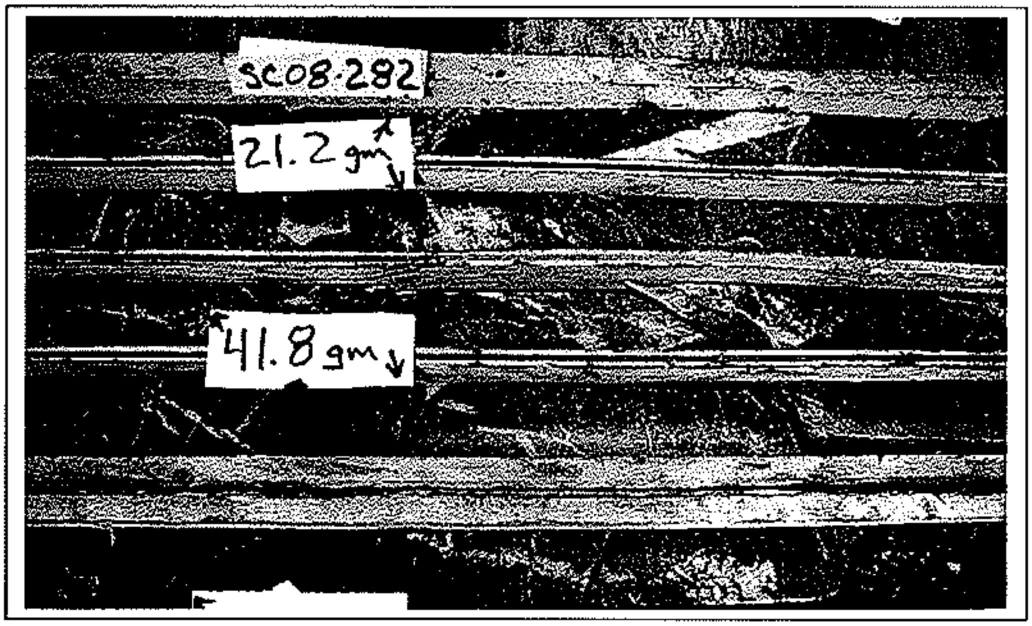


FIGURE 14-3: PHOTOGRAPH OF SILVER COIN HOLE 08-282 ILLUSTRATING THE DIFFICULTY OF ESTIMATING GREADE ON VISUAL INDICATORS

FIGURE 14-4: PHOTOGRAPH OF EARLY MYLONITE ZONE WITH LATER RE-ACTIVIATION OFTHE FAULT

veining and traces of galena and sphalerite. Ghost fragments of what appeared to be flattened pumice fragments suggest a tuffaceous origin for this rock. There is a slight increase in disseminated sulfide content in higher-grade intervals. One block of core showed clearly deformed quartz veins (FIGURE 14-5) as additional evidence for an early pre-deformational mineralizing event.

SC08-252 Interval 560'-585': This hole intersected a strong fault zone with clayey gouge (FIGURE 14-6) that juxtaposes different rock types on the hanging and footwall sides and apparently controlled emplacement of a flow-banded felsic dike. The upper part of the interval is dense green andesite porphyry or a crystal tuff with strong chlorite/sericite alteration with trace disseminated pyrite and minor 1mm quartz/calcite veins. The underlying fault zone is strongly sericitized and full of gouge material. The fault contains an interval of strongly banded felsic rock containing possible pumice fragments. This is followed by 10' of pure gouge with only 2' of core recovery for the interval. At the bottom of the zone, the rock appears to be a strongly chlorite/sericite/carbonate altered andesite flow. This hole also supports the idea that this fault zone was active at more than one time.

SC08-279 Interval 67'-105': Starting at 67' the rock is fine-grained gray tuff breccias with sparse quartz veinlets and 1mm to 4cm breccia fragments. It is pervasively sericitized. 2mm quartz veins contain 1-2mm pyrite crystals with trace sphalerite. At 83' the color becomes tan to light-brown and pyrite content increases, with 1% galena in 5mm quartz veins. At 85-89' the core is very broken and most of the interval from 89-99', which assayed over 23 gm/t Au, is missing.

SC08-282 Interval 419'-560': As other geologists have noted, identification of both lithology and grade of the core at Silver Coin is very difficult. This interval in drillhole SC08-282 is a great example of the both issues. The host rock is a mixed green andesite with zones of probable crystal tuff. The presence of altered feldspar phenocrysts suggests a primary lithology of crystal tuff but the strong alteration often leaves just ghosts of the crystals and in places it is possible that these represent secondary K-spar from potassic alteration. As shown in FIGURE 14-3, dramatic swings in gold grade with no apparent visual indicators are common. The rock type, degree of quartz veining and sulfide content are similar from sample to sample. One minor difference is increased galena content in the interval 470'-473' and this corresponds roughly to a 5' interval assaying 58 gm/t Au. However, in this and other holes, galena content is not a reliable indicator of gold values.

SC08-306 Interval 372'-448': This hole is located at the north end of the resource area. The lower part of the drillhole (380'-410') hosts an interval of 30' @ 8.8 gm/t Au that lies immediately above a fault zone. The highest grade sample is a 10' zone (380' - 390') assaying 13.6 gm/t Au; and the interval is unusual in that it actually looks mineralized. The rock is a dark gray to black, pyrite-rich sheared breccia. The black color is likely black chlorite since it does not have a black streak, but it could also be graphite. The lower part of this hole also contains zones of felsic rock, and one possibility is that it is a fault-mixed zone of andesite and felsite between overlying andesite and underlying felsite.

General Comments about the Core:

- The rock is very altered and very difficult to identify with certainty or consistency
- There is no reliable way to predict assay grade from visual inspection. The degree of quartz veining, silicification, sulfide content, and even presence of galena and sphalerite are inconsistent and unreliable indicators of gold grade.
- The logs need more detailed lithologic logging.

FIGURE 14-5: PHOTOGRAPH OF SILVER COIN HOLE 08-237 DEFORMED QUARTZ VEIN

FIGURE 14-6: PHOTOGRAPH OF SILVER COIN HOLE 08-252 CLAYEY GOUGE ZONE AT CONTACT WITH FELSIC VOLCANIC ROCK

- There seem to be at least two generations of sphalerite. One is red and the other is light to dark brown.
- There may be two generations of galena, one being shiny and cubic and the other being darker and subhedral.
- The core boxes and the footage blocks are both poorly marked. There are metal tags with the hole and box numbers inscribed on the ends of the boxes but the magic marker markings are often difficult to read and most likely will not survive with time. In the future, loggers should include a metal tag in the core box with the hole and box number plus the box interval in meters.
- The core is drilled and marked in the boxes in feet while the logs and assays are all in meters. This makes comparing the core to the logs inconvenient. One option is to remark the blocks in the core boxes in meters at the time of logging. Another option could be to include from-to columns in feet in the digital drill logs.
- Using preprinted sample tags would allow the logger to staple one of the duplicate tear-off tabs into the core box for a permanent record of the sample number in the box with the core.

Conclusion of Data Verification

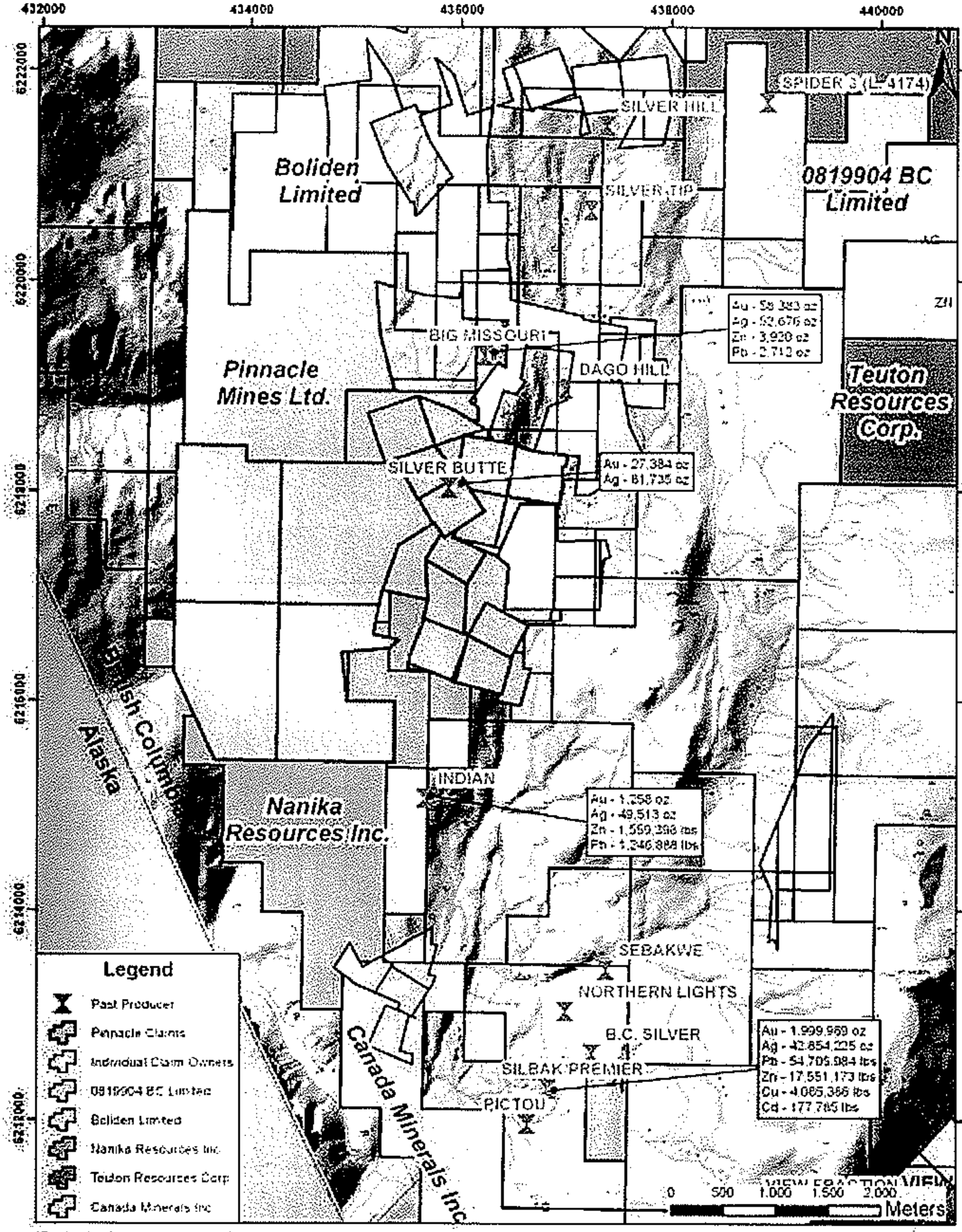
Silver Coin core for the period 2005 through 2008 is stored in secure, covered storage racks in Pinnacle's warehouse and core yard in Stewart, BC. The storage area is enclosed in a 2m high fenced area with locked gates. The warehouse and logging area are attached to the core yard making the facility very well suited to storage, processing, logging and review of the core. The core and core boxes are well organized, clearly marked and in good condition.

Some of the core from 2004 is no longer available and virtually no core remains from drilling done prior to 2004.

During the site visit Mr. Perry visited the pad and mine dump for the now caved Portal Number 2 and proceeded to traverse the full extent of the known deposit from north to south including visiting No Name Lake, the potential site for tailings disposal. While on site several drillhole collars were located with a hand-held GPS to confirm their location compared to the drill collar locations in the Pinnacle data base. Within the accuracy of the hand-held GPS, the measured collar locations confirmed those in the database.

15.0 ADJACENT PROPERTIES

The Silver Coin property is located in an area with several historical mines. Locations of these mines are presented on FIGURE 15-1. The Big Missouri Mine located just to the north of the property produced 768,943 tonnes at an average grade of 2.37 g Au/t and 2.13 g Ag/t in the period from 1938 to 1942. The Indian Mine to the south produced 12,870 tonnes averaging 3.40 g Au/t, 119.7 g Ag/t, 4.40 % Pb and 5.50 % Zn. The property is contiguous with the large Premier Gold property which produced, between 1918 and 1979, 4.2 million tonnes of ore at a recovered grade of 13.4 g Au/t, 301 g Ag/t, 2.3% Cu, 0.6% Pb and 0.2% Zn (BCEMPR production statistics). Reportedly, 6,500,000 tonnes of 2.16 g Au/t and 80.23 g Ag/t were mined by Westmin in the period 1988 to 1995. Reported remaining reserves include 300,000 tonnes of 8 g Au/t. **All of the above information is included on the Minefill website posted by the Ministry of Energy, Mines and Petroleum Resources. Neither Tt nor Mr. Perry verified the information presented in this item nor is this information necessarily indicative of the mineralization present on the Silver Coin property. The historical estimates of reserves quoted in this section are disclosed in accordance with Section 2.4 of the Instrument.**



*Production figures are from the Digital Minfile Database provided by the Ministry of Energy, Mines and Petroleum Resources, Government of British Columbia

**Historically Staked Claims take precedence over claims staked with the Mineral Titles Online System

Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Drawing Provided by/Prepared for: Pinnacle Mines Ltd.	File Name: Fig15-1.cdr	**Figure 15-1 Location of Adjacent Property to the Silver Coin Gold Project**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Metallurgical Testing

A scoping-level metallurgical program was conducted on selected drill core from the Silver Coin Gold Project during the period of 2005 to 2009. Laboratory studies were primarily performed by Process Research Associated Ltd. ("PRA") under the supervision of Mr. Frank Wright. This program investigated several different process routes for the recovery of the contained gold and silver values, including:

- Flotation
- Whole-ore cyanidation
- Cyanidation of flotation concentrates

The results of this metallurgical program are documented in the report, "Metallurgical Study on the Silver Coin Gold Project", prepared by F. Wright Consulting Inc., January 8, 2009, and is included in APPENDIX A

16.1.1 Sample Preparation and Analyses

PRA received 95 samples of split drill core in July 2008 with which to conduct the metallurgical program. Each of the 95 samples was crushed to minus 10 mesh, and then segregated and blended into eight composite samples. Each composite sample represented a continuous interval of drill core from various spatial areas and depths of the resource. A blended master composite (MCI) was also made from the eight composite samples. Analytical work was performed by iPL Laboratories, which has ISO 9001 accreditation. Head analyses for each of the composites are provided in TABLE 16-1.

TABLE 16-1: SUMMARY OF HEAD ANALYSES
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

Composite	Au (g/t)	Ag (g/t)	%Pb	%Zn	%ST
O8-1	0.41	2.3	0.06	0.11	2.2
O8-2	1.35	7.6	0.32	0.57	4.11
O8-3	1.45	8.3	0.11	0.73	4.62
O8-4	1.69	8.9	0.31	1.11	8.44
O8-5	2.88	22.7	0.53	1.4	5.46
O8-6	0.38	5.5	0.02	0.04	2.3
O8-7	1.85	3.5	0.07	0.25	2.45
O8-8	1.96	5.2	0.02	0.03	5.27
MC1	1.87	7.1	0.07	0.57	4.55

16.1.2 Flotation Studies

Flotation studies were conducted to evaluate rougher flotation as a function of grind on both low-and high-sulfur composites. These tests provided a preliminary indication of the grind fineness required to maximize gold recovery into a rougher flotation concentrate. These tests were followed by open-circuit cleaner flotation studies to evaluate the extent to which the rougher flotation concentrate could be upgraded. A locked-cycle flotation test was then conducted to evaluate the effect of recycling intermediate process streams on overall gold and silver recovery into a final cleaner flotation concentrate.

Rougher Flotation – Grind Vs. Recovery

Scoping-level rougher flotation studies were conducted on both low-sulfur and high-sulfur composites to evaluate the effect of grind fineness on gold recovery. It was found that the contained gold is very amenable to recovery into a bulk sulfide concentrate, and that gold recovery is relatively insensitive to grind fineness over the range tested. Gold recovery from the low-sulfur composite was about 94 percent (except at the finest grind where gold recovery increased to 98 percent). Gold recovery from the high sulfur composite was 97-99 percent over the grinds tested. The results of these tests are summarized in TABLE 16-2. These tests indicate that primary gold recovery can be accomplished at a relatively coarse grind. Additional studies will be required to optimize the grind size.

TABLE 16-2: SUMMARY OF GRIND VS. ROUGHER FLOTATION RECOVERY
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

		Grind	Calc. Head		Bulk Rougher Conc.	
Test No.	**Composite**	**P80, microns**	**Au (g/t)**	**Wt %**	**Au Recovery**	**S Recovery**
F1	Low Sulfur	51	1.26	37.8	98.0	98.9
F2	Low Sulfur	71	0.91	30.1	93.9	97.7
F3	Low Sulfur	113	0.75	28.1	93.8	98.1
F4	Low Sulfur	170	0.85	24.6	93.8	97.5
F6	High Sulfur	53	1.97	32.2	99.3	99.3
F7	High Sulfur	70	2.23	29.6	98.7	99.4
F8	High Sulfur	113	1.91	26.3	98.5	99.0
F9	High Sulfur	183	1.96	23.2	96.9	98.0

Rougher Flotation Kinetics

Gold recovery as a function of rougher flotation retention time was also monitored at each of the grinds tested. Cumulative gold recovery and cumulative rougher concentrate weight percent as a function of rougher flotation retention time are shown in TABLE 16-3. These results indicate that gold recovery into the rougher concentrate is nearly complete after about 10-15 minutes of flotation. Laboratory flotation times beyond this resulted in pulling more sulfides into the concentrate without contributing significant additional gold recovery.

TABLE 16-3: CUMULATIVE GOLD RECOVERY VS. FLOTATION RENTION TIME
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

Test No.	Composite	Grind P80, microns	Calc. Head Au (g/t)	Cumulative Gold Recovery At Specified Retention Time 5'	10'	15'	20'	Cumulative Concentrate Wt% At Specified Retention Time 5'	10'	15'	20'
F1	Low Sulfur	51	1.26	90.4	97.1	97.7	98.0	11.5	23.6	30.6	37.8
F2	Low Sulfur	71	0.91	90.4	92.4	93.2	93.9	12.3	18.6	24.4	30.1
F3	Low Sulfur	113	0.75	88.7	91.7	92.9	93.8	11.0	18.3	23.5	28.1
F4	Low Sulfur	170	0.85	88.7	91.3	92.9	93.8	10.6	15.8	20.6	24.6
F6	High Sulfur	53	1.97	83.6	98.7	99.2	99.3	6.9	18.2	25.8	32.2
F7	High Sulfur	70	2.23	95.1	96.1	98.6	98.7	13.6	20.0	25.1	29.6
F8	High Sulfur	113	1.91	95.5	97.5	98.2	98.5	12.2	17.2	22.0	26.3
F9	High Sulfur	183	1.96	90.9	95.4	96.3	96.9	10.4	15.4	19.6	23.2

Open-Cycle Cleaner Flotation

Open-circuit cleaner-flotation studies were conducted in order to determine the extent to which rougher-flotation concentrates could be upgraded. Although a number of different cleaning procedures were tested, the more standard approach of regrinding, followed by cleaner flotation at an elevated pH of 11.5 was considered the most effective. For these tests, the bulk rougher concentrate was reground to P_{80} 74 microns and subjected to 3 to 4 stages of cleaner flotation at pH 11.5. This resulted in overall gold recoveries ranging from 68-82 percent into cleaner concentrates grading 25 g au/t to 203 g Au/t. The results of these tests are summarized in TABLE 16-4. It should be noted that the relatively low recoveries into the final cleaner concentrate are not indicative of actual plant recoveries since these tests were done in open circuit without considering the effect of recycling the intermediate products. The results of a locked-cycle test, which was designed to recycle intermediate test products, are discussed in the next section.

Locked-Cycle Flotation

A locked-cycle test was conducted on the master composite (MCI) in order to evaluate the effect of recycling the intermediate flotation products. This test was run for 6 cycles according to the process flowsheet. The rougher concentrate and first cleaner scavenger concentrate were cycled to the regrind mill, the rougher-scavenger concentrate was recycled back to primary grinding, and the cleaner flotation tailings were recycled counter-currently to each preceding stage of cleaner flotation.

TABLE 16-4: SUMMARY OF OPEN CIRCUIT CLEANER FLOTATION TESTS (ELEVATED PH) PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009							
		Grind	Calc. Head	Rougher Conc.	2nd or 3rd Cleaner Conc.		
Test No.	Composite	P80, microns	Au (g/t)	Au Recovery (%)	Wt%	Au (g/t)	Au Recovery
F20	O8-4	74	1.73	91	0.9	131	68.2
F21	O8-7	74	2.55	88	2.7	76	80.5
F22	O8-1	74	0.41	94	1.3	25	78.3
F23	O8-3	74	1.70	95	1.3	103	78.8
F24	O8-8	74	2.27	85	0.8	203	71.5
F25	MC1	74	1.77	92	2.3	63	81.6

This locked-cycle test resulted in 94.7 percent overall gold recovery into a fourth cleaner concentrate grading 110 g Au/t, and demonstrated that relatively high gold recoveries and upgrading could be anticipated from a continuously operated flotation circuit designed to regrind and recycle intermediate products. The results of the locked-cycle test on the MCI composite are summarized in TABLE 16-5.

TABLE 16-5: SUMMARY OF LOCKED-CYCLE TERSTS (6 CYCLES)* PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009										
		Grind	Calc. Head		Cl. Conc	4th Cleaner Conc. Grade			Cl. Conc. Recovery	
Test No.	Comp.	P80, microns	Au (g/t)	Ag (g/t)	Wt%	Au (g/t)	Ag g/t)	%ST	Au (%)	Ag (%)
FLC1 -08	MC1	74	2.07	5.45	1.8	110	273	42.1	94.7	89.4

- Calculation excludes the intermediate products from 6th cycle.

16.1.3 Cyanidation

Preliminary cyanidation tests were conducted to evaluate the potential of whole-ore cyanidation and cyanidation of cleaner flotation concentrates.

Whole-Ore Cyanidation

Whole-ore cyanidation was tested on two composite blends by both straight cyanidation (without carbon) and by carbon-in-leach (CIL) cyanidation. The two composite blends consisted of 1:1 blends of composites 08-1 and 08-2 to represent a low-sulfide composite and a 1:1 blend of composites 08-5 and 08-6 to represent a high-sulfide composite. The cyanidation tests were all conducted at a slurry density of 40 percent solids at a cyanide concentration of 2 g/l NaCN and pH 10.5. All tests were run for 96 hours, but no intermediate sampling was done to evaluate the leach kinetics. The results of these tests are summarized in TABLE 16-6 and the following observations can be made:

- CI and CIL-1 cyanidation tests at a grind of P_{80} 70 microns resulted in gold extractions of 89 percent and 87 percent, respectively. The relatively similar gold extractions for these two tests are an indication that this composite blend did not exhibit preg-robbing characteristics. Silver extraction for both tests ranged from 68-71 percent.
- Cyanide consumption for Tests CI and CIL-1 averaged 4.2 kg/tonne NaCN and hydrated lime consumption averaged 0.5 kg/tonne $Ca(OH)_2$.
- C2 and CIL-2 cyanidation tests at a grind of P_{80} 67 microns resulted in gold extractions of 75 percent and 85 percent, respectively. The fact that gold extraction from test C2 was 10 percent lower than Test CIL-2 is a preliminary indication that preg-robbing may be exhibited by this composite blend. Silver extraction for both tests was about 62 percent.
- Cyanide consumption for Tests C2 and CIL-2 averaged 4.0 kg/tonne NaCN and hydrated lime consumption averaged 0.45 kg/tonne $Ca(OH)_2$.
- Cyanide consumption is considered relatively high, and could likely be reduced by inclusion of pre-aeration at high pH to passivate some sulfide mineral surfaces prior to cyanidation.
- Some additional tests were conducted that included gravity concentration of the gold followed by cyanidation of the gravity tailing. Although the tests were very preliminary in nature and are not reported in this presentation, they do serve to indicate that gold recovery could be enhanced by the inclusion of gravity concentration as part of the process flowsheet.

TABLE 16-6: SUMMARY OF WHOLE ORE CYANIDATION TESTS
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

		Grind	Calc. Head		Extraction, %	
Test No.	Composite	P80, microns	Au (g/t)	Ag (g/t)	Au	Ag
CI	O8-1 + O8-2	70	1.40	6.8	89.3	68.3
CIL-1	O8-1 + O8-2	71	0.84	5.3	86.8	71.5
C2	O8-5 + O8-6	67	2.08	17.1	75.2	61.9
CIL-2	O8-5 + O8-6	67	1.79	14.8	84.9	62.9

Cyanidation of Flotation Concentrates

Selected flotation concentrates produced from the composites were reground to approximately P_{80} minus 55 microns and subjected to CIL cyanidation. These tests were conducted for 96 hours and included, cyanide at 2 g/l NaCN, pH maintained at 10.5–11.0 and carbon concentration at 20 g/l. The results of these tests are summarized in TABLE 16-7 and indicate gold extractions of 90-96 percent from flotation cleaner concentrates. Cyanide consumption averaged 4.5 kg/tonne of concentrate and hydrated lime consumption averaged 0.32 kg/tonne of concentrate

TABLE 16-7: SUMMARY OF FLOTATION CONCENTRATE CYANIDATION TESTS PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009						
		Grind	Assay Head		Extraction, %	
Test No.	Composite	P80, microns	Au (g/t)	Ag (g/t)	Au	Ag
CILF-17	O8-2	55	20	56	92	56
CILF-18	O8-3	63	20	86	90	52
CILF-19	O8-5	55	40	224	94	81
CILF-20	O8-4	n/a	118	382	96	64
CILF-21	O8-7	40	77	98	93	88

16.1.4 Comminution Studies

Comminution tests were conducted on two surface trench samples dug into bedrock in areas considered to be typical of the ore body. Sample A was characterized as rock cut by quartz-carbonate veinlets, estimated to be about 10-15 percent silicified and containing about 2-3 percent sulfides. Sample B was described as 60–70 percent silicified and containing 5-7 percent sulfides. Both samples were sent to the Metso Mineral Research and Test Center for testing, which included determination of the Bond Crushability Index, Bond Paddle Abrasion and Bond Ball Mill Index. The results of this work are documented in the Metso Test Report, prepared for Pinnacle Mines dated October 21, 2009, and which is included in APPENDIX B.

The comminution test results are summarized in TABLE 16-8. The ore would be characterized as medium hard and highly abrasive. It is recommended that additional comminution testing be done on selected drill core intervals during the next level of study to evaluate the variability within the ore body.

TABLE 16-8: SUMMARY OF COMMINUTION TEST RTESULTS PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009				
	Bond	Crushability Index	Bond Ball Mill Index	
Sample	Abrasion Index	KWh/st	kWh/st	kWh/tonne
A	0.425	11.69	15.80	17.40
B	0.587	10.89	15.43	17.00

16.2 Processing

The metallurgical data show the possible process routes for Silver Coin ore include:

- All Flotation
- Flotation followed by cyanidation of the flotation concentrate.

The first option does not require the use of cyanide and is considered the base-case process route due to concerns regarding the use of cyanide at the project site. This option would result in the production of a low grade flotation concentrate requiring shipment to an off-site smelter.

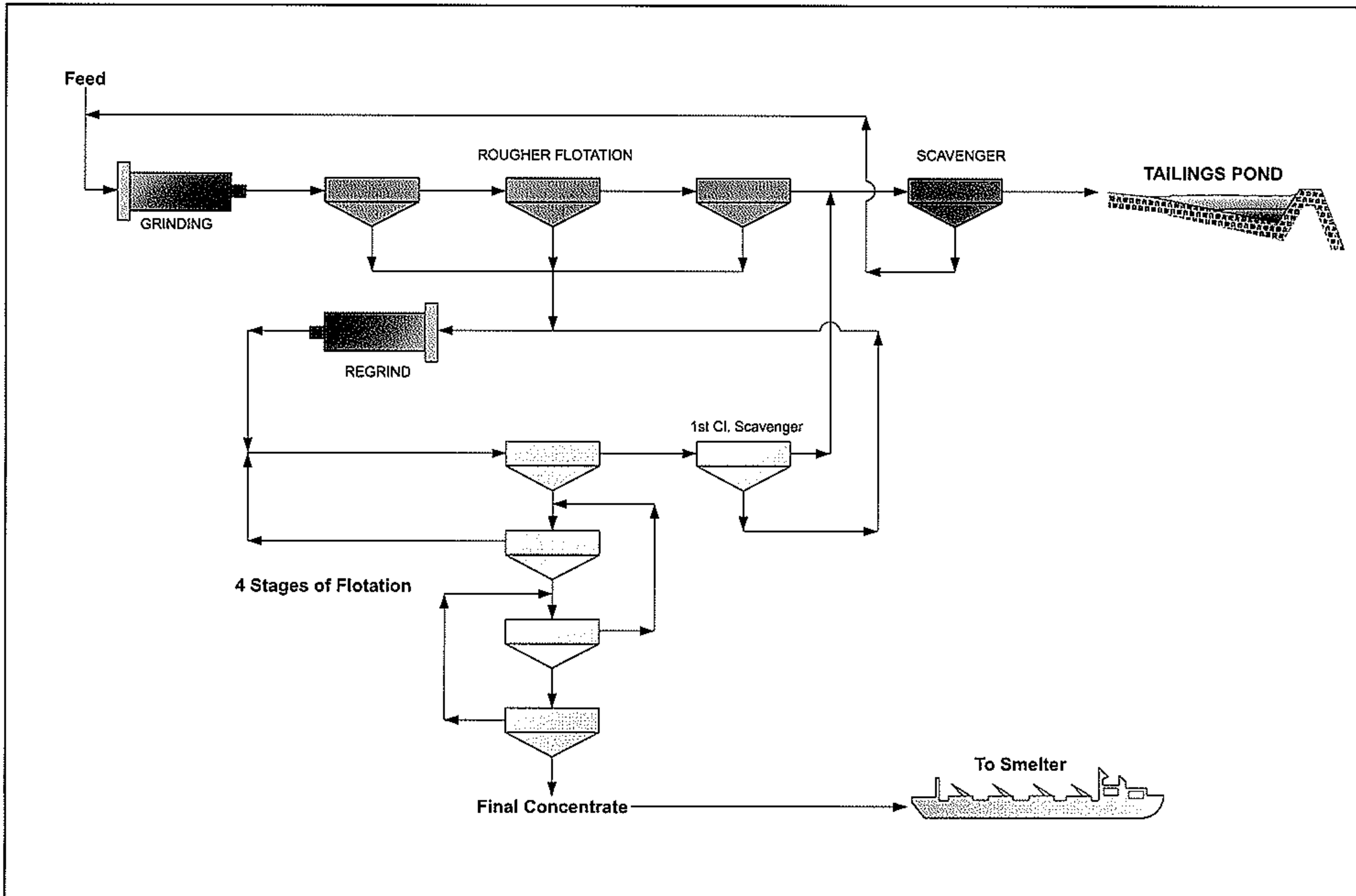
The second option involves the use of cyanide and would result in the production of a readily marketable gold-silver dore' product at site.

16.2.1 All-Flotation Process Flowsheet

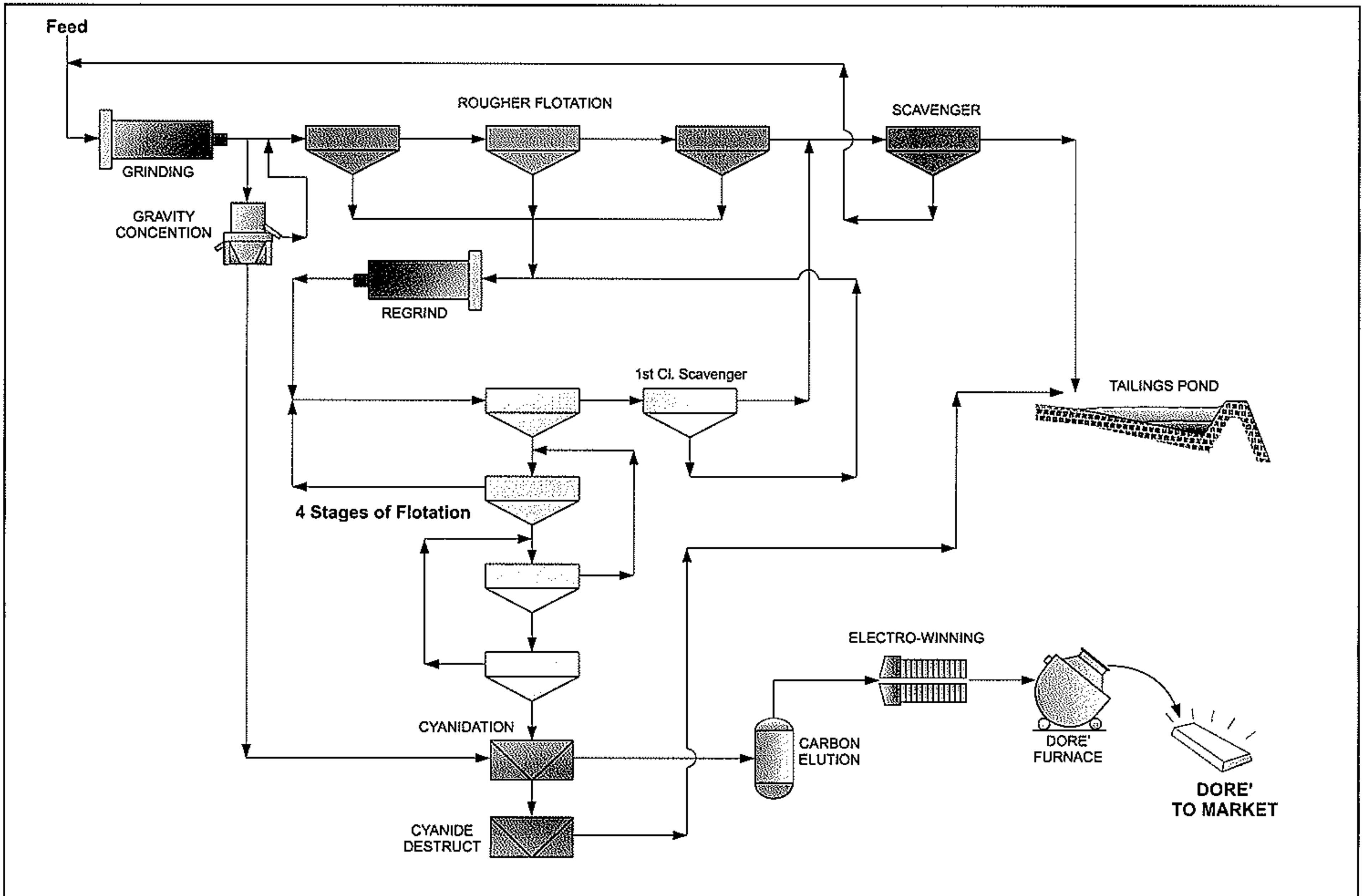
The all-flotation process flowsheet, which has been selected as the base-case for this study is shown in FIGURE 16-1. The process flowsheet would include grinding to approximately 80 percent passing 100 mesh followed by rougher bulk sulfide/gold flotation and rougher-scavenger flotation. The rougher concentrate would be reground and subjected to multiple stages of cleaner flotation to produce an upgraded bulk sulfide/gold concentrate that could potentially be shipped to an off-site smelter for refining. It must be emphasized that significant additional process development testwork is required to define the process design criteria required for an all-flotation processing facility.

16.2.2 Gravity-Flotation-Cyanidation Flowsheet

The gravity-flotation-cyanidation flowsheet is illustrated schematically in FIGURE 16-2. The process flowsheet would include grinding to approximately 80 percent passing 200 mesh in closed circuit with a centrifugal gravity concentrator. The gravity concentrate would be processed in the cyanidation circuit and the cyclone overflow would advance to a bulk sulfide/gold flotation circuit, which would include bulk sulfide rougher and scavenger flotation followed by regrinding and multiple stages of cleaner flotation. The precious metal bearing bulk sulfide would then be thickened and advanced to a cyanidation (CIL or CIP) followed by carbon elution, electro-winning and refining of the gold and silver values. The cyanidation tailings would be processed in a cyanide destruction circuit prior to discharge to the tailing pond. Again, it must be emphasized that only scoping-level metallurgical studies have been conducted, and significant additional process development testwork is required to define the process design criteria required for a gravity-flotation-cyanidation processing facility.



Issued by: TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Prepared for: *Pinnacle Mines Ltd.*	File Name: Fig16-1.cdr	Figure 16-1 All Flotation Process Flowsheet
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 01/06/2010	



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig16-2.cdr	
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 01/06/2010	

Figure 16-2
Flotation - Cyanidation Process Flowsheet

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Tt completed an independent mineral resource and reserve estimate of the contained gold in the Silver Coin deposit. Several computer programs were used in this analysis. Geostatistics and resource estimation was done with MicroModel®. Additional statistical analysis was done with Statistica®, and Excel®. Three-dimensional wireframes of modeled faults and model visualization was done with GemCom software.

Tt calculated resources for the Silver Coin deposit using both current and historical data from trenches, surface drilling and underground drilling. Both the new and historical data was verified using the original assay certificates. Tt had the advantage to carefully critique the methodologies used by two earlier resources estimates. The 2007 Minefill resource estimate used grade-shell wireframes to constrain ordinary kriging. The 2008 Snowden resource estimate employed both grade-shell wireframes and mapped faults to constrain multiple indicator kriging. Tt agrees with these earlier estimators observation that geologic wireframes of lithology are suspect due to the complex and discontinuous three dimensional distributions of silicification, brecciation and sulfidation. This complexity is a summation of interpretations done by different geologists from at least five companies and the inherent complexity of the subsurface geology. Tt used Pinnacle's re-interpretation of subsurface faulting to constrain its estimate using multiple-pass ordinary kriging.

17.1 Overview

- Establish block model parameters.
- Data validation was done by Pinnacle Mines using original assay certificates.
- Data was imported into Statistca®, GemCom® and MicroMine®.
- Data were analyzed for errors by statistical methods such as histograms, probability graphs and multi-metal (Au, Ag, Cu, Pb and Zn) correlation plots.
- Interpretation of Mineralized Zones above two subsurface faults.
- Coding of block model and drillhole data within the mineralized zones was done by GemCom® and MicroMine®.
- Bench compositing.
- Comparison of assay and composite data distributions.
- Density data statistically analyzed.
- Analysis of extreme data values and application of top cuts for gold (30 g/t) and silver (130 g/t).
- Correlation between composites for all metals i.e. Au to Ag, Au to Zn etc.
- Variogram analysis and modeling.
- Check of variogram models and multi-pass kriging parameters using jackknife analysis.
- Analysis of a kriging error break point used in resource confidence classification.
- Assignment of final kriging parameters for all metals.
- Estimation of Au, Ag, Cu, Pb and Zn into blocks using multi-pass ordinary kriging (OK).
- Statistical analysis of kriged block values using histograms.

- Visual inspection of block model values with drillhole composites using GemCom®.
- An additional study comparing block models produced with only surface drillhole data and then with only underground drillhole data.
- Selected sections showing kriged gold and silver, rock codes and resource confidence classification.
- Resource tabulation and reporting.

17.2 Model Parameters

TABLE 7-1 shows the Micromodel® parameters. The block model consists of blocks 10x10x5 m in dimension. The total model contains a potential of 131 rows, 121 columns, and 161 levels. The model has no rotation and is 1210m east-west by 1310m north-south by 805m high. A large percentage of the blocks are "air blocks". Sample, composite and block grade labels are silver, gold, copper, lead and zinc. They are silver (AgG, cAgG, kAgG), gold (AuG, cAuG, kAuG), copper (Cu%, cCu%, kCu%), lead (Pb%, cPb%, kPb%), zinc (Zn%, cZn%, kZn%). Two additional composite labels for gold and silver are listed. The first; xAgG and is for silver with a high cut at 130 g/t. The second, xAuG, is for gold with a high cut a 30g/t.

```
CURRENT TIME  :  22-Dec-09 09:04 AM
PROJECT TITLE : Silver Coin 10x10x5m Block Size
RUNTIME TITLE : Project Parameters

SAMPLE    LABELS      COMPOSITE LABELS      GRADE     LABELS

 1        AgG          1         cAgG        1        kAgG   [10x10x5m               ]
 2        AuG          2         cAuG        2        kAuG   [10x10x5m               ]
 3        Cu%          3         cCu%        3        kCu%   [10x10x5m               ]
 4        Pb%          4         cPb%        4        kPb%   [10x10x5m               ]
 5        Zn%          5         cZn%        5        kZn%   [10x10x5m               ]
                       6         xAgG
                       7         xAuG

ORIGIN IS LOCATED AT   6217400.00  NORTH     435200.00  EAST        397.50  ELEVATION
ROTATION ANGLE FROM NORTH CLOCKWISE TO THE LEFT BOUNDARY IS :    0.00

NUMBER OF ROWS     :  131                         ROW DIMENSION      :     10.00  METERS
NUMBER OF COLUMNS  :  121                         COLUMN DIMENSION   :     10.00  METERS
NUMBER OF LEVELS   :  161                         LEVEL DIMENSION    :      5.00  METERS

MODEL EXTENTS ARE:

             MINIMUM      MAXIMUM
  EASTING:  435200.00   436410.00
 NORTHING: 6217400.00  6218710.00
ELEVATION:     397.50     1202.50
```

TABLE 17-1: SILVER COIN GOLD PROJECT – BLOCK MODEL PARAMETERS

17.3 General Drill Hole Statistics

TABLE 17-2 shows the general drill hole and trench statistics for the assay values. Note that total count of 774 is broken out into 412 Surface drillholes, 287 Under Ground drillholes and 75 Trenches. The original assay values have sampling intervals that vary from one to three meters.

The average gold grade for surface drill holes (SDH) is 1.25 g Au/t, while for underground drillholes (UDH) it is 1.76 g Au/t. This is an apparent enhancement of the average gold grade of almost a half a gram between surface and underground data. This observation will be tested in more detail at the end of this chapter. FIGURE 17-1 shows, in plan view, the location of trench data (green), underground drillhole data (blue) and surface drillhole data (red).

	NORTHING	EASTING	ELEVATION	AZIMUTH	DIP	DEPTH
MINIMUM	6216707.0	435001.1	554.0	0.0	-87.4	2.0
MAXIMUM	6218677.0	436716.0	1040.4	352.0	90.0	755.8
AVERAGE	6217992.9	435815.7	911.9	153.2	31.3	106.7
RANGE	1970.0	1714.9	486.4	352.0	177.4	753.8

TOTAL COUNT 774
TOTAL LENGTH 82616.1

ASSAY INTERVAL 1 TO 3 METERS

Surface DH

TOTAL DRILLHOLES = 412
AVERAGE VALUES OF SELECTED DATA

LABEL	NUMBER	AVERAGE	STD DEVIATION	MIN. VALUE	MAX. VALUE	# MISS.
AgG	24448	6.17357	29.71736	0.03000	2453.00000	1674
AuG	24354	1.24970	49.89862	0.00500	7660.20020	1768
Cu%	20222	0.01504	0.47625	0.00030	66.30000	5900
Pb%	20349	0.06792	0.39980	0.00100	27.50000	5773
Zn%	20395	0.19873	1.10278	0.00100	85.00000	5727

Under Ground DH

TOTAL DRILLHOLES = 287
AVERAGE VALUES OF SELECTED DATA

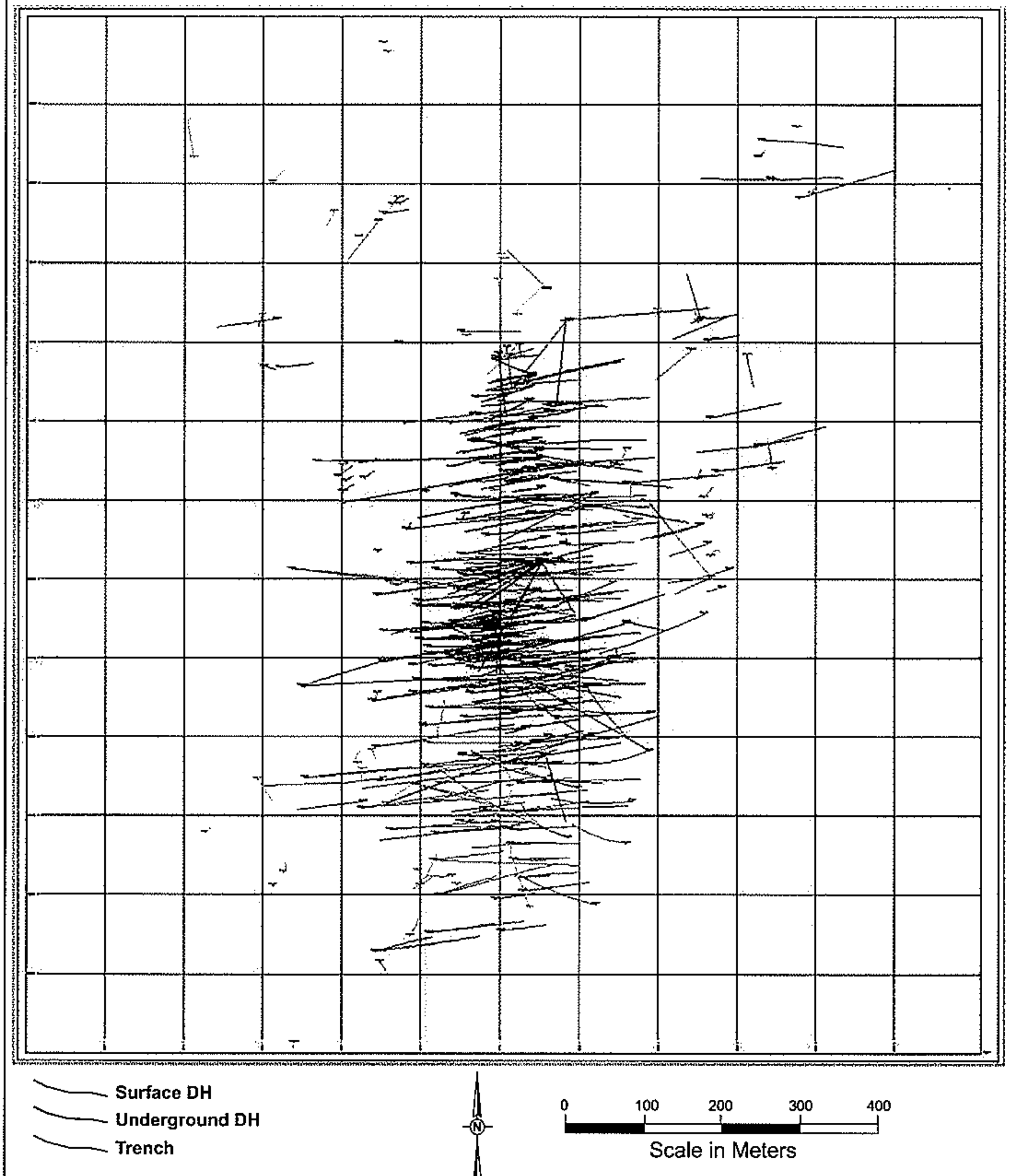
LABEL	NUMBER	AVERAGE	STD DEVIATION	MIN. VALUE	MAX. VALUE	# MISS.
AgG	5153	10.91116	28.93257	0.07000	643.00000	4735
AuG	9087	1.76509	7.98445	0.03000	388.17999	801
Cu%	3087	0.05418	0.23271	0.00030	3.66000	6801
Pb%	3139	0.18362	0.58104	0.00040	14.40000	6749
Zn%	2939	0.58992	1.90441	0.00200	30.90000	6949

Trench

TOTAL DRILLHOLES = 75
AVERAGE VALUES OF SELECTED DATA

LABEL	NUMBER	AVERAGE	STD DEVIATION	MIN. VALUE	MAX. VALUE	# MISS.
AgG	546	18.47606	127.21934	0.00000	2923.00000	6
AuG	546	0.91033	2.47766	0.00000	46.00000	6
Cu%	546	0.05895	0.27659	0.00000	4.76000	6
Pb%	546	0.24022	1.26232	0.00000	25.20000	6
Zn%	546	0.52702	1.53585	0.00000	19.80000	6

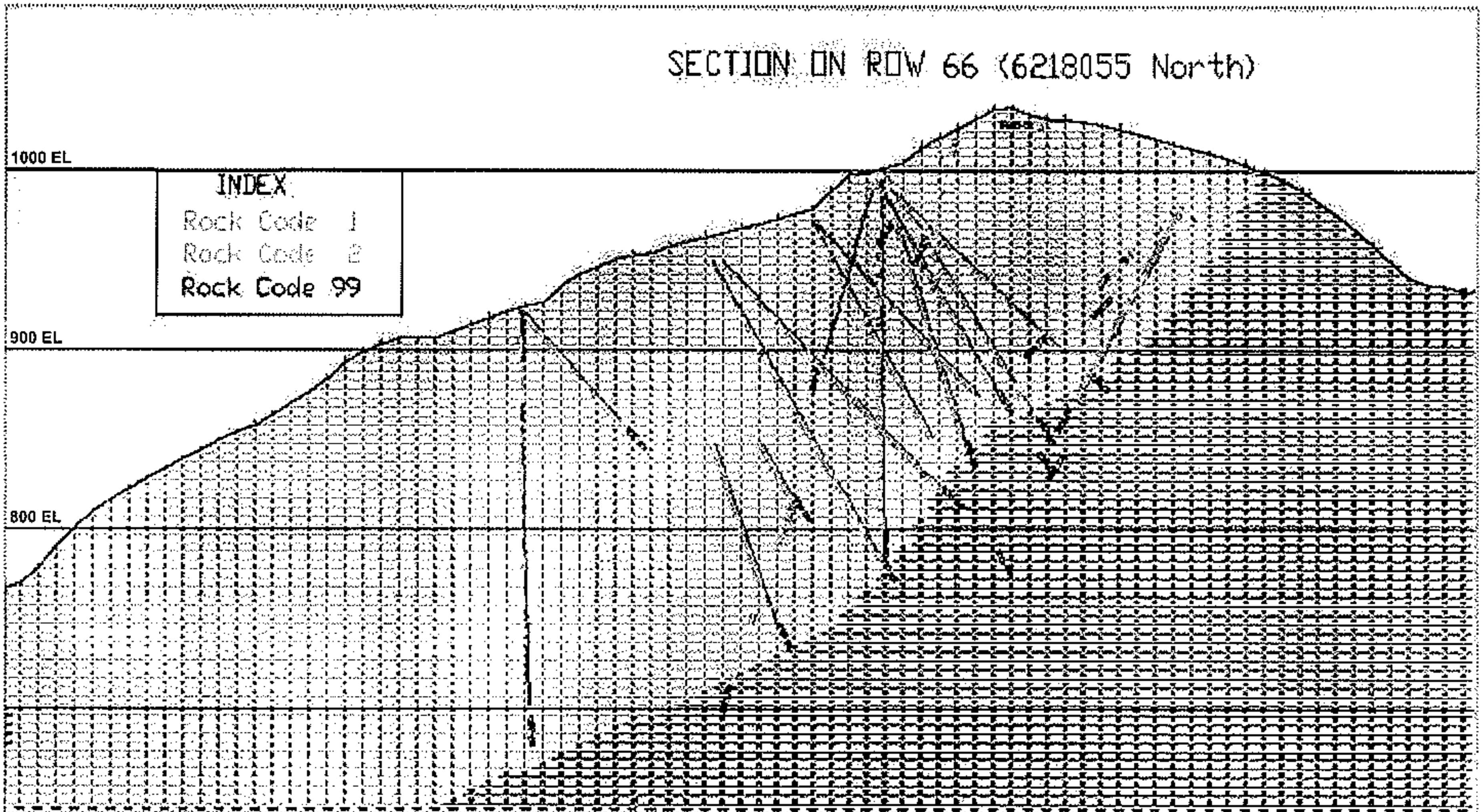
TABLE 17-2: DRILLHOLE AND TRENCH ASSAY STATISTICS



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Pinnacle Mines Ltd.***	Fig17-1.cdr	**Figure 17-1 Plan View of the Drillhole and Trench Locations**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

FIGURE 17-2 shows the block coding as a west-east section at north 6218055. Two subsurface faults are believed to act as a floor to mineralization, particularly gold. Blocks below the faults are coded as 99 (blue). Blocks above the first fault are coded as 1 (red). To the north end of the deposit there seems to be a second splay to this surface lying below the first. The blocks above the second fault, but below the first fault have been coded as 2 (cyan). Further analysis has shown there is no significant distinction in composites coded with rock codes 1 or 2. Hence the two codes have been lumped together.

Note that there is no coding for overburden. FIGURE 17-3 shows two photos of the surface of the deposit illustrating that the alluvium layer is typically extremely thin. A significant part of the Silver Coin resource essentially outcrops. Photos A and B are typical of bulldozer cuts for drill roads over the top of the deposit showing competent bedrock essentially at grass roots. The steep northern face of the deposit has zones of transported overburden where down-slope movement has resulted in local thicker accumulations of loose material. However, a review of the site in general suggests that the effects of overburden are negligible given a vertical block size of five-meters.



NUMBER OF ROCK TYPES FOUND = 3

CODE	COUNT	MINCOL	MAXCOL	MINROW	MAXROW	MINLEV	MAXLEV
1	187648	21	80	1	102	51	129
2	8329	68	95	61	98	91	127
99	316742	21	101	1	121	51	127

Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: *Pinnacle Mines Ltd.*	File Name: Fig17-2.cdr	**Figure 17-2**
	Project: Silver Coin Gold Project	Project Number: 114-311007	**Example Cross Section Illustrating the Rock Model Coding**
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

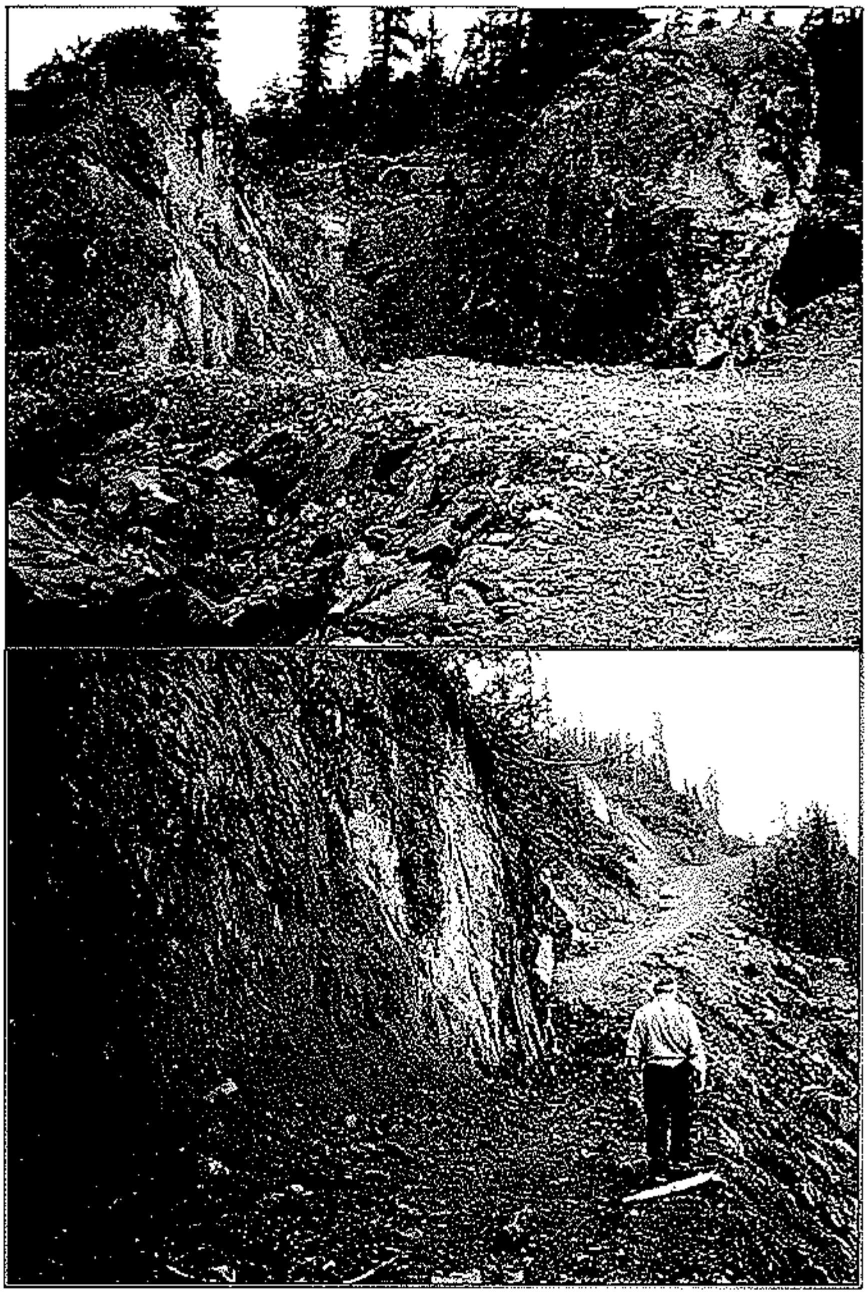


FIGURE 17-3: PHOTOGRAPHS ILLUSTRATING THE "THIN" VENEER OF OVERBURDEN WHICH WAS NOT GIVEN A DISTINCT ROCK CODE

17.4 Density

FIGURE 17-4 shows the statistical analysis of 266 density (specific gravity) measurements. The average density is 2.848. The Snowden resource used 2.85, while the Minefill Services resource used 2.86. Due to the longer upper tail, the higher value of 2.86 is probably more appropriate. In any event, a t-test indicates that there is no statistical difference between the two values. For the Tt resource, rock codes 1, 2 and 99 have been assigned the density of 2.86.

17.5 Top Cut Analysis

Top cuts of extreme grade values is a simple method to help prevent over-estimation of local blocks from a few composites. FIGURE 17-5 shows the statistical analysis of determining that 30 g/t represents a reasonable top cut for composited gold values. All composites highlighted in yellow will be reassigned a grade of 30 g/t.

FIGURE 17-6 shows the statistical analysis of determining that 130 g/t represents a reasonable top cut for composited silver values. All composites highlighted in yellow will be reassigned a grade of 130 g/t. No top cut was used for copper, lead or zinc.

17.6 Statistical Validity of Assay to Composite

FIGURES 17-7 through 17-11 show the statistics of both sample and composite data for gold, silver, copper, lead and zinc. Note that the statistics have been developed from log transformed assay data. Gold and silver show additional statistics of the composite data affected by their respective top cut.

In all cases, the statistical review shows that the compositing appears valid and did not inappropriately distort the underlying assay distribution. Gold composite data appears to follow a unimodal, somewhat triangular distribution. Silver composite data appears to follow a more classic lognormal distribution. The low composite grades for copper, lead and zinc, truncated by detection limits, apparently distort the lower end of the bell shaped lognormal distribution. This truncation effect is most noticeable for lead and zinc.

17.7 Correlation of Metals

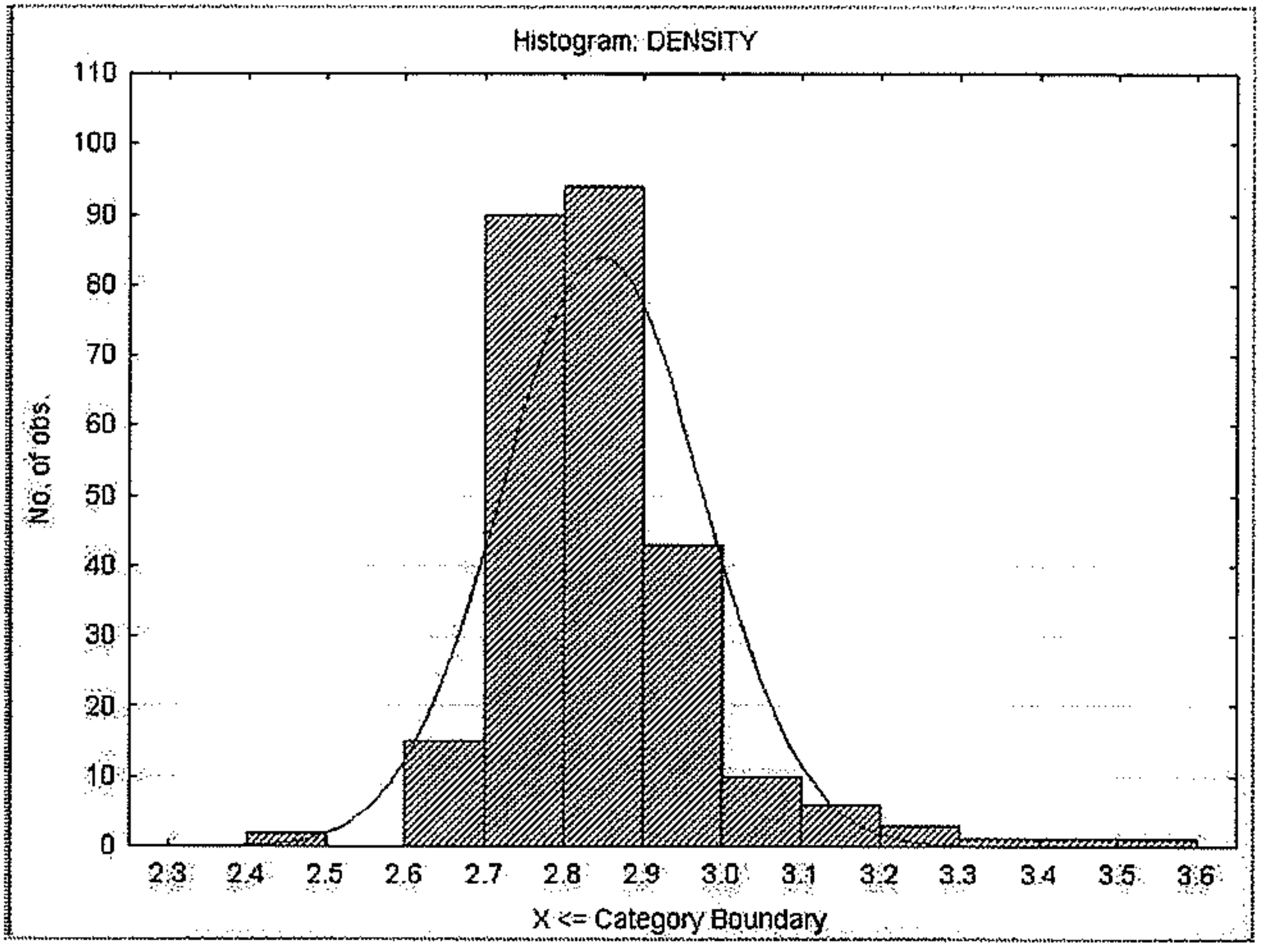
There is a positive correlation of the composited data for the other metals. The correlations with gold are:

- Au to Ag: 0.62
- Au to Cu: 0.54
- Au to Pb: 0.68
- Au to Zn: 0.67

FIGURE 17-13 shows a correlation "scatter plot" of the composited data for gold and zinc

FIGURE 17-14 shows a correlation "scatter plot" of the composited data for lead and zinc. The higher correlation of 0.866 appears to reflect the common close relationship of lead and zinc in many ore deposits.

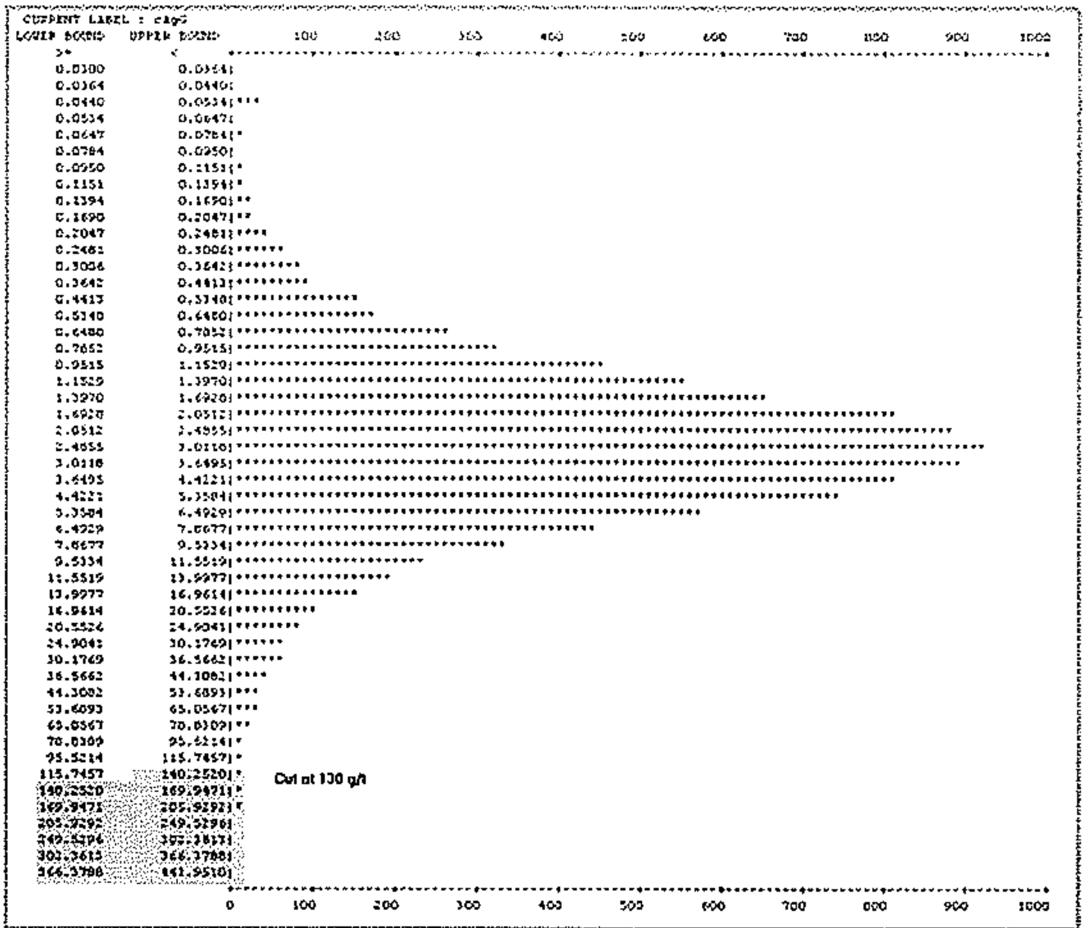
FIGURE 17-15 shows a correlation "scatter plot" of the composited data for copper and zinc.

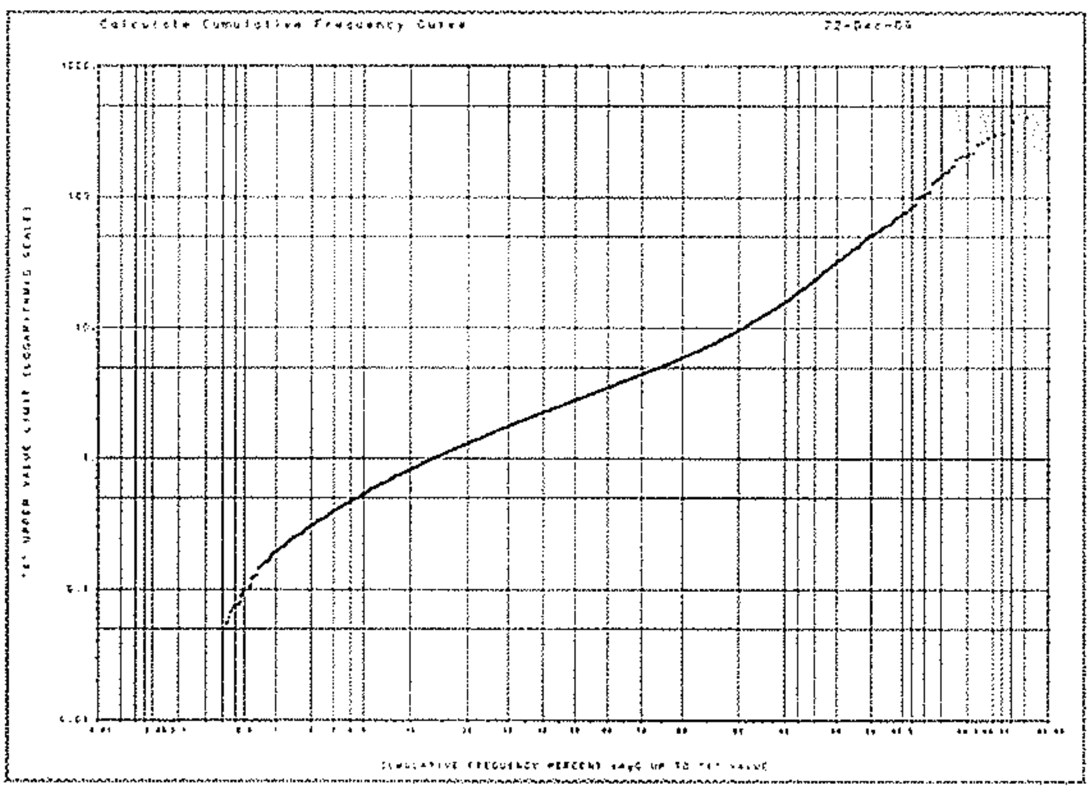


Silver Coin Density Data					
	Valid N	Mean	Minimum	Maximum	Std.Dev.
DENSITY	266	2.848	2.5	3.54	0.126

Test of means against reference constant (value) (density)							
	Mean	Std.Dv.	N	Std.Err.	Reference	t-value	df	p
DENSITY	2.848	0.1265	266	0.0077	2.860	-1.52	265	0.129

Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Pinnacle Mines Ltd.***	Fig17-4.cdr	**Figure 17-4**
	Project: Silver Coin Gold Project	Project Number: 114-311007	**Density Histogram and Statistics**
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

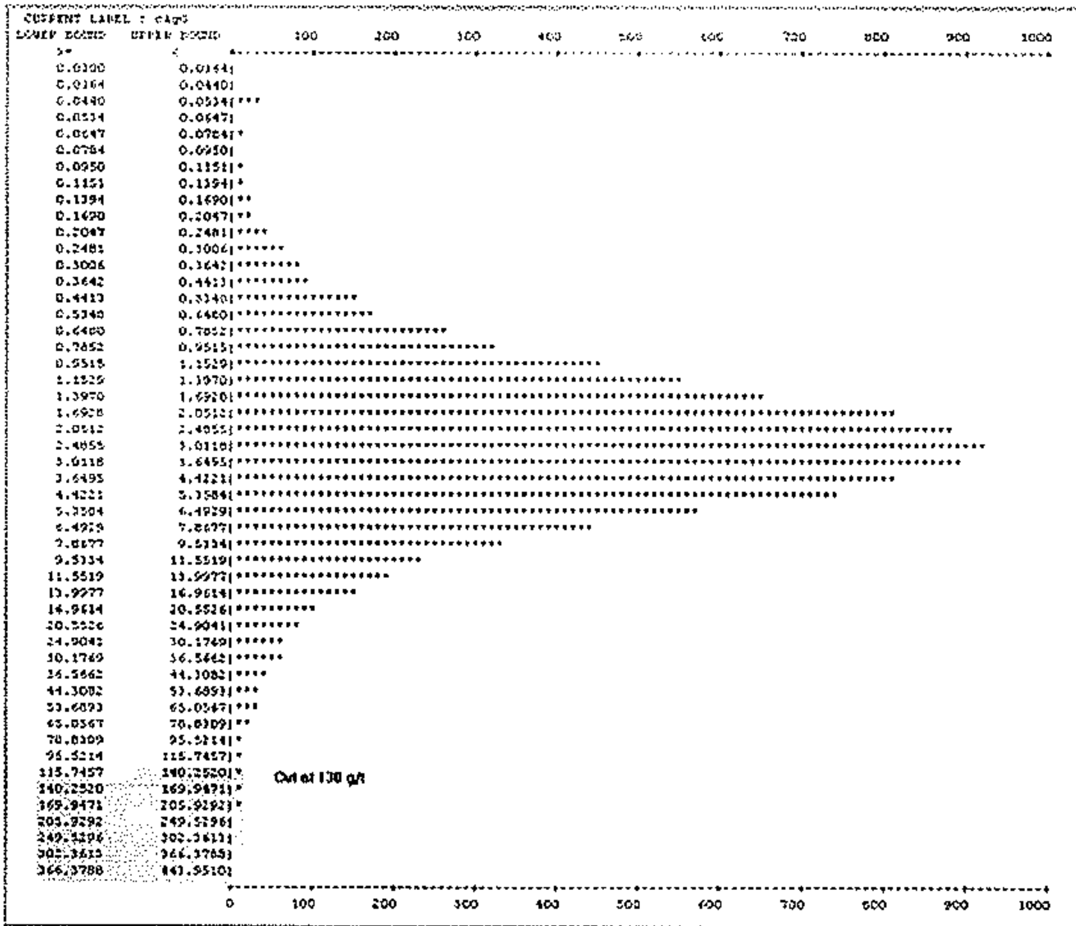


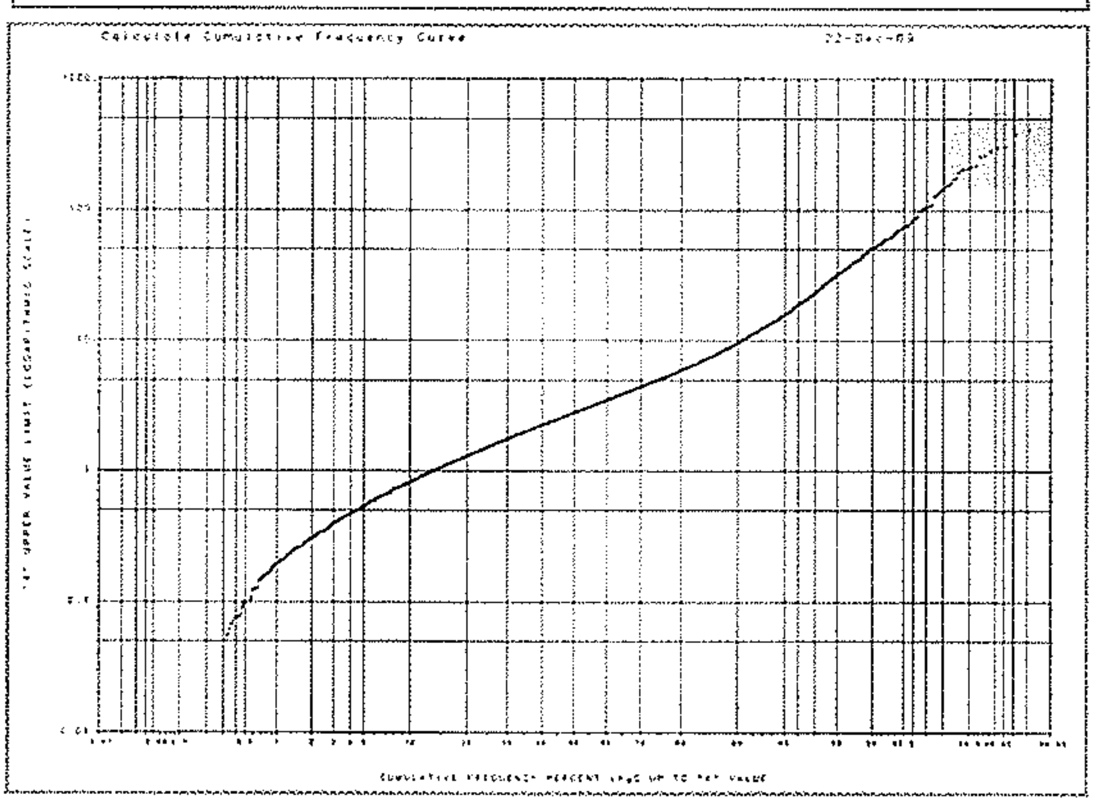


Issued by:

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Prepared for: **Pinnacle Mines Ltd.**	File Name: Fig17-6.cdr	**Figure 17-6**
Project: Silver Coin Gold Project	Project Number: 114-311007	**Silver Composites Cut at 130 g/t**
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	





Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Pinnacle Mines Ltd.***	Fig17-6.cdr	**Figure 17-6** **Silver Composites Cut at 130 g/t**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

NOTE: DH CLASS LIMITED BY
All
1 = Surface DH
2 = UG DH
3 = Trench

DATA TYPE IS SAMPLE
CURRENT LABEL : AgG

ROCK TYPE	SAMPLE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
ALL	6415	30	0	30117	0.03000	2923.0	7.2134	1158.5	34.036	4.7185	1.0186	1.5673	1.2519	6.0635	1.9478

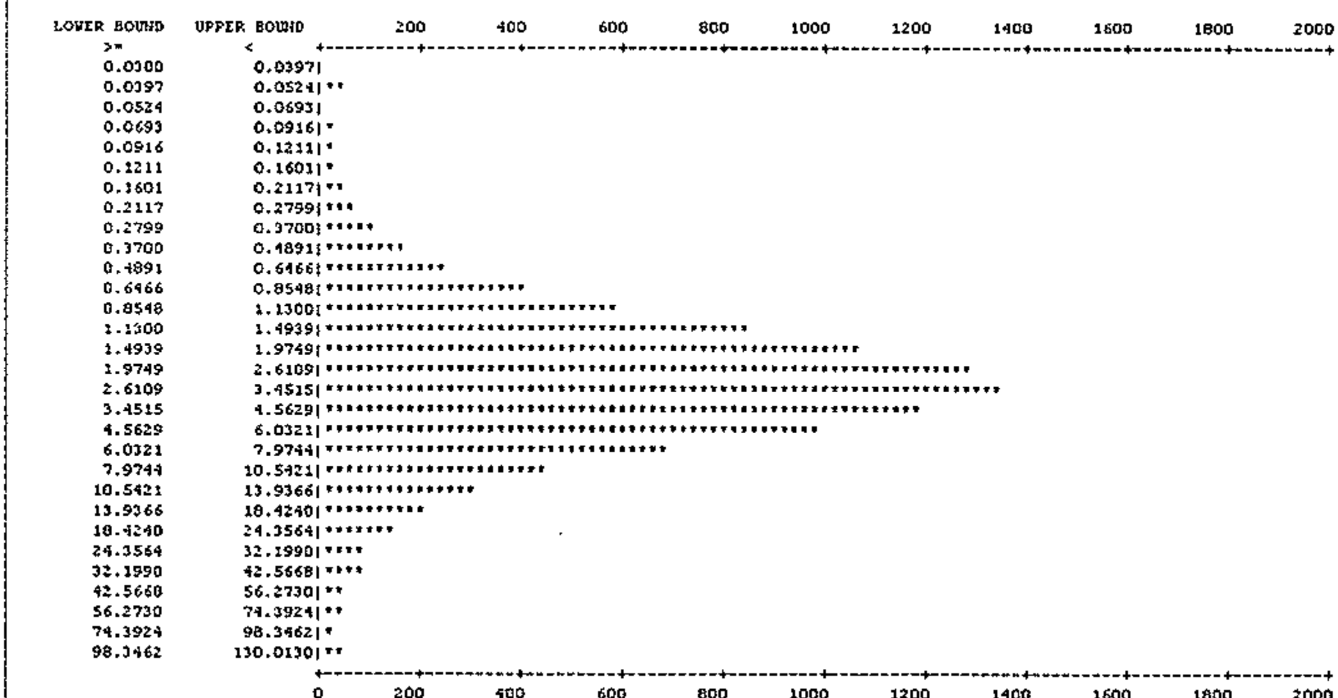
DATA TYPE IS COMPOSITE
CURRENT LABEL : cAgG

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
1	3520	0	0	9394	0.03000	443.91	5.7498	232.38	15.244	2.6512	1.0918	1.0836	1.0410	5.1223	1.3983
2	66	5	0	164	0.30616	71.213	4.0006	45.217	6.7243	1.6808	0.9233	0.7865	0.8868	3.73067	1.0935
99	372	0	0	800	0.05000	79.114	2.6451	25.880	5.0872	1.9233	0.4357	0.9633	0.9815	2.50263	1.2730
ALL	3958	5	0	10358	0.03000	443.91	5.4823	214.18	14.635	2.6695	1.0385	1.1005	1.0490	4.8974	1.4162

DATA TYPE IS COMPOSITE
CURRENT LABEL : xAgG

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
1	3520	0	0	9394	0.03000	130.00	5.4764	112.98	10.629	1.9409	1.0904	1.0723	1.0355	5.0865	1.3864
2	66	5	0	164	0.30616	71.213	4.0006	45.217	6.7243	1.6808	0.9233	0.7865	0.8868	3.73067	1.0935
99	372	0	0	800	0.05000	79.114	2.6451	25.880	5.0872	1.9233	0.4357	0.9633	0.9815	2.50263	1.2730
ALL	3958	5	0	10358	0.03000	130.00	5.2344	105.77	10.284	1.9648	1.0372	1.0901	1.0441	4.8660	1.4052

25 Ag composites were above 130 gpt



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-8.cdr	**Figure 17-8 Silver Assays Statistics**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

NOTE: DH CLASS LIMITED BY
All
1 = Surface DH
2 = UG DH
3 = Trench

DATA TYPE IS SAMPLE
CURRENT LABEL : AgG

ROCK TYPE	SAMPLE COUNT: MISSING	SAMPLE COUNT: BELOW LIMITS	SAMPLE COUNT: ABOVE LIMITS	SAMPLE COUNT: INSIDE LIMITS	UNTRANSFORMED STATISTICS: MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS: LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED: MEAN	COEF. OF VAR.
ALL	6415	30	0	30117	0.03000	2923.0	7.2134	1158.5	34.036	4.7185	1.0186	1.5673	1.2519	6.0635	1.9478

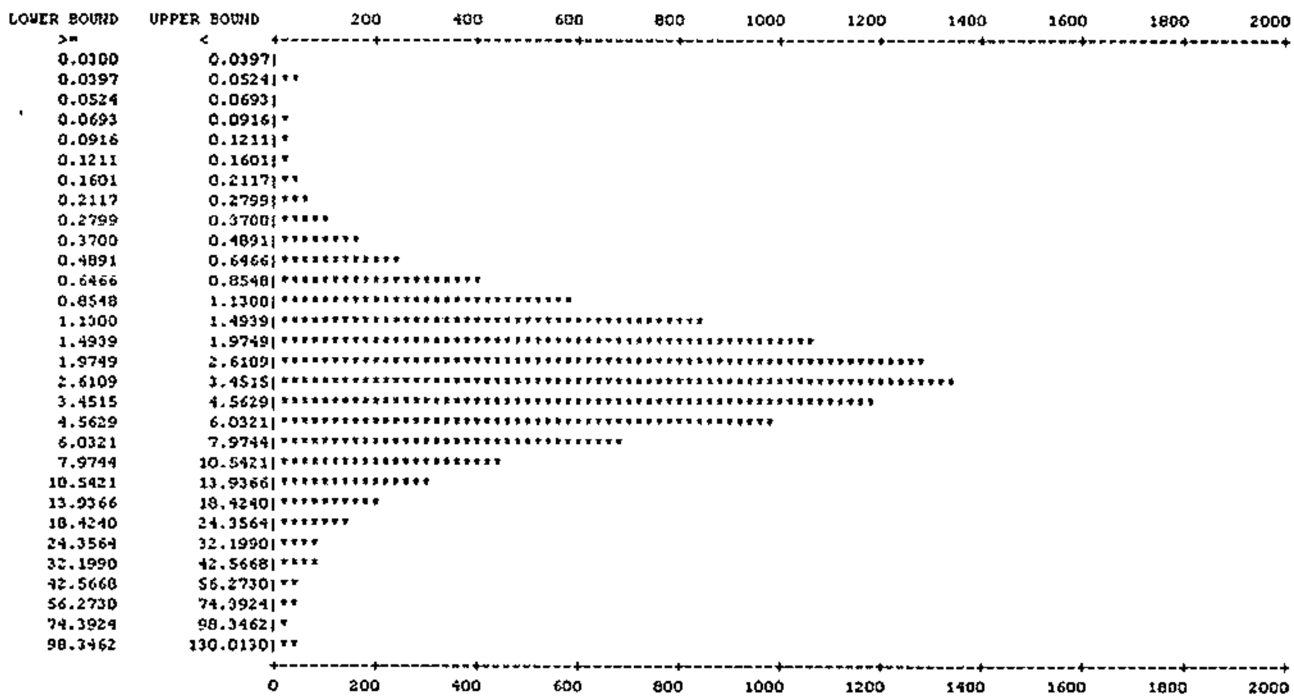
DATA TYPE IS COMPOSITE
CURRENT LABEL : cAgG

ROCK TYPE	COMPOSITE COUNT: MISSING	COMPOSITE COUNT: BELOW LIMITS	COMPOSITE COUNT: ABOVE LIMITS	COMPOSITE COUNT: INSIDE LIMITS	UNTRANSFORMED STATISTICS: MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS: LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED: MEAN	COEF. OF VAR.
1	3520	0	0	9394	0.03000	443.91	5.7498	232.38	15.244	2.6512	1.0918	1.0836	1.0410	5.1223	1.3983
2	66	5	0	164	0.30616	71.213	4.0006	45.217	6.7243	1.6808	0.9233	0.7865	0.8868	3.73067	1.0935
99	372	0	0	800	0.05000	79.114	2.6451	25.880	5.0872	1.9233	0.4357	0.9633	0.9815	2.50263	1.2730
ALL	3958	5	0	10358	0.03000	443.91	5.4823	214.18	14.635	2.6695	1.0385	1.1005	1.0490	4.8974	1.4162

DATA TYPE IS COMPOSITE
CURRENT LABEL : xAgG

ROCK TYPE	COMPOSITE COUNT: MISSING	COMPOSITE COUNT: BELOW LIMITS	COMPOSITE COUNT: ABOVE LIMITS	COMPOSITE COUNT: INSIDE LIMITS	UNTRANSFORMED STATISTICS: MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS: LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED: MEAN	COEF. OF VAR.
1	3520	0	0	9394	0.03000	130.00	5.4764	112.98	10.629	1.9409	1.0904	1.0723	1.0355	5.0865	1.3864
2	66	5	0	164	0.30616	71.213	4.0006	45.217	6.7243	1.6808	0.9233	0.7865	0.8868	3.73067	1.0935
99	372	0	0	800	0.05000	79.114	2.6451	25.880	5.0872	1.9233	0.4357	0.9633	0.9815	2.50263	1.2730
ALL	3958	5	0	10358	0.03000	130.00	5.2344	105.77	10.284	1.9648	1.0372	1.0901	1.0441	4.8660	1.4052

25 kg composites were above 130 gpt



Issued by:

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Prepared for:	File Name:	
Pinnacle Mines Ltd.	Fig17-8.cdr	
Project: Silver Coin Gold Project	Project Number: 114-311007	
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

Figure 17-8
Silver Assays Statistics

NOTE: DH CLASS LIMITED BY
All
1 = Surface DH
2 = UG DH
3 = Trench

DATA TYPE IS SAMPLE
CURRENT LABEL : Cu%

ROCK TYPE	SAMPLE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
ALL	12707	28	0	23827	0.000300	66.300	0.02113	0.20148	0.44887	21.2410	-5.5167	1.4529	1.2054	0.0083	1.8099

DATA TYPE IS COMPOSITE
CURRENT LABEL : cCu%

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
1	4886	0	0	8028	0.000320	3.2795	0.01204	0.00560	0.07485	6.2192	-5.5292	1.0685	1.0337	0.0068	1.3824
2	128	5	0	102	0.000400	0.02860	0.00497	0.000018	0.00420	0.8440	-5.6208	0.7055	0.8399	0.0052	1.0123
99	416	0	0	756	0.000500	0.38860	0.00694	0.000333	0.01826	2.6302	-5.5224	0.8833	0.9399	0.0062	1.1912
ALL	5430	5	0	8886	0.000320	3.2795	0.01152	0.00509	0.07137	6.1942	-5.5297	1.0488	1.0241	0.0067	1.3617

```
LOWER BOUND   UPPER BOUND        200       400       600       800      1000      1200      1400      1600      1800      2000
   >=             <     +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
 0.0003         0.0004|
 0.0004         0.0006|***********
 0.0006         0.0008|*******
 0.0008         0.0011|*****************
 0.0011         0.0015|********************
 0.0015         0.0020|**********************************************
 0.0020         0.0028|***********************************************************
 0.0028         0.0038|****************************************************************************
 0.0038         0.0051|******************************************************************
 0.0051         0.0070|************************************************
 0.0070         0.0095|*********************************
 0.0095         0.0129|*********************
 0.0129         0.0175|*************
 0.0175         0.0238|*******
 0.0238         0.0324|*****
 0.0324         0.0441|****
 0.0441         0.0600|***
 0.0600         0.0816|***
 0.0816         0.1110|*
 0.1110         0.1510|**
 0.1510         0.2054|*
 0.2054         0.2795|*
 0.2795         0.3802|*
 0.3802         0.5173|
 0.5173         0.7037|
 0.7037         0.9574|
 0.9574         1.3025|
 1.3025         1.7720|
 1.7720         2.4108|
 2.4108         3.2798|
                        +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                        0        200       400       600       800      1000      1200      1400      1600      1800      2000
```

Issued by:

TETRATECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Prepared for:	File Name:	
Pinnacle Mines Ltd.	Fig17-9.cdr	
Project: Silver Coin Gold Project	Project Number: 114-311007	Figure 17-9 Copper Assays Statistics
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

Figure 17-9
Copper Assays Statistics

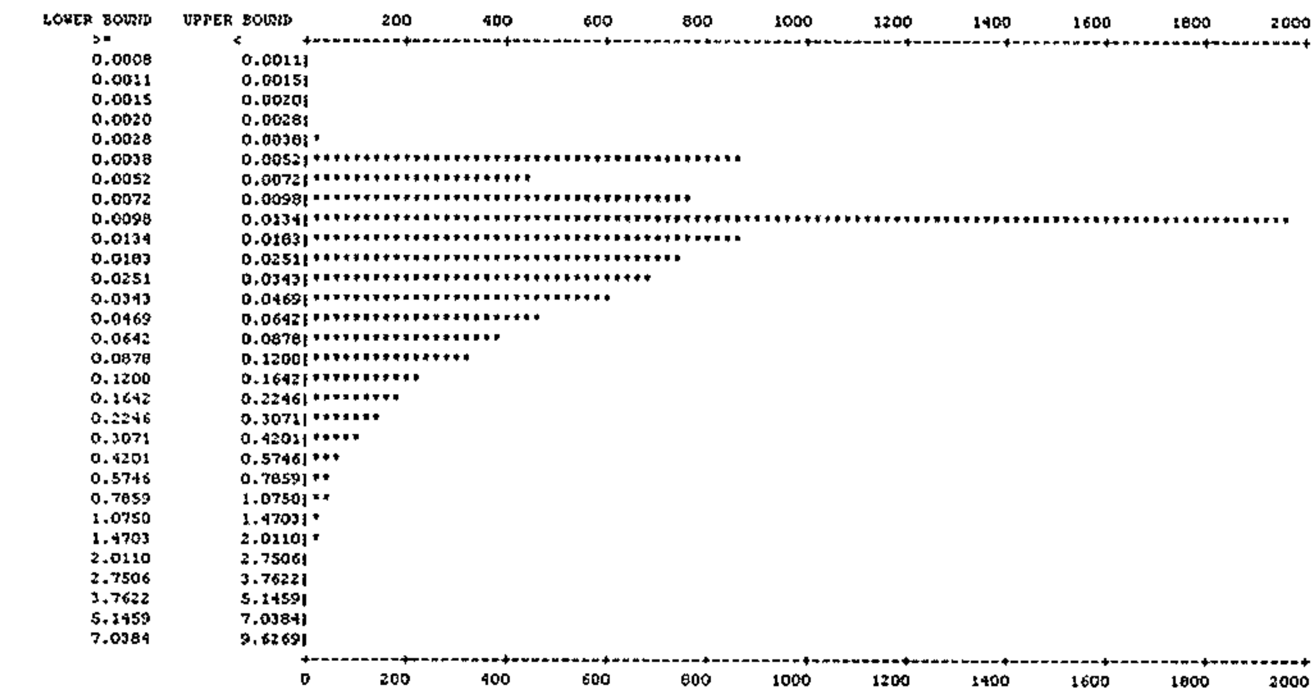
NOTE: DH CLASS LIMITED BY
All
1 = Surface DH
2 = UG DH
3 = Trench

DATA TYPE IS SAMPLE
CURRENT LABEL : Pb%

ROCK TYPE	SAMPLE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
ALL	12528	27	0	24007	0.000400	27.500	0.08704	0.21767	0.46676	5.3626	-3.8624	1.7972	1.3406	0.0516	2.2433

DATA TYPE IS COMPOSITE
CURRENT LABEL : cPb%

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
1	4849	0	0	8065	0.000800	9.6259	0.06397	0.04579	0.21399	3.3451	-3.7877	1.4475	1.2031	0.0467	1.8035
2	128	5	0	102	0.00340	0.50000	0.03128	0.00323	0.05684	1.8170	-4.0369	0.8611	0.9280	0.0272	1.1687
99	416	0	0	756	0.00500	0.78567	0.01924	0.00230	0.04801	2.4956	-4.5015	0.6622	0.8138	0.0154	0.9691
ALL	5393	5	0	8923	0.000800	9.6259	0.05981	0.04178	0.20441	3.4178	-3.8510	1.4140	1.1891	0.0431	1.7642



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-10.cdr	**Figure 17-10** **Lead Assays Statistics**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

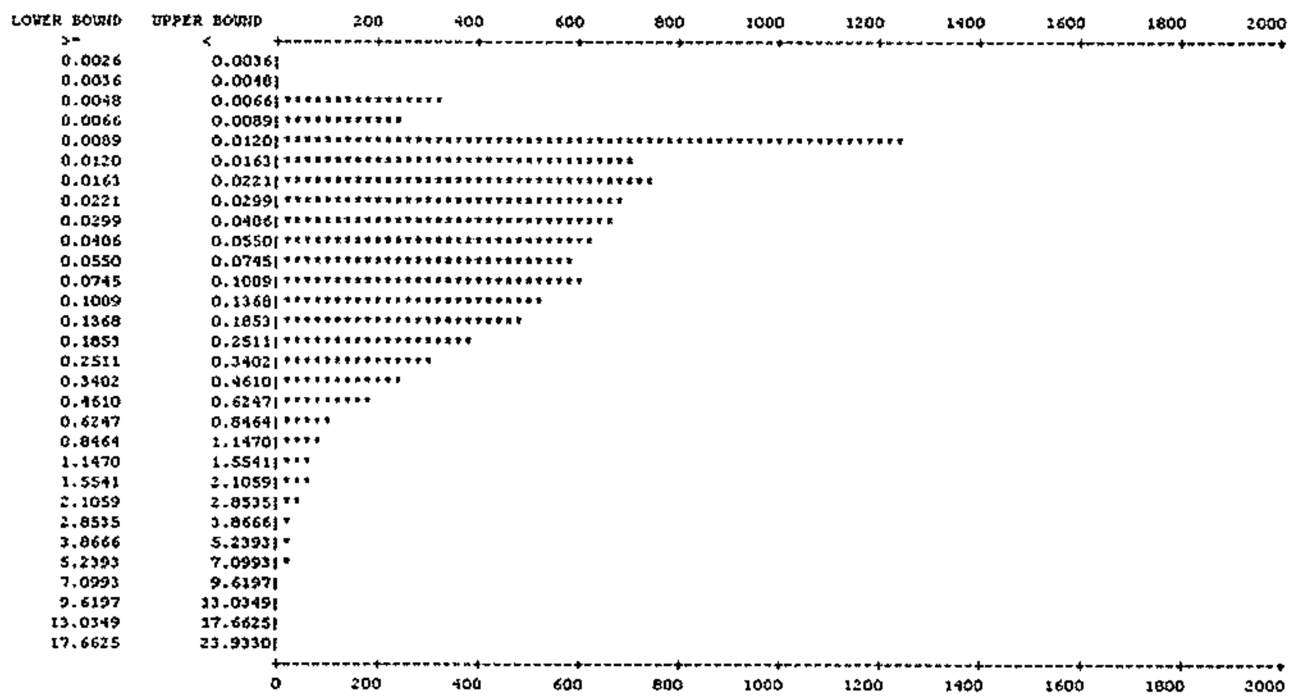
NOTE: DH CLASS LIMITED BY
All
1 - Surface DH
2 - UG DH
3 - Trench

DATA TYPE IS SAMPLE
CURRENT LABEL : Zn%

ROCK TYPE	SAMPLE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
ALL	12682	28	0	23852	0.00100	65.000	0.25467	1.5588	1.2485	4.9024	-3.1028	2.5537	1.5980	0.1611	3.4431

DATA TYPE IS COMPOSITE
CURRENT LABEL : cZn%

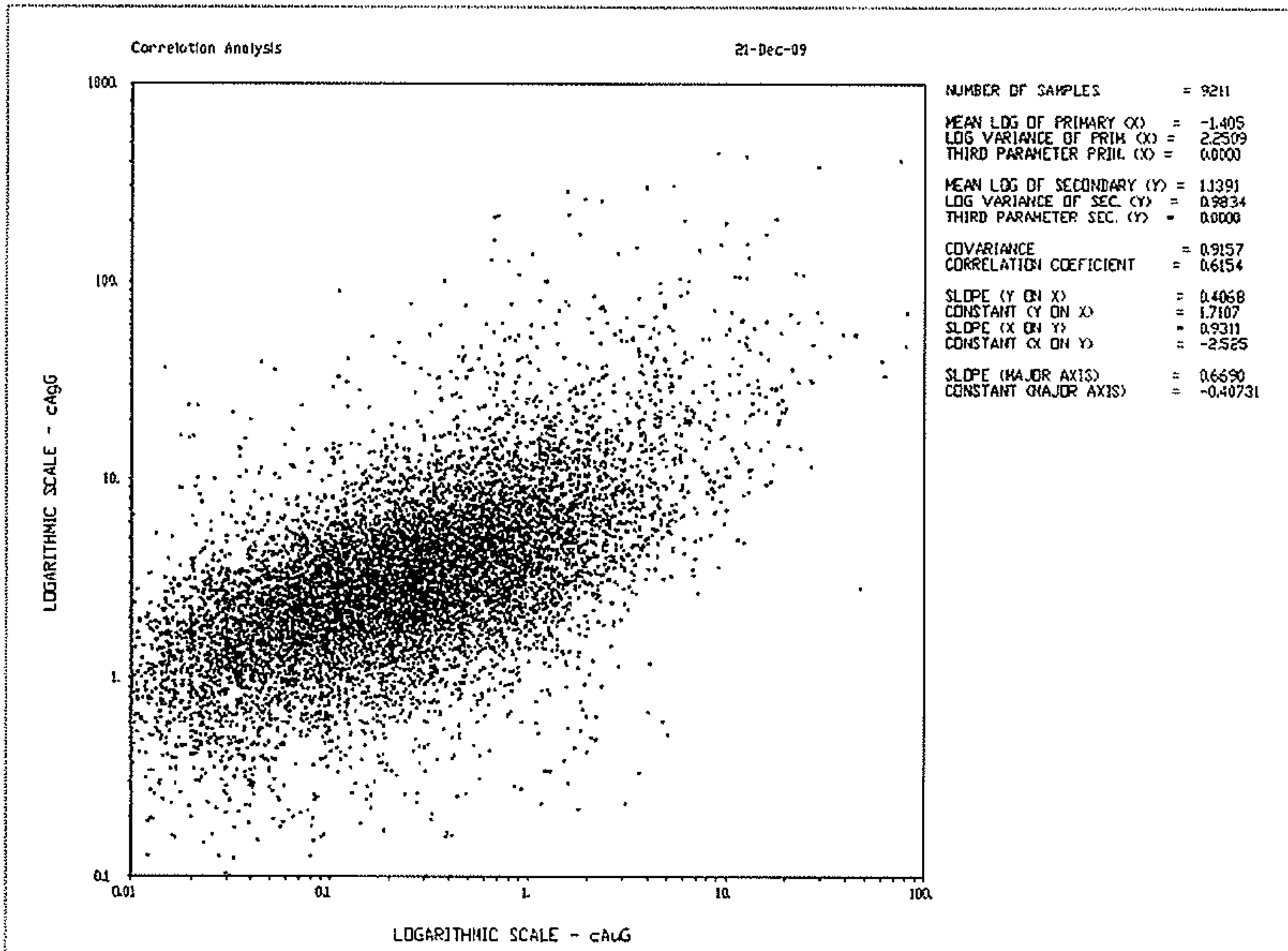
ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
1	4875	0	0	8039	0.00263	23.931	0.18522	0.47808	0.69143	3.7330	-3.0104	2.0463	1.4305	0.1371	2.5960
2	128	5	0	102	0.00578	0.85400	0.06838	0.01151	0.10731	1.5693	-3.3078	1.1308	1.0634	0.0644	1.4485
99	416	0	0	756	0.00500	5.9982	0.06665	0.07576	0.27524	4.1298	-3.6159	1.2843	1.1332	0.0511	1.6162
ALL	5419	5	0	8897	0.00263	23.931	0.17381	0.43975	0.66314	3.8154	-3.0653	2.0002	1.4143	0.1268	2.5279



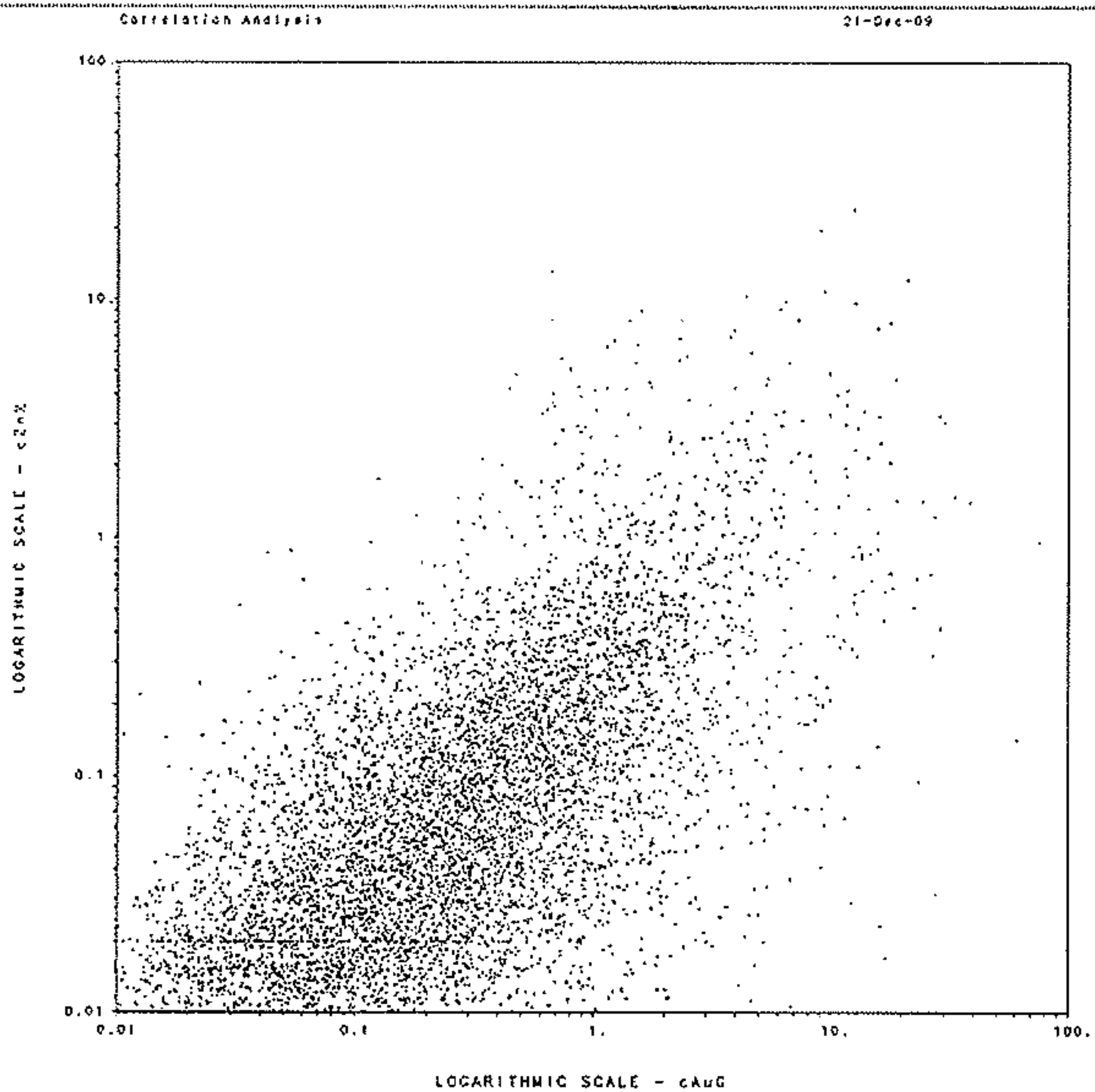
Issued by: TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Prepared for:	File Name:	
Pinnacle Mines Ltd.	Fig17-11.cdr	Figure 17-11 Zinc Assays Statistics
Project: Silver Coin Gold Project	Project Number: 114-311007	
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

Figure 17-11
Zinc Assays Statistics



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-12.cdr	**Figure 17-12** **Scatter Plot showing Correlation of Gold and Silver Composites**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	



NUMBER OF SAMPLES	=	7344
MEAN LOG OF PRIMARY (X)	=	-1.506
LOG VARIANCE OF PRIM. (X)	=	2.1980
THIRD PARAMETER PRIM. (X)	=	0.0000
MEAN LOG OF SECONDARY (Y)	=	-2.377
LOG VARIANCE OF SEC. (Y)	=	1.8499
THIRD PARAMETER SEC. (Y)	=	0.0000
COVARIANCE	=	1.5453
CORRELATION COEFFICIENT	=	0.6577
SLOPE (Y ON X)	=	0.6121
CONSTANT (Y ON X)	=	-1.901
SLOPE (X ON Y)	=	0.7277
CONSTANT (X ON Y)	=	3.7154
SLOPE (MAJOR AXIS)	=	0.6697
CONSTANT (MAJOR AXIS)	=	0.90733

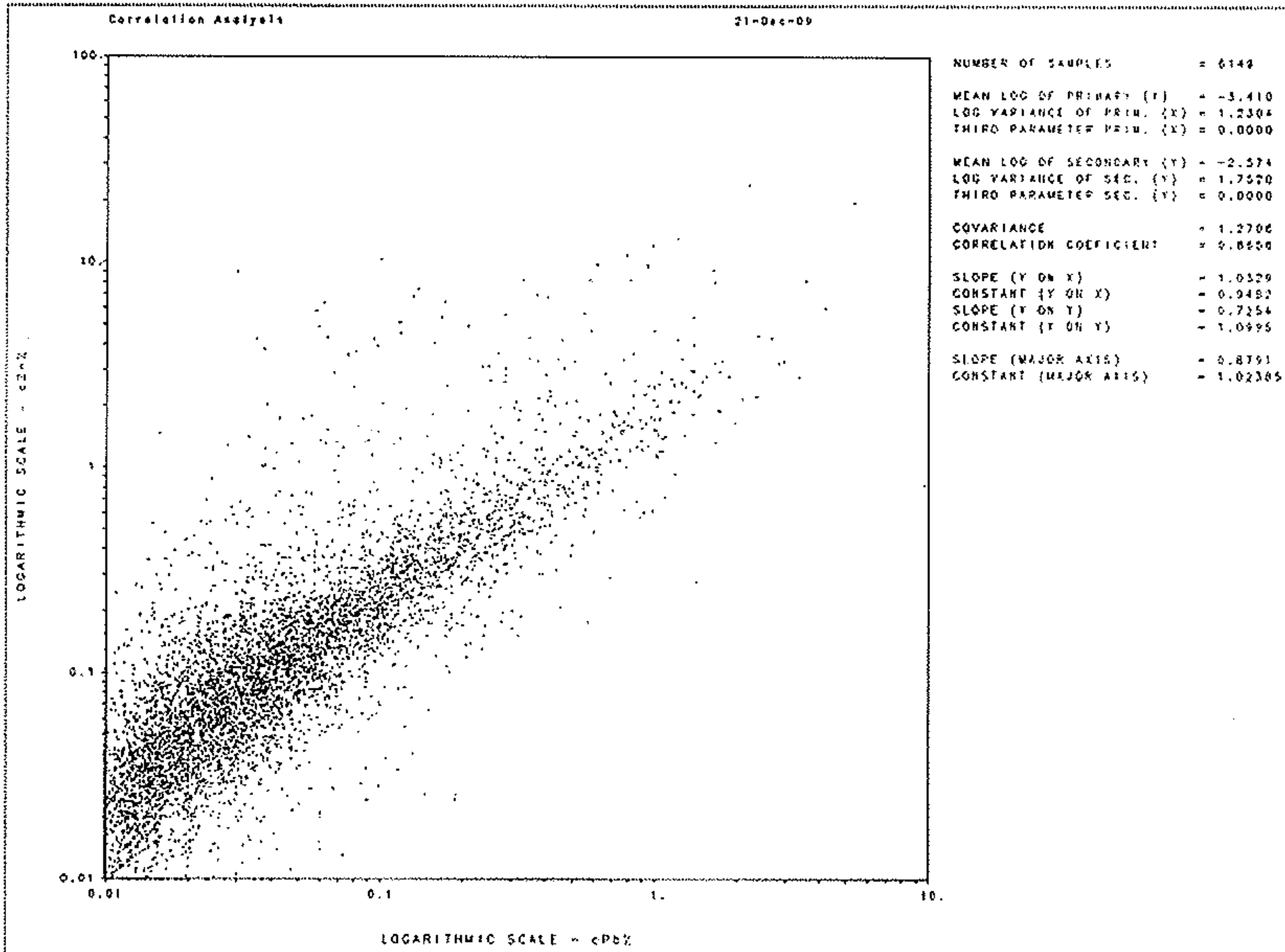
Issued by:

TETRA TECH

350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Prepared for:	*Pinnacle Mines Ltd.*	File Name:	Fig17-13.cdr
Project:	Silver Coin Gold Project	Project Number:	114-311007
Project Location:	Stewart, British Columbia	Date of Issue:	12/24/2009

Figure 17-13
Scatter Plot showing Correlation of Gold and Zinc Composites

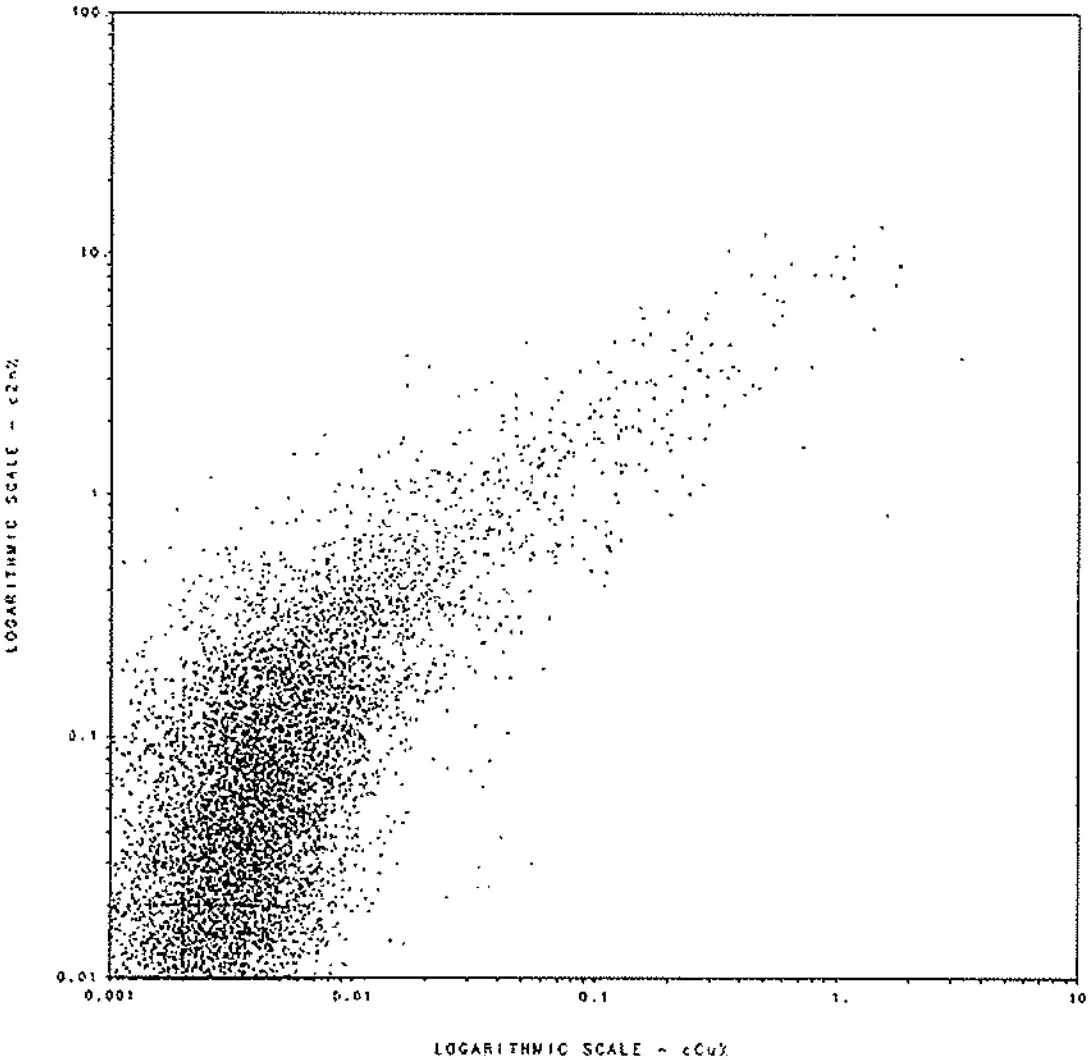


Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-13.cdr	
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

Figure 17-14
Scatter Plot showing Correlation of Lead and Zinc Composites

Correlation Analysis

21-Dec-09



NUMBER OF SAMPLES	= 7237
MEAN LOG OF PRIMARY (X)	= -5.405
LOG VARIANCE OF PRIN. (X)	= 0.9099
THIRD PARAMETER PRIN. (X)	= 0.0000
MEAN LOG OF SECONDARY (Y)	= -2.941
LOG VARIANCE OF SEC. (Y)	= 1.8147
THIRD PARAMETER SEC. (Y)	= 0.0000
COVARIANCE	= 0.9222
CORRELATION COEFFICIENT	= 0.7177
SLOPE (Y ON X)	= 1.0135
CONSTANT (Y ON X)	= 2.6375
SLOPE (X ON Y)	= 0.5082
CONSTANT (X ON Y)	= -0.250
SLOPE (MAJOR AXIS)	= 0.7609
CONSTANT (MAJOR AXIS)	= 1.19376

Issued by:	Prepared for:	File Name:
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-13.cdr
	Project: Silver Coin Gold Project	Project Number: 114-311007
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 17-15
Scatter Plot showing Correlation of Copper and Zinc Composites

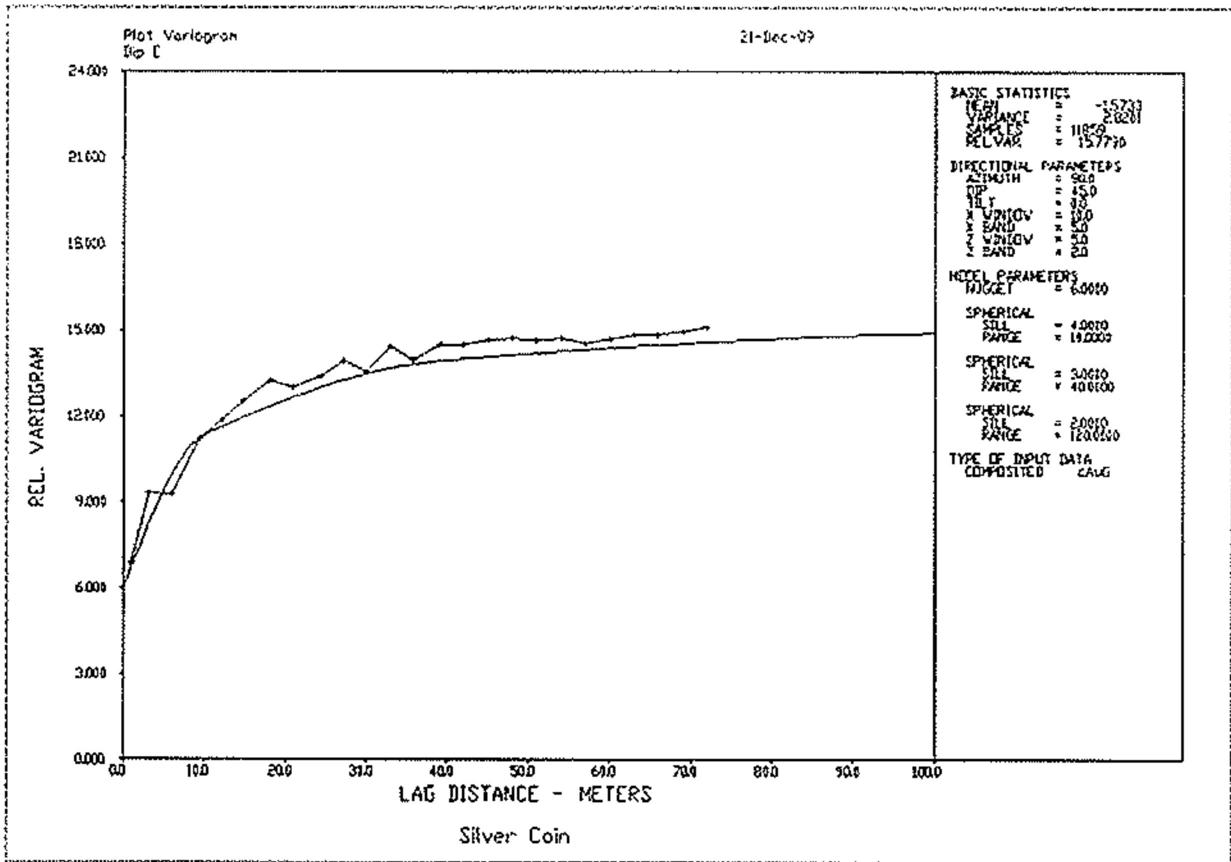
17.8 Variography

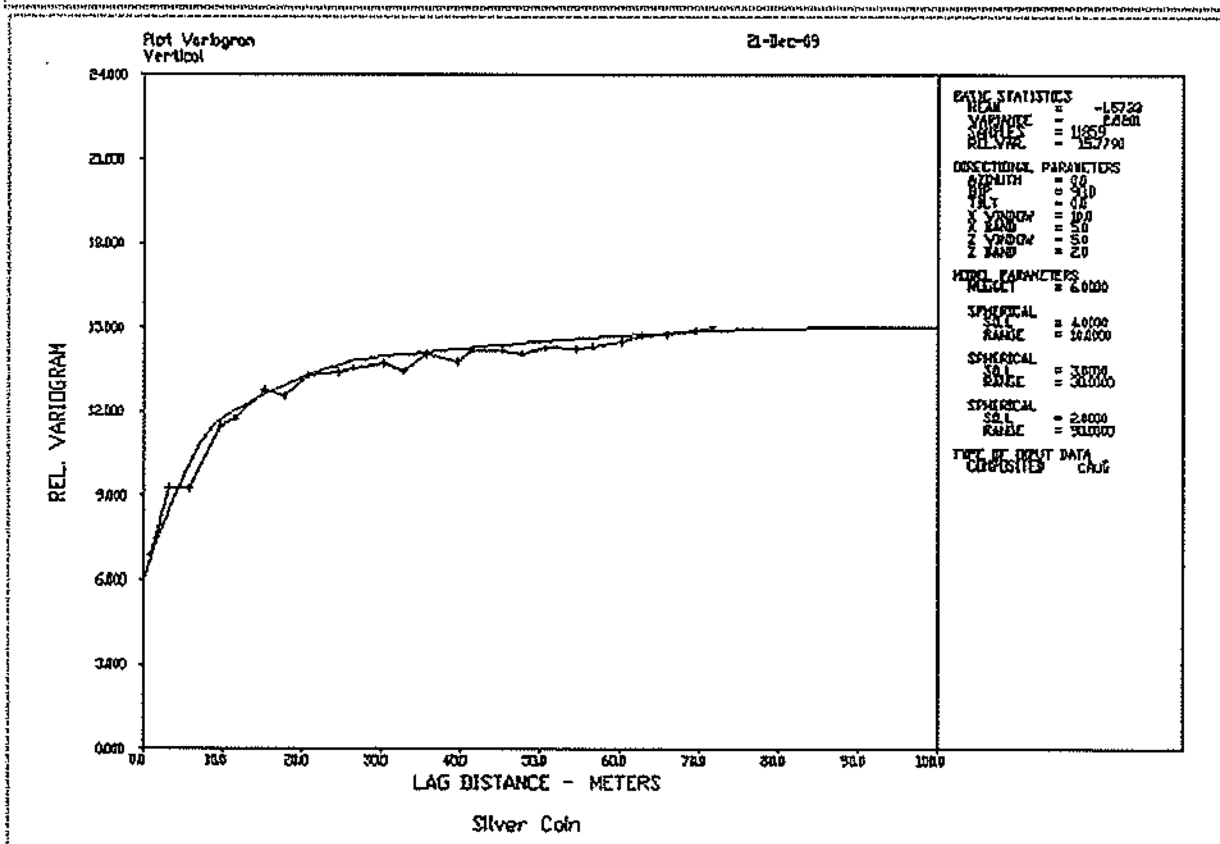
FIGURE 17-16 shows two directional relative variograms for composited gold. The nugget effect is high, with a value of 6, which is 40% of the ultimate sill of 15. The top variogram explores the spatial relationship of composite data looking east (Azimuth 90) with a 45 degree dip. The bottom variogram analyzes the vertical direction. There are some differences, but in large part they are similar in structure. Both models are based on three nested spherical models listed in the figure.

Silver (FIGURE 17-17) and zinc (FIGURE 17-18) show spatial structures that are similar to gold. Lead and copper (no figures) also show similarity to gold's spatial structure.

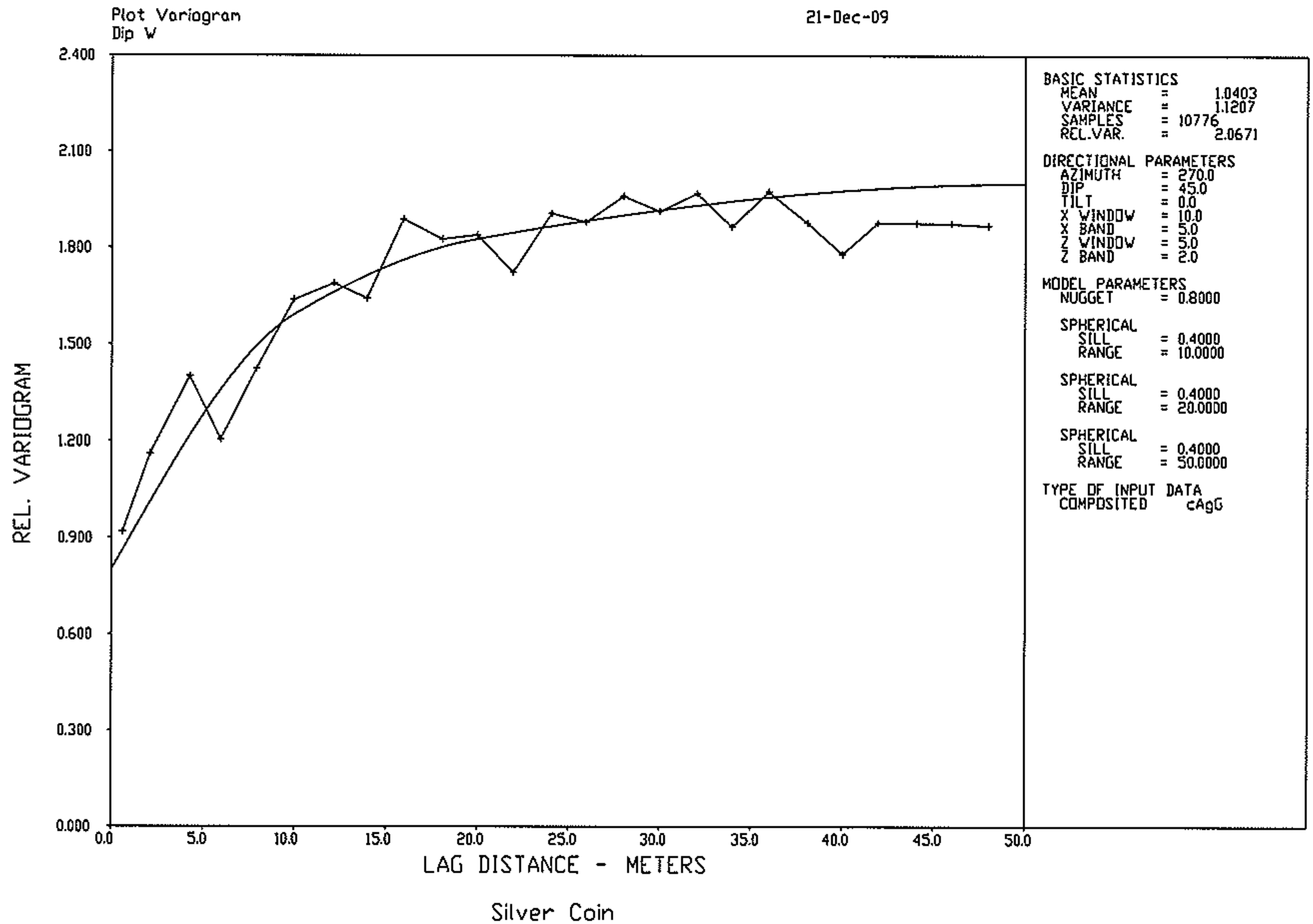
17.9 Jackknife Study

FIGURE 17-19 shows the results of jackknifing gold values using the gold variogram model discussed above and a maximum search window of 11 meters. Jackknifing (also called model validation), sequentially removes each composite and then uses surrounding data and the variogram parameters to estimate its value. The estimate (est) and the original value are then compared. FIGURE 17-19A is a histogram of the difference. The x-axis is in real gold values. Note that the histogram shows a bias where it is not centered at zero but at -0.018. FIGURE 17-19B shows the histogram of the original value and the estimate plotted side-by-side. There is a bias of a log mean of -1.0958 for the original value versus -0.6992. Taking the exponential, these equate to real values of 0.334 g Au/t for the original data versus 0.497 g Au/t. This positive bias is an expected aspect of kriging when the average of the estimated blocks is below the overall mean of the deposit (e.g. 0.84 g Au/t).



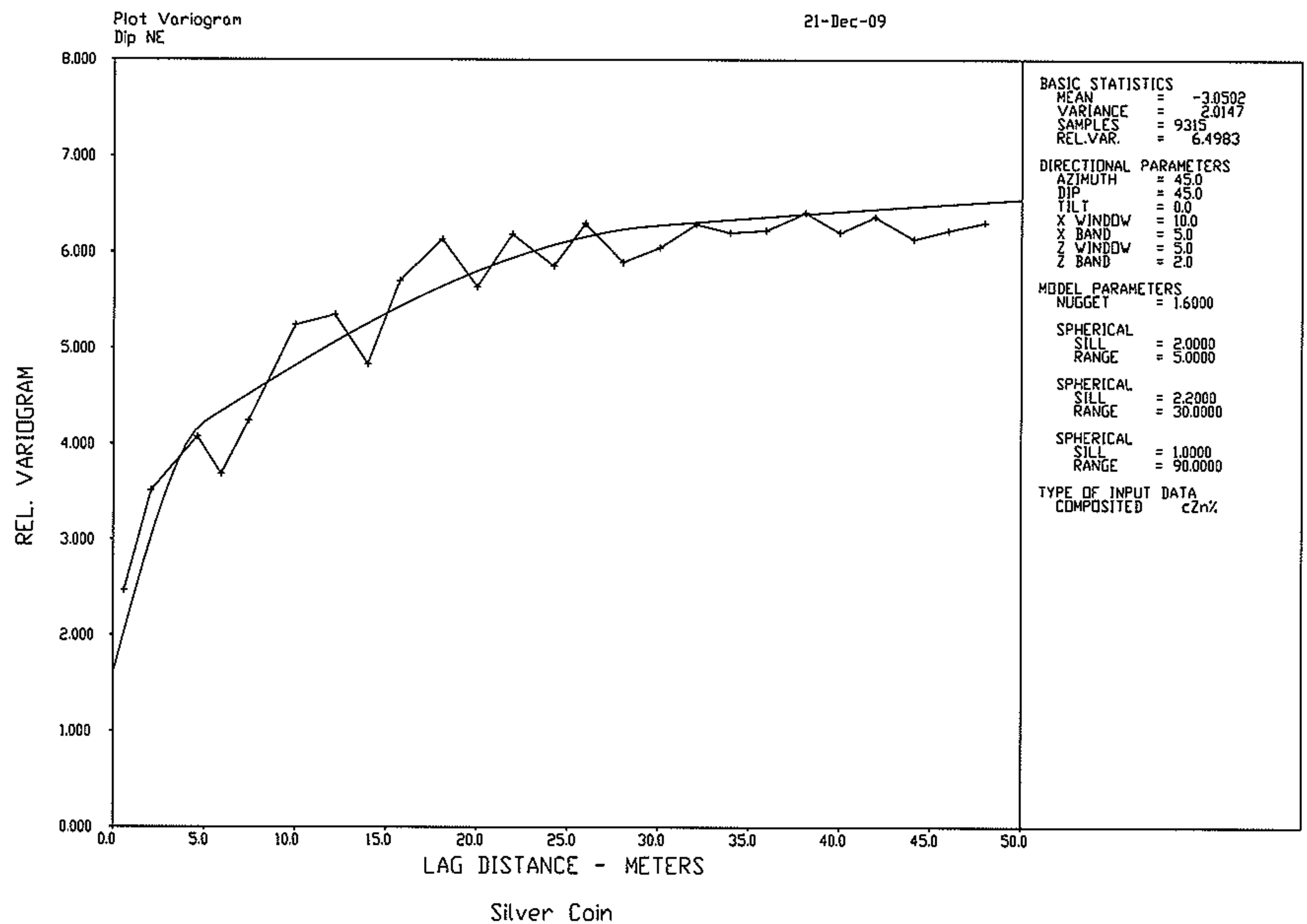


Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Pinnacle Mines Ltd.***	Fig17-16.cdr	**Figure 17-16 Selected Relative Variograms Gold Composites**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

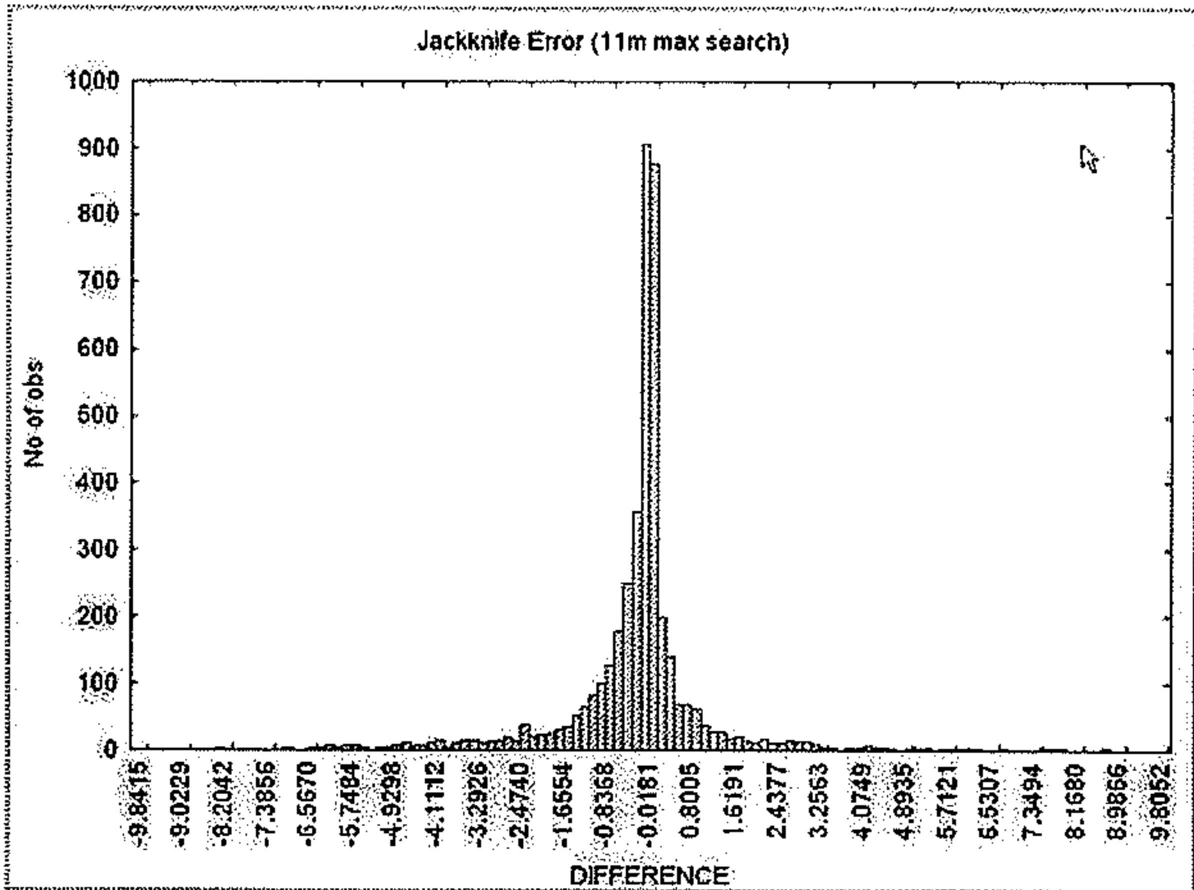


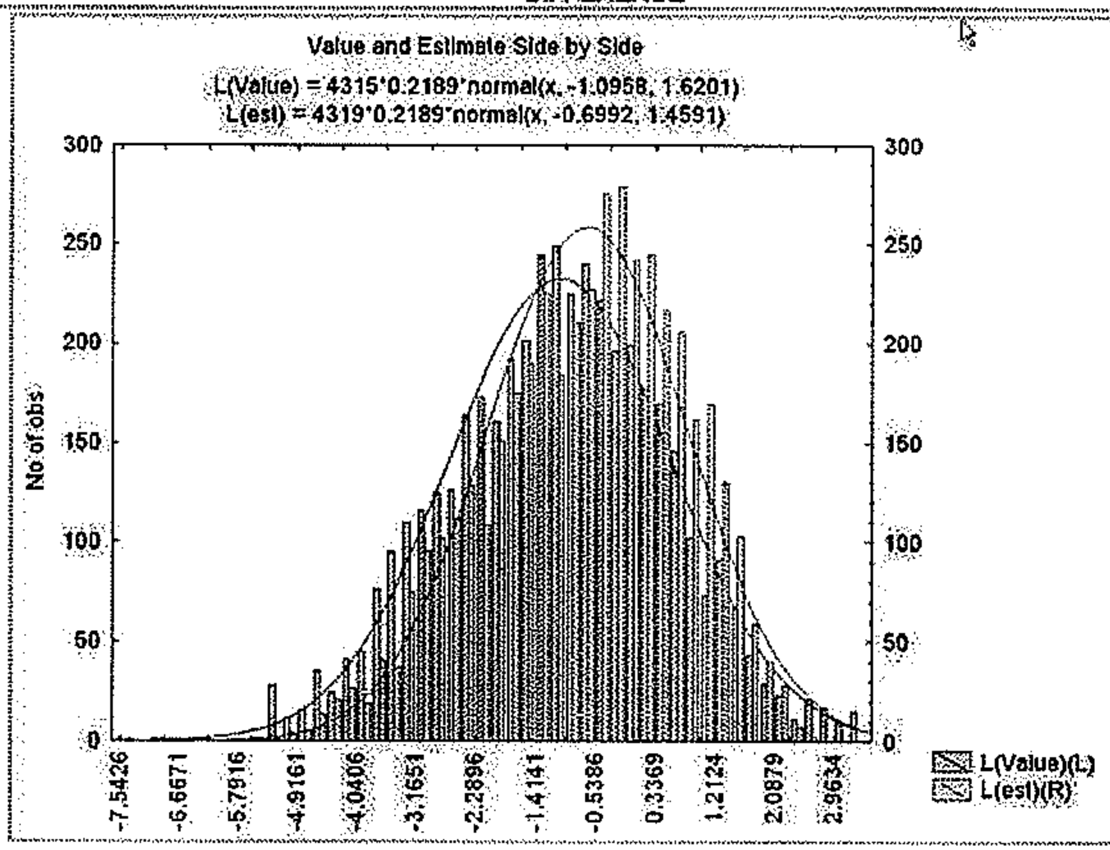
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-17.dwg	Figure 17-17 Variogram of Silver Composites (West, Dipping 45 degrees)
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

Figure 17-17
Variogram of Silver Composites (West, Dipping 45 degrees)



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-18.dwg	Figure 17-18 Variogram of Zinc Composites (NE, Dipping 45 degrees)
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	





Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-19.cdr	**Figure 17-19** **Jackknife Plots showing Gold Composites**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

17.10 Kriging Results

Kriging was done using ordinary kriging. The parameters used for the kriging analysis are summarized in FIGURE 17-20.

Matching Codes			Anisotropy				MIF Search Ranges					Variogram Parameters						
xAu (Gold Value Composited to 5m, cut at 30 g/t)																		
Composite Codes	Block Codes	Zone Name	Axis	Anisotropy Axis Length (m)	Anisotropy Rotation	Type[3]	Resource Class[5]	Resource Code[2]	Maximum Search Range	Number Closest Pts/Max Pts Single Drillhole	Min Pts Required to Estimate	Rotation	Length	Nugget[1]	Nested	Model Type[4]	Sill[1]	Range (m)
1&2	1&2	Above Faults 1&2	Primary	35	90	Az	M	1	11	15/99	2	90	100	6.0	1	Sph	4	10
			Second	35	45	Dip	I	2&3	20	15/99	2	45	100		2	Sph	3	40
			Tertiary	15	0	Tilt	F	4&5&6	50	15/99	2	0	40		3	Sph	2	120
99	99	Below Faults	Primary	35	90	Az	M	1	11	15/99	2	90	100	6.0	1	Sph	4	10
			Second	35	45	Dip	I	2&3	20	15/99	2	45	100		2	Sph	3	40
			Tertiary	15	0	Tilt	F	4&5&6	50	15/99	2	0	40		3	Sph	2	120
xAg, cCu, cPb, cZn uses the gold variogram and search parameters and produces no classified results																		
1&2	1&2	Above Faults 1&2	Primary	35	90	Az						90	100	6.0	1	Sph	4	10
			Second	35	45	Dip						45	100		2	Sph	3	40
			Tertiary	15	0	Tilt			50	15/99	2	0	40		3	Sph	2	120
99	99	Below Faults	Primary	35	90	Az						90	100	6.0	1	Sph	4	10
			Second	35	45	Dip						45	100		2	Sph	3	40
			Tertiary	15	0	Tilt			50	15/99	2	0	40		3	Sph	2	120

Notes
1 Unitize General Relative (All variogram structures are transformed to relative variograms from log variograms)
2 Kriging Error is used to adjust preliminary class 1,3,5 to a final final resource class of 1,2,3,4,5&6
3 Az=Azimuth is clockwise (CW) from North, Dip is positive when downward, Tilt rotates CW around primary axis.
4 Sph=Spherical, Lin=Linear, Exp=Exponential, Gau=Gaussian
5 M=Measured, I=Indicated, F=Inferred

FIGURE 17-20: SUMMARY OF THE KRIGING PARAMETERS USED IN RESOURCE ESTIMATION

FIGURE 17-21 summarizes the block kriging results for gold. Results are in grams per tonne gold.

FIGURE 17-22 summarizes the block kriging results for silver. Results are in grams per tonne silver.

FIGURE 17-23 summarizes the block kriging results for copper. Results are in percent copper.

FIGURE 17-24 summarizes the block kriging results for lead. Results are in percent lead.

FIGURE 17-25 summarizes the block kriging results for zinc. Results are in percent zinc.

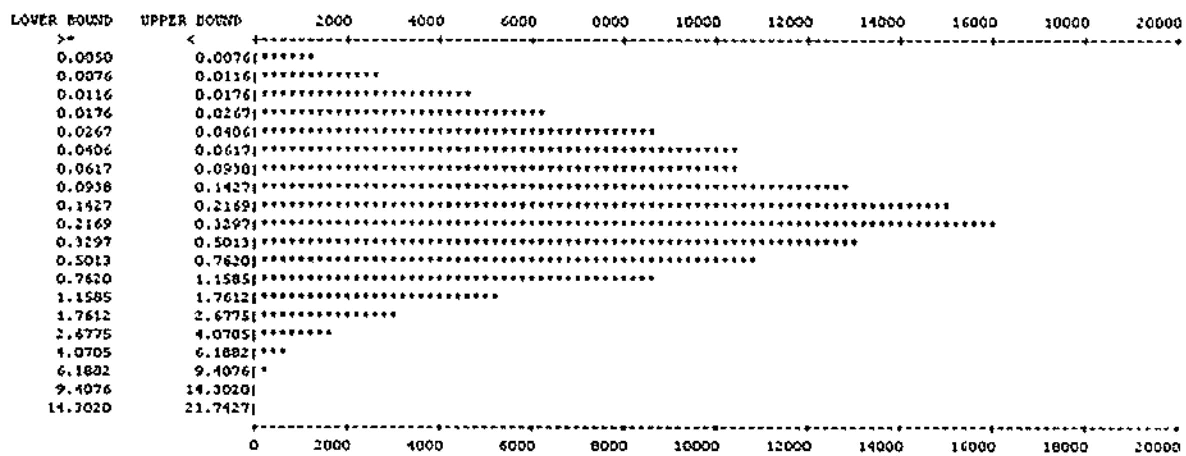
17.11 Comparing Surface Drillholes to Underground Drillholes

An initial comparison of gold assays from surface drilling (SDH) with underground drilling (UDH) was done in TABLE 17-2. The results suggested that the UDH data is enhanced an average of one half gram gold as compared to SDH data. This comparison is suspect in that while UDH data is focused on the mineralized zone, the SDH data reaches across unmineralized areas as well. A better comparison was done by kriging blocks using each data set. Figure 26A graphically shows the histogram of the difference in kriged grades. FIGURE 17-26B shows the side-by-side histograms of the two estimates for 603 blocks classified as measured. The average of the SDH is 0.947, which is derived by taking the exponential of the log average -0.0543. The average of the UDH is 0.903, which is derived by taking the exponential of the log average -.1026. The difference is a slight enhancement of the SDH data of 0.0447 g/t.

RUNTIME TITLE : Calculate Statistics
PROJECT TITLE : 10x10x5m
CURRENT LABEL : (G102) Kriged Grade XAuG

ROCK TYPE	BLOCK COUNT MISSING	BLOCK COUNT BELOW LIMITS	BLOCK COUNT ABOVE LIMITS	BLOCK COUNT INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
1	92373	0	0	95275	0.00500	21.741	0.53000	0.72263	0.85008	1.6660	-1.4124	1.5469	1.2438	0.5278	1.9228
2	2869	11	0	5449	0.00500	6.5502	0.57780	0.63131	0.79455	1.3751	-1.3949	1.9983	1.4136	0.6732	2.5251
99	288297	0	0	30445	0.00500	2.3003	0.11394	0.04096	0.20239	1.7763	-2.9953	1.4227	1.1928	0.1019	1.7744
ALL	383539	11	0	131169	0.00500	21.741	0.42089	0.58927	0.76764	1.8239	-1.7791	1.9840	1.4085	0.4552	2.5044

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ (ALL VALUES < UPPER BOUND)	PERCENT	CUM MEAN	CUM FREQ (ALL VALUES >= LOWER BOUND)	PERCENT	CUM MEAN
0.0050	0.0076	1116	0.85	0.0063	1116	0.85	0.0063	131169	100.00	0.4209
0.0076	0.0116	2596	1.98	0.0100	3712	2.83	0.0089	130053	99.15	0.4244
0.0116	0.0176	4661	3.55	0.0142	8373	6.38	0.0119	127457	97.17	0.4329
0.0176	0.0267	6138	4.68	0.0222	14511	11.06	0.0162	122796	93.62	0.4408
0.0267	0.0406	8684	6.62	0.0334	23195	17.68	0.0227	116658	88.94	0.4712
0.0406	0.0617	10354	7.89	0.0505	33549	25.58	0.0313	107974	82.32	0.5064
0.0617	0.0938	10492	8.00	0.0766	44041	33.58	0.0421	97620	74.42	0.5548
0.0938	0.1427	12739	9.71	0.1178	56780	43.29	0.0591	87128	66.42	0.6124
0.1427	0.2169	15085	11.50	0.1783	71865	54.79	0.0841	74389	56.71	0.6971
0.2169	0.3297	16048	12.23	0.2706	87913	67.02	0.1181	59304	45.21	0.8290
0.3297	0.5013	13095	9.98	0.4059	101008	77.01	0.1554	43256	32.98	1.0362
0.5013	0.7620	10888	8.30	0.6174	111896	85.31	0.2004	30161	22.99	1.3099
0.7620	1.1585	8632	6.58	0.9449	120528	91.89	0.2537	19273	14.69	1.7011
1.1585	1.7612	5194	3.96	1.4150	125722	95.85	0.3017	10641	8.11	2.3146
1.7612	2.6775	2936	2.24	2.1166	128658	98.09	0.3436	5447	4.15	3.1717
2.6775	4.0705	1555	1.19	3.2417	130213	99.27	0.3782	2511	1.91	4.3021
4.0705	6.1882	628	0.48	4.8446	130841	99.75	0.3996	956	0.73	6.2370
6.1882	9.4076	234	0.18	7.4289	131075	99.93	0.4122	328	0.25	8.9029
9.4076	14.3020	71	0.05	11.3161	131146	99.98	0.4181	94	0.07	12.5721
14.3020	21.7427	23	0.02	16.4495	131169	100.00	0.4209	23	0.02	16.4493



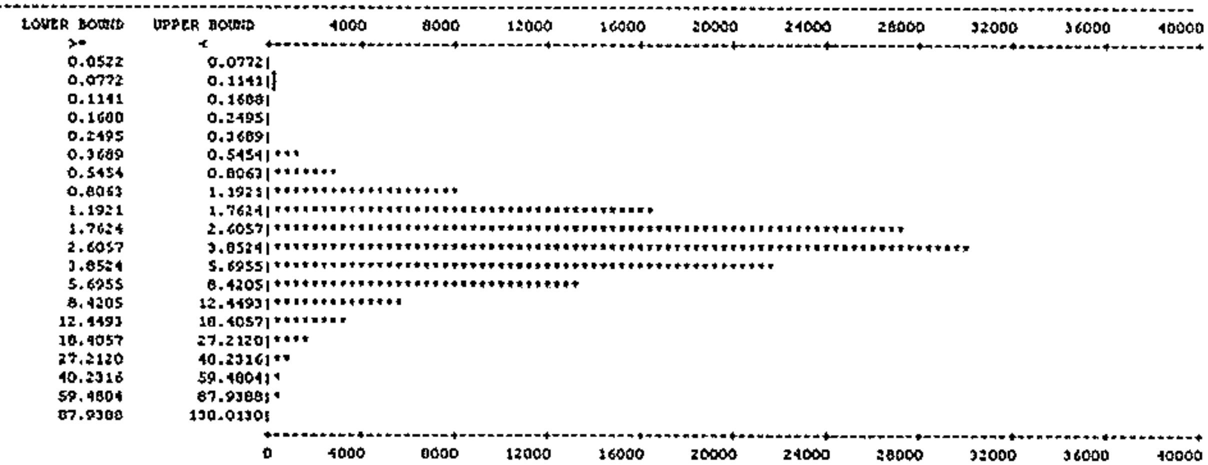
Issued by:	Prepared for:	File Name:
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-21.cdr
	Project: Silver Coin Gold Project	Project Number: 114-311007
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 17-21
Block Statistics of Kriged Gold g/t

RUNTIME TITLE : Calculate Statistics
PROJECT TITLE : 10x10x5m
CURRENT LABEL : (G101) Kriged Grade KAgG

ROCK TYPE	BLOCK COUNT: MISSING	BLOCK COUNT: BELOW LIMITS	BLOCK COUNT: ABOVE LIMITS	BLOCK COUNT: INSIDE LIMITS	UNTRANSFORMED STATISTICS: MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR.	LOG-TRANSFORMED STATS: LOG MEAN	LOG VAR.	LOG STD.DEV.	LOG-DERIVED: MEAN	COEF. OF VAR.
1	91994	0	0	95654	0.05222	130.00	4.8717	39.246	6.2647	1.2859	1.2702	0.4904	0.7060	4.5693	0.8038
2	2888	3	0	5438	0.18712	44.487	4.7362	25.597	5.0593	1.0682	1.2372	0.5532	0.7437	4.5437	0.8595
99	285546	0	0	31196	0.09854	41.043	2.6938	7.6097	2.7730	1.0294	0.6695	0.5842	0.7643	2.61602	0.8908
ALL	380428	3	0	132288	0.05222	130.00	4.3525	32.093	5.6651	1.3016	1.1272	0.5855	0.7652	4.1369	0.8922

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ (ALL VALUES < UPPER BOUND)	PERCENT	CUM MEAN	CUM FREQ (ALL VALUES >= LOWER BOUND)	PERCENT	CUM MEAN
0.0522	0.0772	1	0.00	0.0522	1	0.00	0.0522	132288	100.00	4.3525
0.0772	0.1141	2	0.00	0.0884	3	0.00	0.0763	132287	100.00	4.3526
0.1141	0.1688	8	0.01	0.1350	11	0.01	0.1190	132285	100.00	4.3526
0.1688	0.2495	32	0.02	0.2277	43	0.03	0.1999	132277	99.99	4.3529
0.2495	0.3689	132	0.10	0.3139	175	0.13	0.2859	132245	99.97	4.3539
0.3689	0.5454	1035	0.78	0.4676	1210	0.91	0.4413	132113	99.87	4.3579
0.5454	0.8063	2879	2.18	0.6773	4089	3.09	0.6075	131078	99.09	4.3886
0.8063	1.1921	8186	6.19	1.0163	12275	9.28	0.8801	128199	96.91	4.4720
1.1921	1.7624	16385	12.39	1.4886	28660	21.66	1.2280	120013	90.72	4.7077
1.7624	2.6057	27360	20.68	2.1947	56020	42.35	1.7001	103628	78.34	5.2167
2.6057	3.8524	29939	22.63	3.1609	85959	64.98	2.2089	76268	57.65	6.3008
3.8524	5.6955	21645	16.36	4.6504	107604	81.34	2.7000	46329	35.02	8.3298
5.6955	8.4205	13077	9.89	6.7738	120681	91.23	3.1414	24684	18.66	11.5562
8.4205	12.4493	5649	4.27	10.1060	126330	95.50	3.4529	11607	8.77	16.9444
12.4493	18.4057	3015	2.28	15.0153	129345	97.78	3.7224	5958	4.50	23.4281
18.4057	27.2120	1697	1.28	22.0119	131042	99.06	3.9592	2943	2.22	32.0468
27.2120	40.2316	661	0.50	31.9954	131703	99.56	4.1000	1246	0.94	45.7139
40.2316	59.4804	324	0.24	46.7537	132027	99.80	4.2046	585	0.44	61.2146
59.4804	87.9388	207	0.16	73.9630	132234	99.96	4.3138	261	0.20	79.1660
87.9388	130.0130	54	0.04	99.1126	132288	100.00	4.3525	54	0.04	99.1112

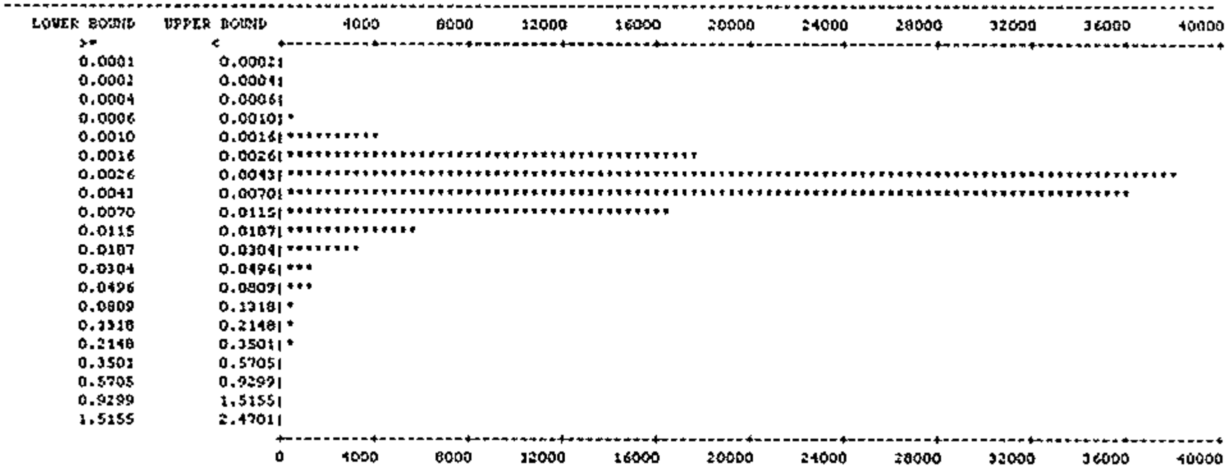


Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-22.cdr	**Figure 17-22**
	Project: Silver Coin Gold Project	Project Number: 114-311007	**Block Statistics of Kriged Silver g/t**
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

RUNTIME TITLE : Calculate Statistics
PROJECT TITLE : 10x10x5m
CURRENT LABEL : (G103) Kriged Grade %Cut

	BLOCK COUNT				UNTRANSFORMED STATISTICS						LOG-TRANSFORMED STATS			LOG-DERIVED	
ROCK TYPE	MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG MEAN	LOG VAR.	LOG STD.DEV	MEAN	COEF. OF VAR.
1	95625	0	0	92023	0.000500	2.4698	0.01480	0.00974	0.09860	6.6680	-5.2639	0.7465	0.8640	0.0075	1.0534
2	4243	3	0	4083	0.000141	0.02561	0.00527	0.000016	0.00394	0.7476	-5.4693	0.4311	0.6566	0.0052	0.7342
99	287854	0	0	28688	0.000500	0.22737	0.00677	0.000097	0.00985	1.4552	-5.3938	0.6564	0.0102	0.0063	0.9632
ALL	387722	3	0	124994	0.000141	2.4698	0.01263	0.00720	0.08488	6.7196	-5.3006	0.7109	0.0479	0.0071	1.0258

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ (ALL VALUES < UPPER BOUND)	PERCENT	CUM MEAN	CUM FREQ (ALL VALUES >= LOWER BOUND)	PERCENT	CUM MEAN
0.0001	0.0002	2	0.00	0.0001	2	0.00	0.0001	124994	100.00	0.0126
0.0002	0.0004	0	0.00	0.0000	2	0.00	0.0001	124992	100.00	0.0126
0.0004	0.0006	167	0.13	0.0005	169	0.14	0.0005	124992	100.00	0.0126
0.0006	0.0010	420	0.34	0.0009	589	0.47	0.0008	124825	99.86	0.0126
0.0010	0.0016	3905	3.12	0.0013	4494	3.60	0.0013	124405	99.53	0.0127
0.0016	0.0026	17570	14.06	0.0022	22064	17.65	0.0020	120500	96.40	0.0131
0.0026	0.0043	37811	30.25	0.0035	59875	47.90	0.0029	102930	82.35	0.0149
0.0043	0.0070	35917	28.73	0.0055	95792	76.64	0.0039	65119	52.10	0.0216
0.0070	0.0115	16244	13.00	0.0087	112036	89.63	0.0046	29202	23.36	0.0414
0.0115	0.0187	5525	4.42	0.0143	117561	94.05	0.0050	12958	10.37	0.0823
0.0187	0.0304	3079	2.46	0.0231	120640	96.52	0.0055	7433	5.95	0.1328
0.0304	0.0496	1367	1.09	0.0386	122007	97.61	0.0059	4354	3.48	0.2104
0.0496	0.0809	1199	0.96	0.0626	123206	98.57	0.0064	2987	2.39	0.2091
0.0809	0.1318	528	0.42	0.1032	123734	98.99	0.0068	1788	1.43	0.4409
0.1318	0.2148	401	0.32	0.1656	124135	99.31	0.0073	1260	1.01	0.5024
0.2148	0.3501	353	0.28	0.2731	124488	99.60	0.0081	859	0.69	0.7770
0.3501	0.5705	151	0.12	0.4219	124639	99.72	0.0086	506	0.40	1.1205
0.5705	0.9299	96	0.08	0.7210	124735	99.79	0.0091	355	0.28	1.4291
0.9299	1.5155	78	0.06	1.2187	124813	99.86	0.0099	259	0.21	1.6915
1.5155	2.4701	181	0.14	1.8953	124994	100.00	0.0126	181	0.14	1.8953



Issued by: TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

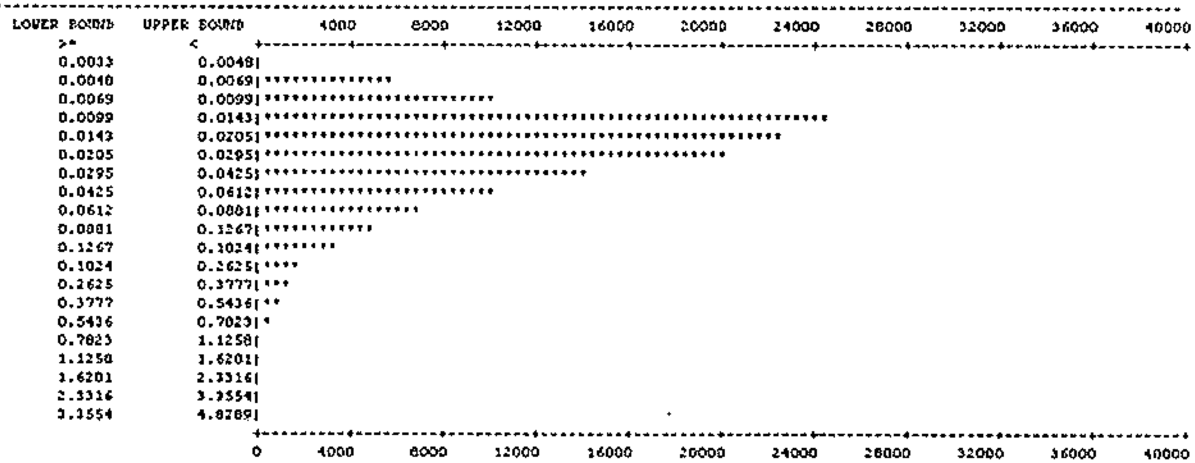
Prepared for:	File Name:
Pinnacle Mines Ltd.	Fig17-23.cdr
Project: Silver Coin Gold Project	Project Number: 114-311007
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 17-23
Block Statistics of Kriged Copper %Cu

RUNTIME TITLE : Calculate Statistics
PROJECT TITLE : 10x10x5m
CURRENT LABEL : (G104) Kriged Grade %Pb

ROCK TYPE	BLOCK COUNT: MISSING	BLOCK COUNT: BELOW LIMITS	BLOCK COUNT: ABOVE LIMITS	BLOCK COUNT: INSIDE LIMITS	UNTRANSFORMED STATISTICS: MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS: LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED: MEAN	COEF. OF VAR.
1	95508	0	0	92060	0.00499	4.8284	0.05245	0.01707	0.13066	2.4912	-3.5422	0.8358	0.9142	0.0440	1.1431
2	4243	3	0	4083	0.00333	0.30515	0.03946	0.00249	0.04992	1.2649	-3.7103	0.7881	0.8877	0.0363	1.0951
99	287854	0	0	28888	0.00500	0.33501	0.01884	0.000864	0.02939	1.5599	-4.3051	0.4631	0.6805	0.0170	0.7674
ALL	387685	3	0	125031	0.00333	4.8284	0.04426	0.01305	0.11423	2.5811	-3.7240	0.8506	0.9223	0.0369	1.1580

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ (ALL VALUES < UPPER BOUND)	PERCENT	CUM MEAN	CUM FREQ (ALL VALUES >= LOWER BOUND)	PERCENT	CUM MEAN
0.0033	0.0048	56	0.04	0.0034	56	0.04	0.0034	125031	100.00	0.0443
0.0048	0.0069	5411	4.33	0.0058	5467	4.37	0.0057	124975	99.96	0.0443
0.0069	0.0099	9963	7.97	0.0084	15430	12.34	0.0075	119564	95.63	0.0460
0.0099	0.0143	24340	19.47	0.0117	39770	31.81	0.0100	109601	87.66	0.0494
0.0143	0.0205	22492	17.99	0.0173	62262	49.80	0.0127	85261	68.19	0.0602
0.0205	0.0295	19940	15.95	0.0246	82202	65.75	0.0155	62769	50.20	0.0756
0.0295	0.0425	13982	11.18	0.0352	96184	76.93	0.0184	42829	34.25	0.0994
0.0425	0.0612	9915	7.93	0.0509	106099	84.86	0.0214	28847	23.07	0.1305
0.0612	0.0881	6902	5.52	0.0730	113001	90.38	0.0246	18932	15.14	0.1722
0.0881	0.1267	4756	3.80	0.1044	117757	94.18	0.0278	12030	9.62	0.2291
0.1267	0.1824	3207	2.56	0.1503	120964	96.75	0.0311	7274	5.82	0.3106
0.1824	0.2625	1524	1.22	0.2134	122488	97.97	0.0333	4067	3.25	0.4370
0.2625	0.3777	1298	1.04	0.3197	123786	99.00	0.0363	2543	2.03	0.5710
0.3777	0.5436	634	0.51	0.4398	124420	99.51	0.0384	1245	1.00	0.8330
0.5436	0.7823	251	0.20	0.6402	124671	99.71	0.0396	611	0.49	1.2411
0.7823	1.1258	114	0.09	0.9048	124785	99.80	0.0404	360	0.29	1.6600
1.1258	1.6201	79	0.06	1.3731	124864	99.87	0.0412	246	0.20	2.0099
1.6201	2.3316	111	0.09	1.9280	124975	99.96	0.0429	167	0.13	2.3112
2.3316	3.3554	49	0.04	2.0510	125024	99.99	0.0440	56	0.04	3.0700
3.3554	4.8289	7	0.01	4.6093	125031	100.00	0.0443	7	0.01	4.6092

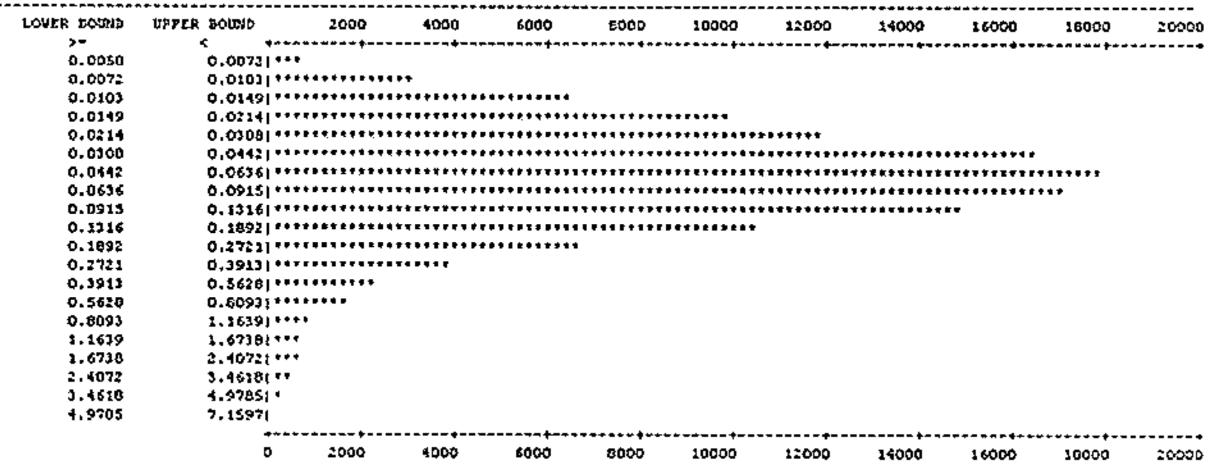


Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-24.cdr	**Figure 17-24 Block Statistics of Kriged Lead %Pb**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

RUNTIME TITLE : Calculate Statistics
PROJECT TITLE : 10x10x5m
CURRENT LABEL : (G105) Kriged Grade %Zn

ROCK TYPE	BLOCK COUNT: MISSING	BLOCK COUNT: BELOW LIMITS	BLOCK COUNT: ABOVE LIMITS	BLOCK COUNT: INSIDE LIMITS	UNTRANSFORMED STATISTICS: MINIMUM	UNTRANSFORMED STATISTICS: MAXIMUM	UNTRANSFORMED STATISTICS: MEAN	UNTRANSFORMED STATISTICS: VARIANCE	UNTRANSFORMED STATISTICS: STD. DEV.	UNTRANSFORMED STATISTICS: COEF. OF VAR	LOG-TRANSFORMED STATS: LOG MEAN	LOG-TRANSFORMED STATS: LOG VAR.	LOG-TRANSFORMED STATS: LOG STD.DEV	LOG-DERIVED: MEAN	LOG-DERIVED: COEF. OF VAR.
1	95566	0	0	92082	0.00500	7.1590	0.15181	0.10746	0.32780	2.1593	-2.5830	1.0910	1.0445	0.1304	1.4061
2	4243	3	0	4083	0.01000	0.83469	0.08139	0.00742	0.08612	1.0581	-2.8860	0.6894	0.8303	0.0708	0.9962
99	287854	0	0	28888	0.00500	4.5328	0.08137	0.12138	0.34840	4.2817	-3.3271	0.9813	0.9906	0.0586	1.2914
ALL	387663	3	0	125053	0.00500	7.1590	0.13324	0.10837	0.32920	2.4707	-2.7648	1.1503	1.0725	0.1120	1.4695

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ (ALL VALUES < UPPER BOUND)	PERCENT (ALL VALUES < UPPER BOUND)	CUM MEAN (ALL VALUES < UPPER BOUND)	CUM FREQ (ALL VALUES >= LOWER BOUND)	PERCENT (ALL VALUES >= LOWER BOUND)	CUM MEAN (ALL VALUES >= LOWER BOUND)
0.0050	0.0072	668	0.53	0.0059	668	0.53	0.0059	125053	100.00	0.1332
0.0072	0.0103	3060	2.45	0.0094	3720	2.98	0.0087	124385	99.47	0.1339
0.0103	0.0149	6391	5.11	0.0128	10119	8.09	0.0113	121325	97.02	0.1371
0.0149	0.0214	9727	7.78	0.0180	19846	15.87	0.0146	114934	91.91	0.1440
0.0214	0.0308	11873	9.49	0.0260	31719	25.36	0.0189	105207	84.13	0.1556
0.0308	0.0442	16477	13.18	0.0373	48196	38.54	0.0252	93334	74.64	0.1721
0.0442	0.0636	17750	14.19	0.0534	65946	52.73	0.0328	76857	61.46	0.2010
0.0636	0.0915	17023	13.61	0.0763	82969	66.35	0.0417	59107	47.27	0.2453
0.0915	0.1316	14771	11.81	0.1096	97740	78.16	0.0520	42084	33.65	0.3137
0.1316	0.1892	10491	8.39	0.1571	108231	86.55	0.0622	27313	21.84	0.4241
0.1892	0.2721	6544	5.23	0.2257	114775	91.78	0.0715	16822	13.45	0.5906
0.2721	0.3913	3896	3.12	0.3209	118671	94.90	0.0797	10278	8.22	0.8229
0.3913	0.5628	2109	1.69	0.4640	120780	96.58	0.0864	6382	5.10	1.1293
0.5628	0.8093	1587	1.27	0.6760	122367	97.85	0.0940	4273	3.42	1.4577
0.8093	1.1639	857	0.69	0.9591	123224	98.54	0.1000	2686	2.15	1.9196
1.1639	1.6738	610	0.49	1.3812	123834	99.03	0.1064	1829	1.46	2.3696
1.6738	2.4072	544	0.44	2.0200	124378	99.46	0.1147	1219	0.97	2.8642
2.4072	3.4610	381	0.30	2.8714	124759	99.76	0.1231	675	0.54	3.5446
3.4618	4.9785	271	0.22	4.3131	125030	99.98	0.1322	294	0.24	4.4172
4.9785	7.1597	23	0.02	5.6434	125053	100.00	0.1332	23	0.02	5.6433

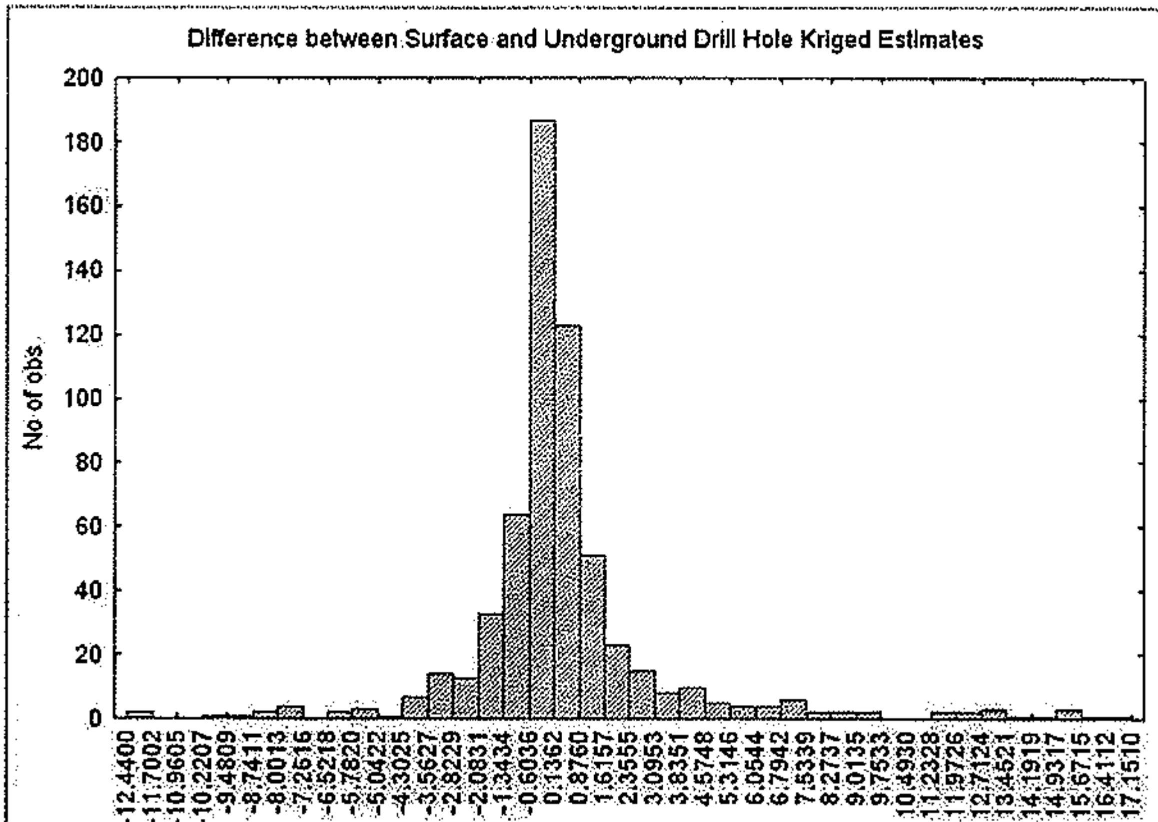


Issued by: TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

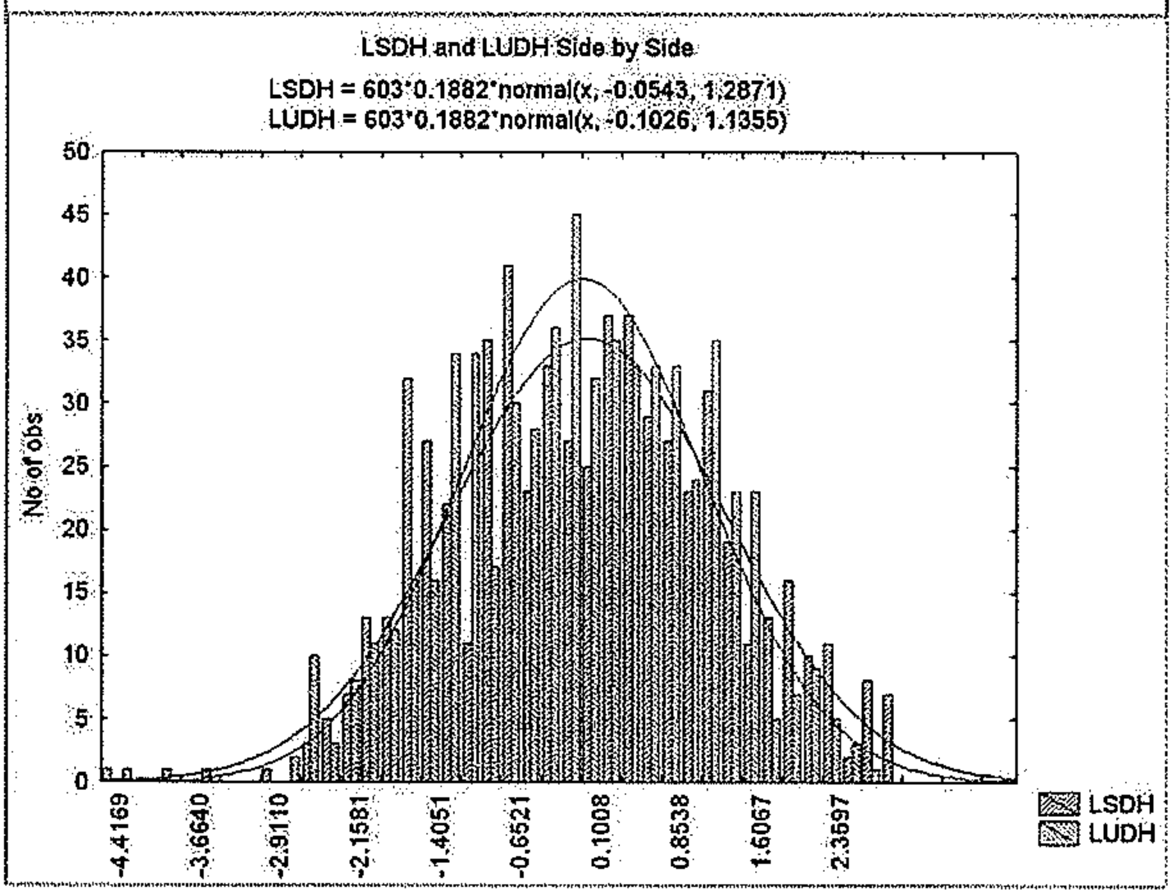
Prepared for:	File Name:
Pinnacle Mines Ltd.	Fig17-25.cdr
Project: Silver Coin Gold Project	Project Number: 114-311007
Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009

Figure 17-25
Block Statistics of Kriged Zinc %Zn

(A)



(B)



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-26.cdr	Figure 17-26 **Kriged Estimates by Surface DH (SDH) & Underground DH (UDH) Compared**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

17.12 Mineral Resource Classification and Reporting

Resource Classification Criteria

The Minefill Services study classified measured resources as blocks estimated from sample spacing that was less than 11m. Minefill also reported blocks estimated from samples with spacing less than 20m were classified as indicated and any spacing greater as inferred. Tt considers these spacing values as a reasonable starting point. Micromodel® allows for the jackknifing technique to apply search conditions that can duplicate the spacing.

FIGURE 17-27 is a scatter plot of the results of a gold jackknife study limited to search window of 11 meters. The correlation at the 11m distance is 0.755. This represents an initial assignment to a measured class. The Tt resource class for measured is 1.

FIGURE 17-28 is a scatter plot of the results of a gold jackknife study limited to search window that is greater than 11 meters and less than 20 meters. The correlation at this sampling geometry is 0.615. This represents an initial assignment to an indicated class. The Tt resource class for indicated is 3.

FIGURE 17-29 is a scatter plot of the results of a gold jackknife study limited to search window that is greater than 20 meters. The correlation at this sampling geometry is 0.532. This represents an initial assignment to an inferred class. The Tt resource class for inferred is 5.

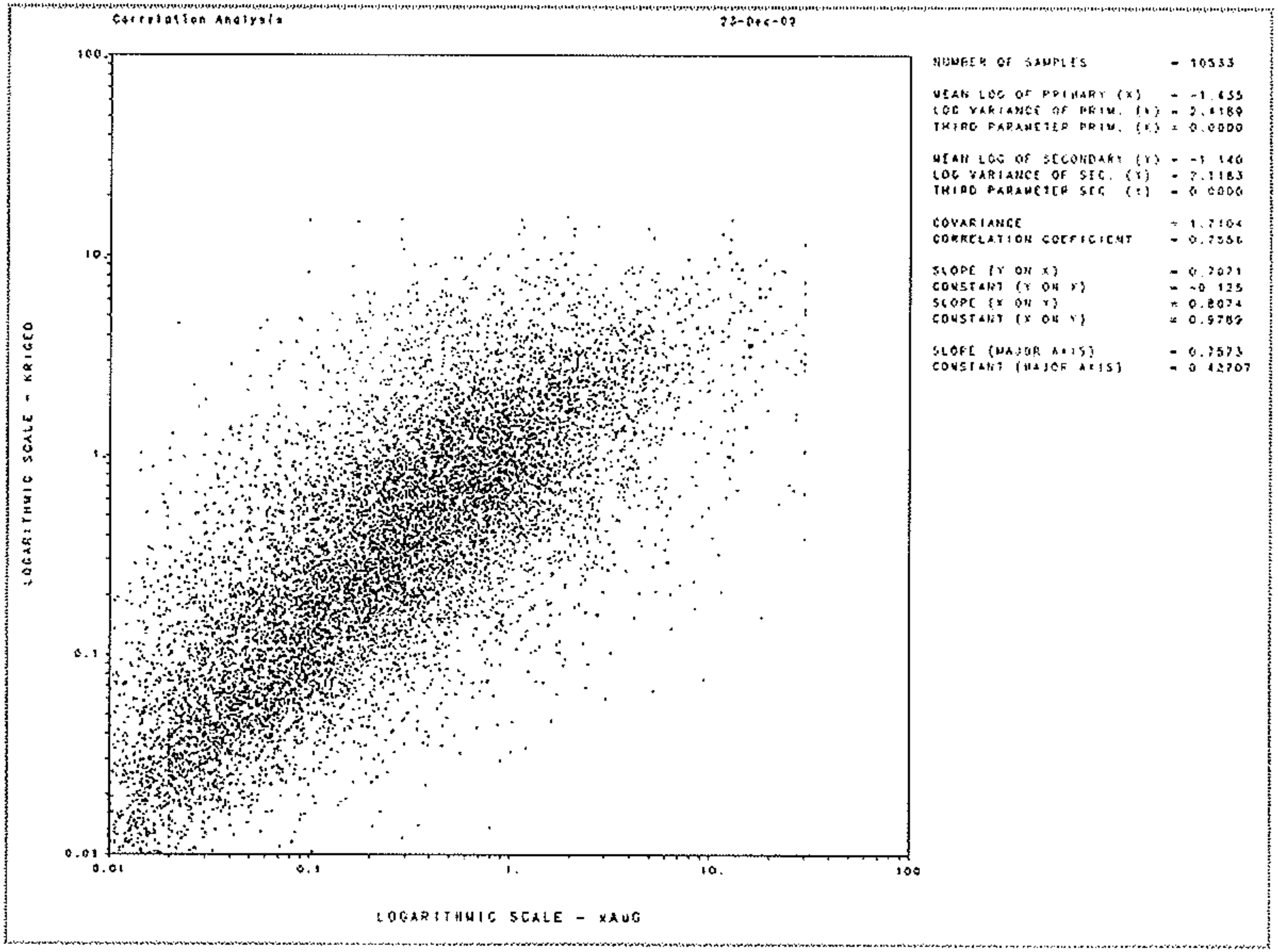
Kriging is a mathematical algorithm that has many similarities to regression. For every estimate; kriging also produces a kriging error. The kriging error embodies a quantitative measurement of the quality of the kriging estimate. It is much more sophisticated than a simple measure of sample spacing. Kriging error takes into account both the anisotropy of the deposit, hence the direction that samples are from the block and whether the there are areas that are over sampled (i.e. clustering of data). FIGURE 17-30 plots the kriging errors for the Tt block model on a cumulative frequency graph. Note that a straight line plotted on this type of graph represents a normal distribution. For this block model, two lines are required to fit the data. These two lines represent a mixture of two normal populations of kriging error. The "break" in the fit, represents the kriging error going from the one population to the next. This information can be used to adjust the resource classification. Equation 1 is written in an Excel-style conditional formula. The formula shifts the class of a block from measured to indicated or indicated to inferred when its kriging error is greater than 6.

FIGURE 17-31A shows the histogram of the initial resource class assignments. The percentage of blocks in the model for measured is 11.36%, for indicated is 24.9% and inferred is 63.74%.

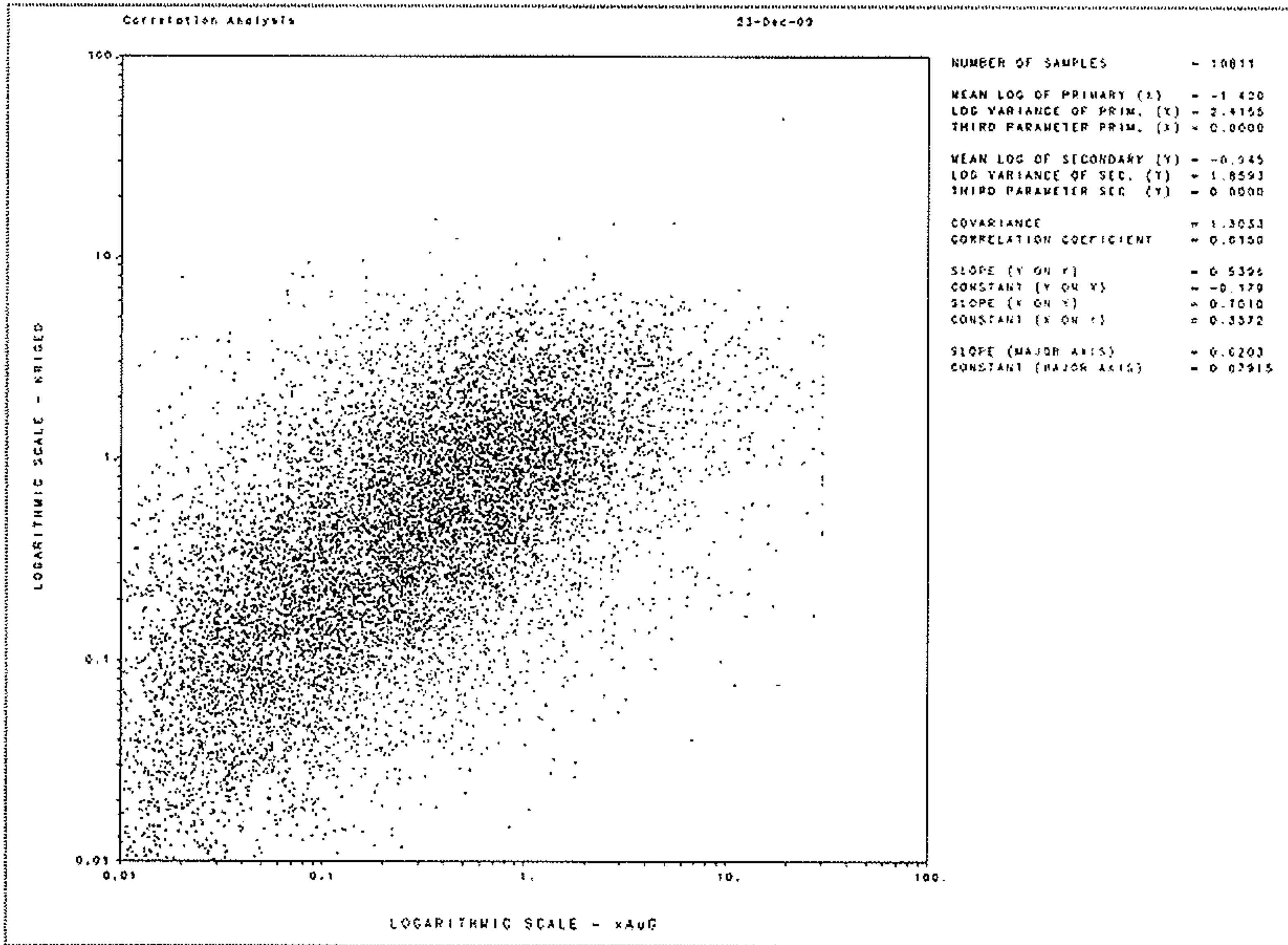
After the kriging error adjustment, FIGURE 17-31C shows the histogram for the final resource class assignments. The updated resource classification has the percentage for measured as 8.74%, for indicated as 18.57% and for inferred as 72.68%. Note that the designation of codes 5 and 6 are combined into the inferred class.

Resource Reporting

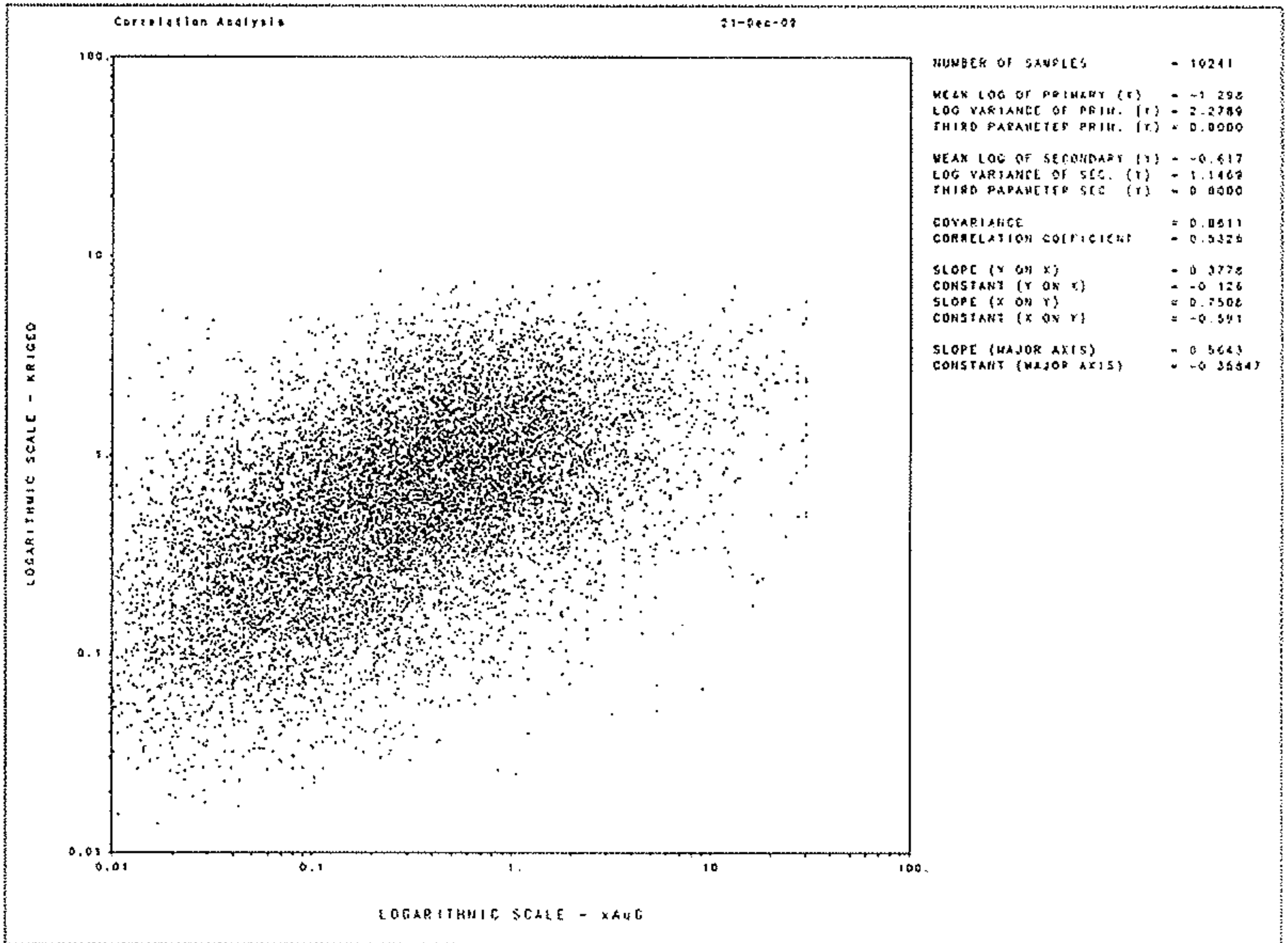
TABLE 17-3 shows the Silver Coin resources tabulated by gold grade and resource classification of measured, indicated and inferred. FIGURES 17-32 through 17-43 illustrate the relationship between estimated resources, lithologic controls, and resource classification. It is Tt's opinion that the reported mineral classes comply with current CIMM definitions for each mineral class. The **BOLDED** line indicates the base case cutoff grade scenario.



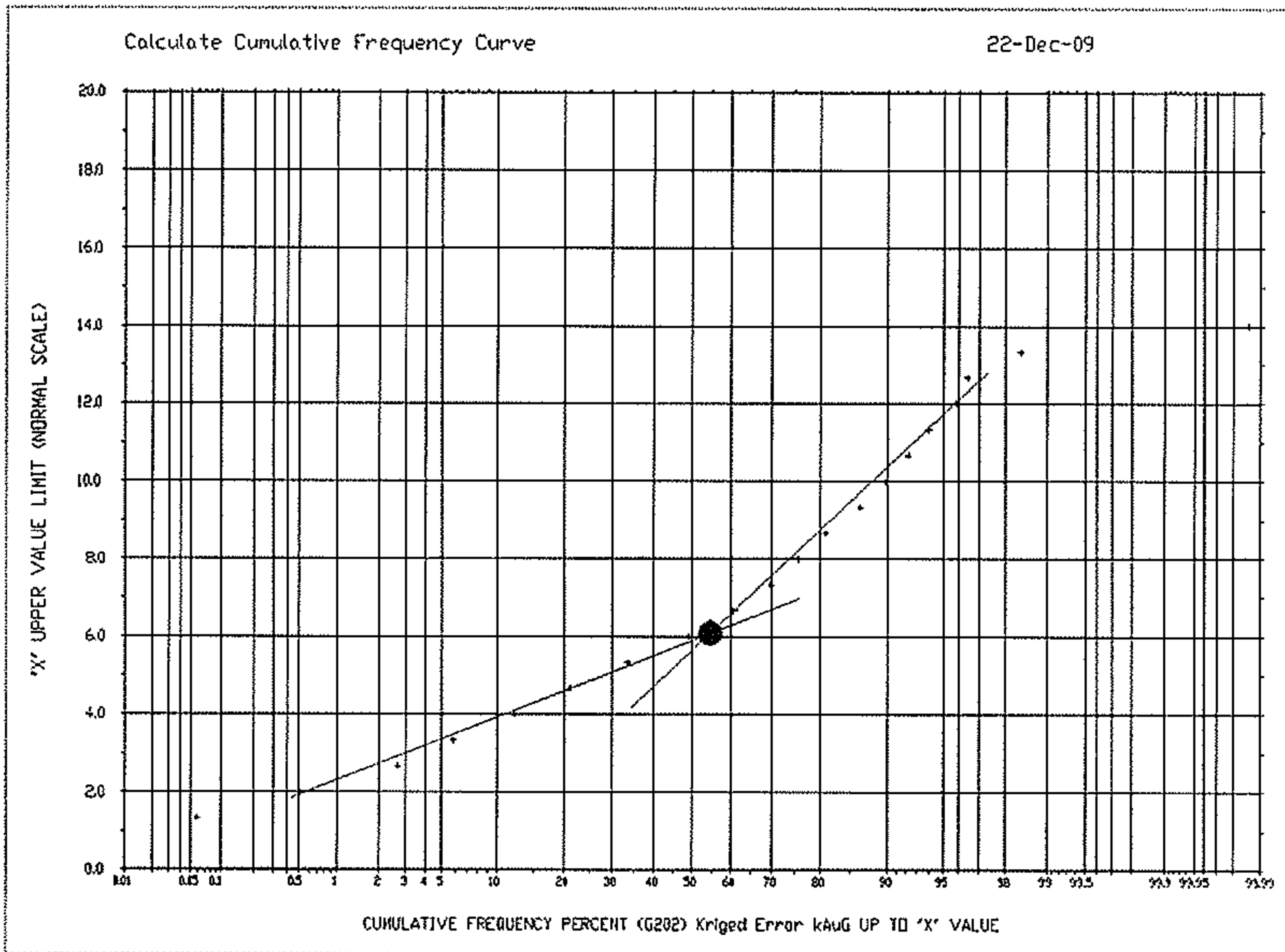
Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: **Pinnacle Mines Ltd.**	File Name: Fig17-27.cdr	**Figure 17-27**
	Project: Silver Coin Gold Project	Project Number: 114-311007	**Scatter Plot showing Results of Jackknife study - <=11m distance**
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	**(Measured class)**



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-28.cdr	**Figure 17-28**
	Project: Silver Coin Gold Project	Project Number: 114-311007	**Scatter Plot showing Results of Gold Jackknife study - >11m & <20m distance**
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	**(Indicated class)**



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-29.cdr	**Figure 17-29** **Scatter Plot showing Results of Gold Jackknife study - >20m & <50m distance (Inferred class)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-30.cdr	**Figure 17-30 Kriging Error Break Point**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	

A) Initial Resource Class, 1=Indicated (11m maximum search), 3 = Indicated (20m max), 5= Inferred (50m max)

```
RUNTIME TITLE : Calculate Statistics
PROJECT TITLE : 10x10x5m
CURRENT LABEL : (G502) Polygon Grade KAuG
    MINIMUM CUT-OFF ENTERED          =        0.100000
    MAXIMUM CUT-OFF ENTERED          =        7.000000
```

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ	PERCENT	CUM MEAN (ALL VALUES < UPPER BOUND)	CUM FREQ	PERCENT	CUM MEAN (ALL VALUES >= LOWER BOUND)
0.1000	1.8250	14902	11.36	1.0000	14902	11.36	1.0000	131180	100.00	4.0476
1.8250	3.5500	32661	24.90	3.0000	47563	36.26	2.3734	116278	88.64	4.4382
3.5500	5.2750	83617	63.74	5.0000	131180	100.00	4.0476	83617	63.74	5.0000
5.2750	7.0000	0	0.00	0.0000	131180	100.00	4.0476	0	0.00	0.0000

```
  LOWER BOUND  UPPER BOUND      10000     20000     30000     40000     50000     60000     70000     80000     90000    100000
      >=            <      +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
     0.1000        1.8250|***************
     1.8250        3.5500|*********************************
     3.5500        5.2750|************************************************************************************
     5.2750        7.0000|
                           +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                           0       10000     20000     30000     40000     50000     60000     70000     80000     90000    100000
```

B) Resource Class Transform Equation modifying Initial to Final using Kriging Error

[Final Class] = IF ([Kriged Error] > 6) , [Initial Class]+1 , [Initial Class]) (Equation 1)

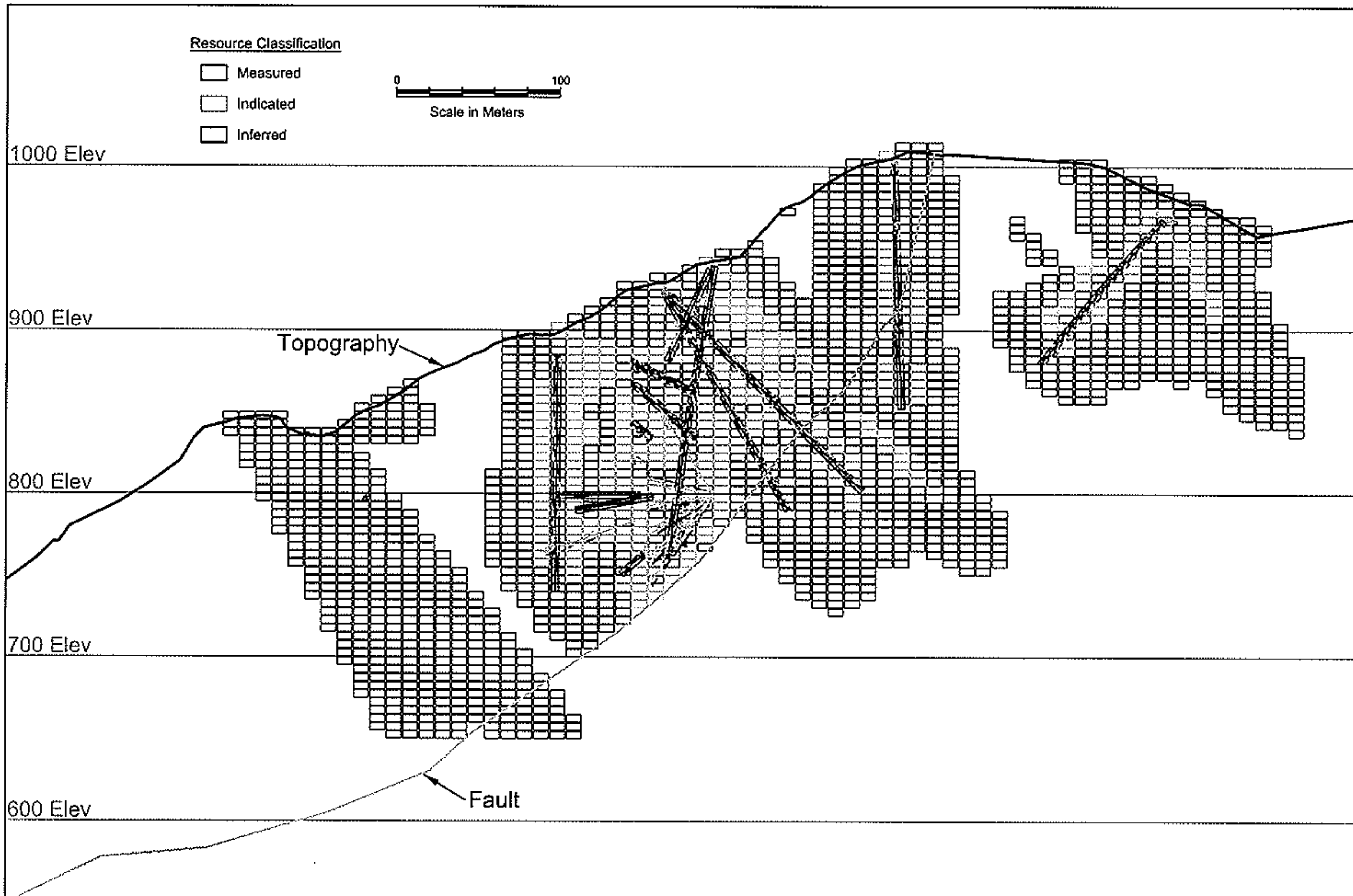
C) Final Resource Class, 1=Measured, 2&3 = Indicated, 4&5 = Drill Hole Inferred, 6 = Geologic Inferred

```
RUNTIME TITLE : Calculate Statistics
PROJECT TITLE : 10x10x5m
CURRENT LABEL : (G602) Triang. Grade KAuG
    MINIMUM CUT-OFF ENTERED          =        0.900000
    MAXIMUM CUT-OFF ENTERED          =        7.000000
```

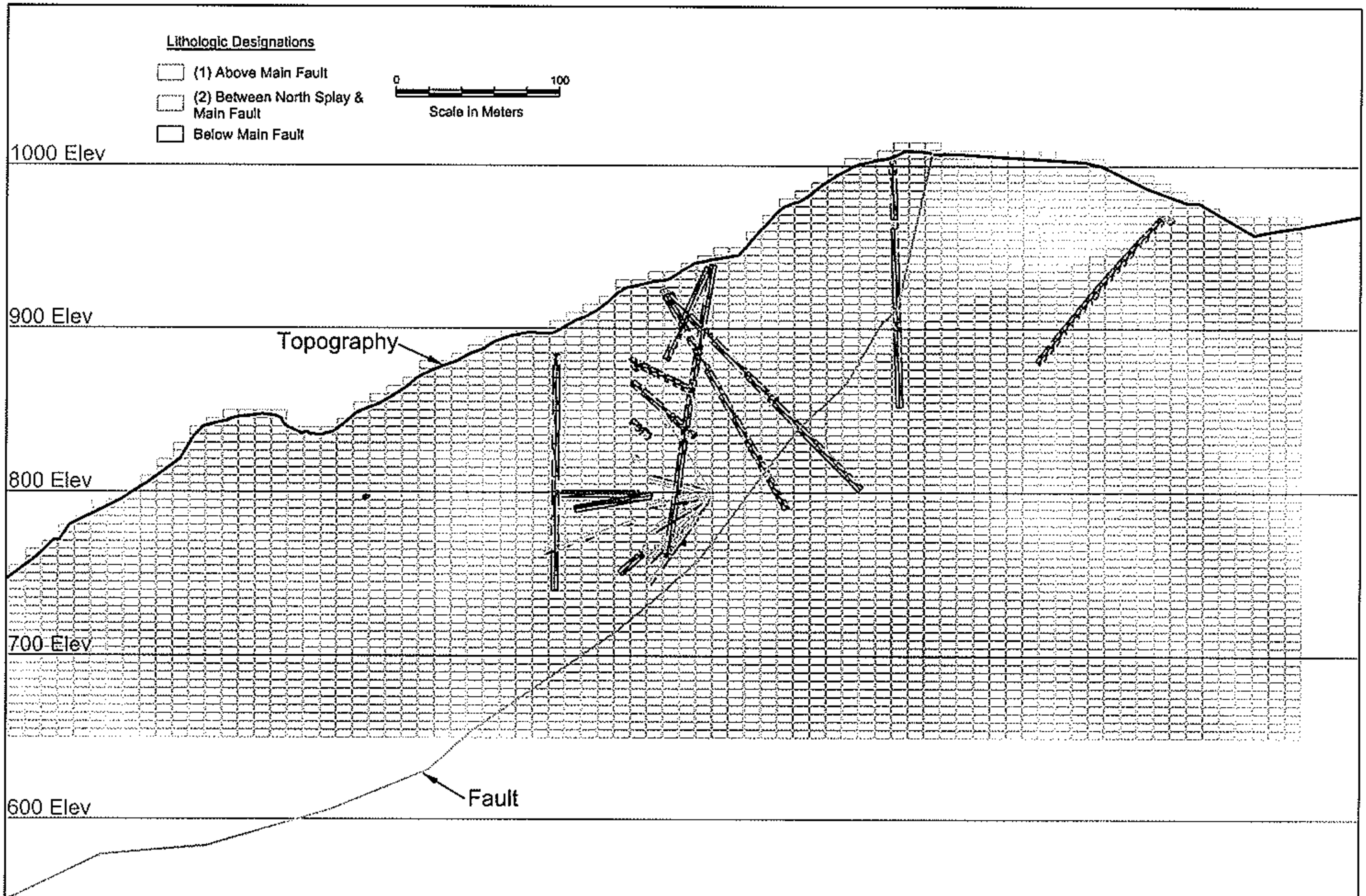
LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ	PERCENT	CUM MEAN (ALL VALUES < UPPER BOUND)	CUM FREQ	PERCENT	CUM MEAN (ALL VALUES >= LOWER BOUND)
0.9000	1.1000	11470	8.74	1.0000	11470	8.74	1.0000	131180	100.00	4.5588
1.1000	3.1000	24362	18.57	2.8591	35832	27.32	2.2640	119710	91.26	4.8997
3.1000	5.1000	43463	33.13	4.7301	79295	60.45	3.6157	95348	72.68	5.4211
5.1000	7.0000	51885	39.55	6.0000	131180	100.00	4.5588	51885	39.55	6.0000

```
  LOWER BOUND  UPPER BOUND       6000     12000     18000     24000     30000     36000     42000     48000     54000     60000
      >=            <      +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
     0.9000        1.1000|*******************
     1.1000        3.1000|*****************************************
     3.1000        5.1000|************************************************************************
     5.1000        7.0000|**************************************************************************************
                           +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                           0        6000     12000     18000     24000     30000     36000     42000     48000     54000     60000
```

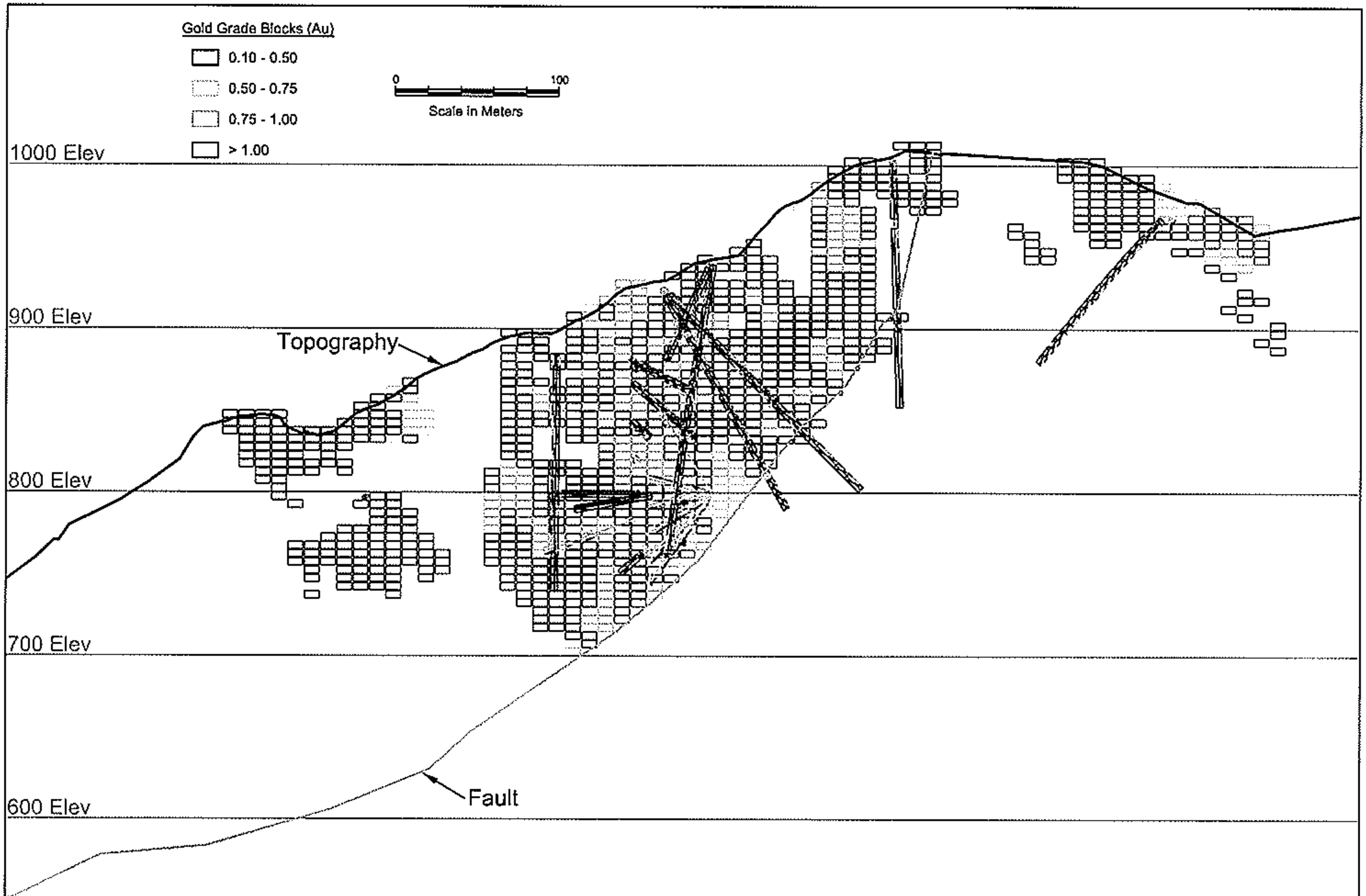
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-31.cdr	**Figure 17-31** **Histogram Visualization of Resource Class Percentages**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/24/2009	



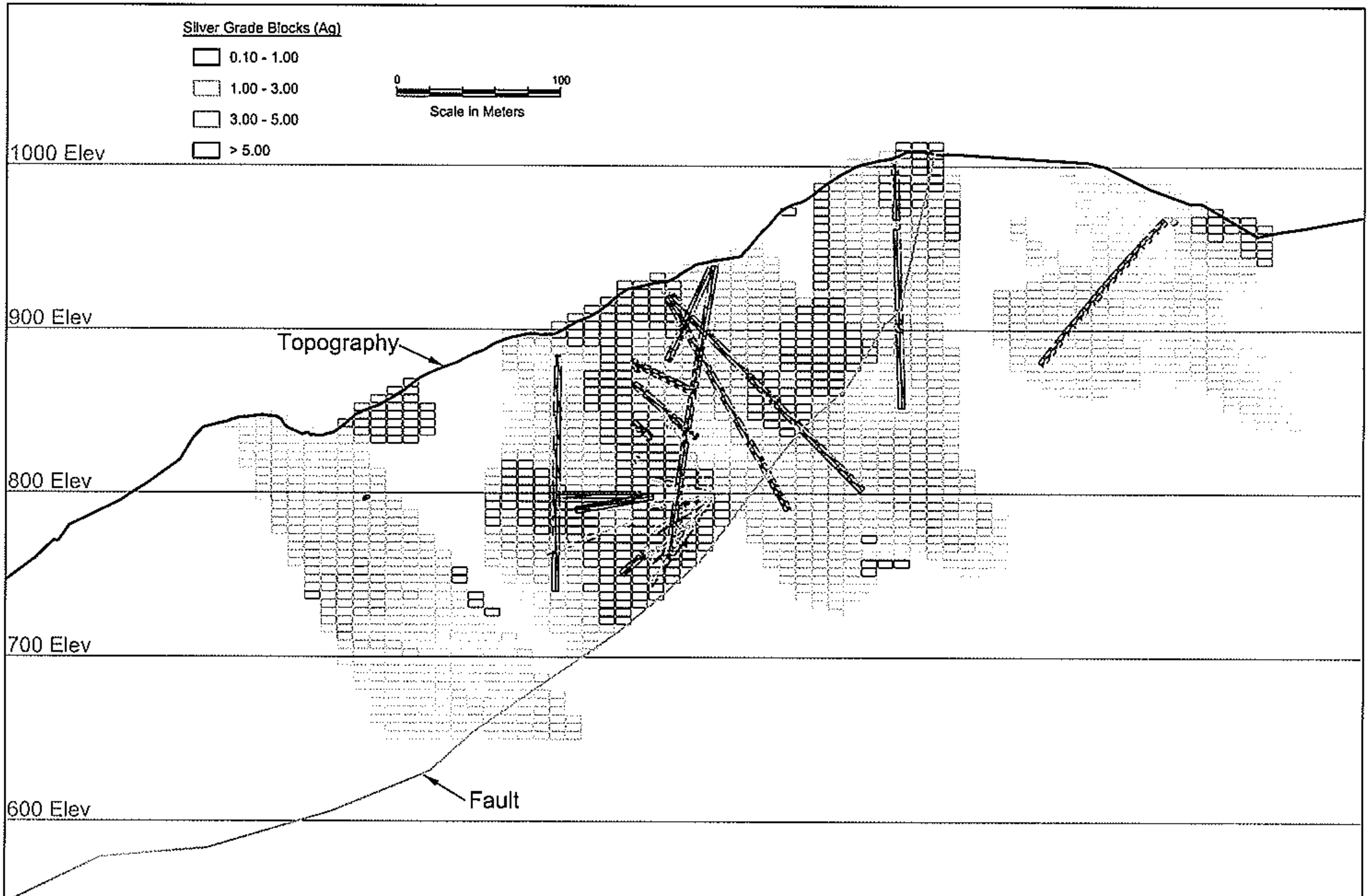
Issued by: TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Prepared for: **Pinnacle Mines Ltd.**	File Name: Fig17-32_35.dwg	Figure 17-32 **East-West Section 6,218,165 N Showing Resource Classification (Looking North)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/28/2009	



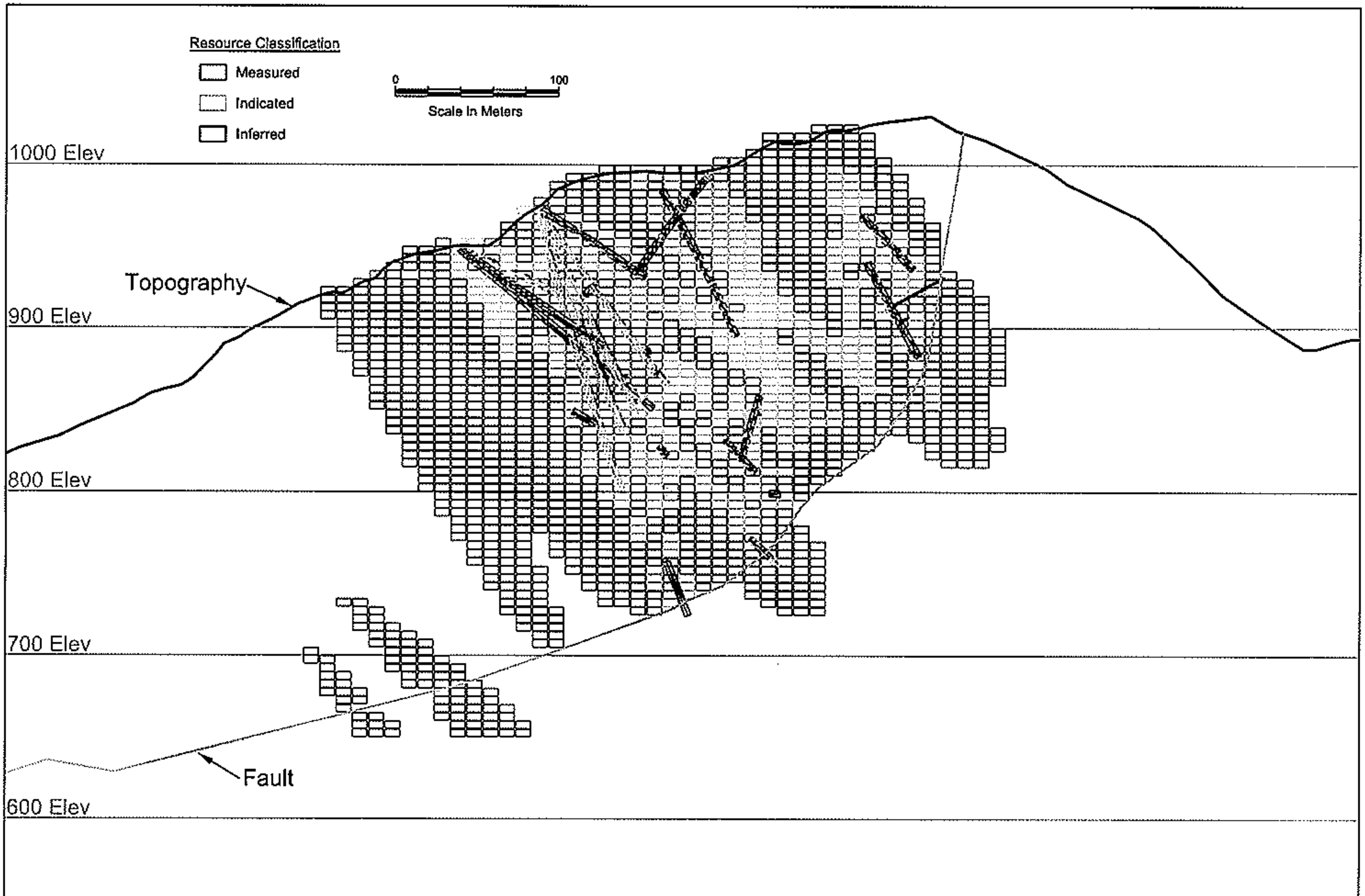
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-32_35.dwg	**Figure 17-33** **East-West Section 6,218,165 N** **Showing Lithologic Designations** **(Looking North)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/29/2009	



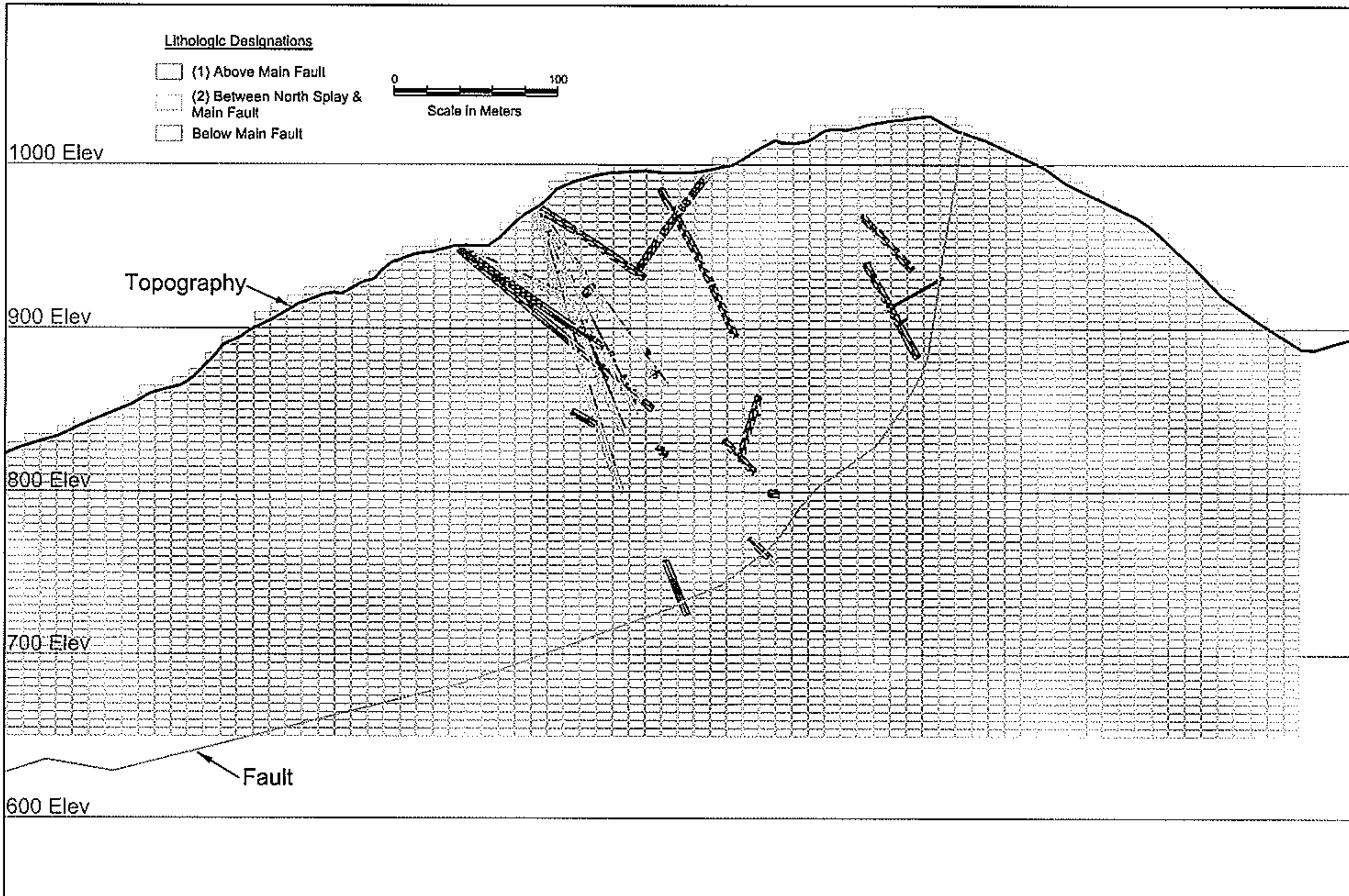
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-32_35.dwg	Figure 17-34 East-West Section 6,218,165 N Showing Gold Distribution by Grade (Looking North)
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/29/2009	



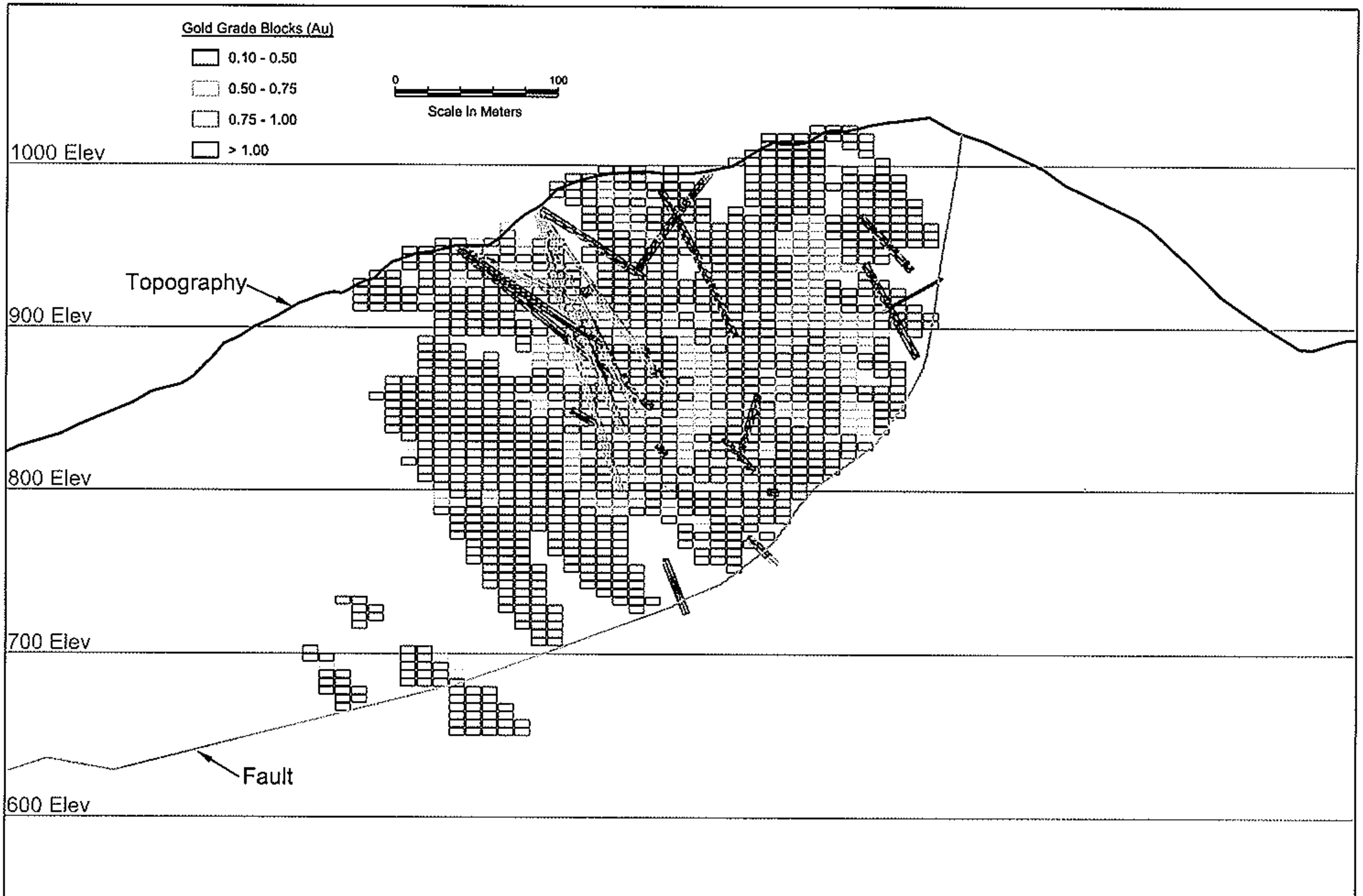
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-32_35.dwg	Figure 17-35 **East-West Section 6,218,165 N** **Showing Silver Distribution by Grade** **(Looking North)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/29/2009	



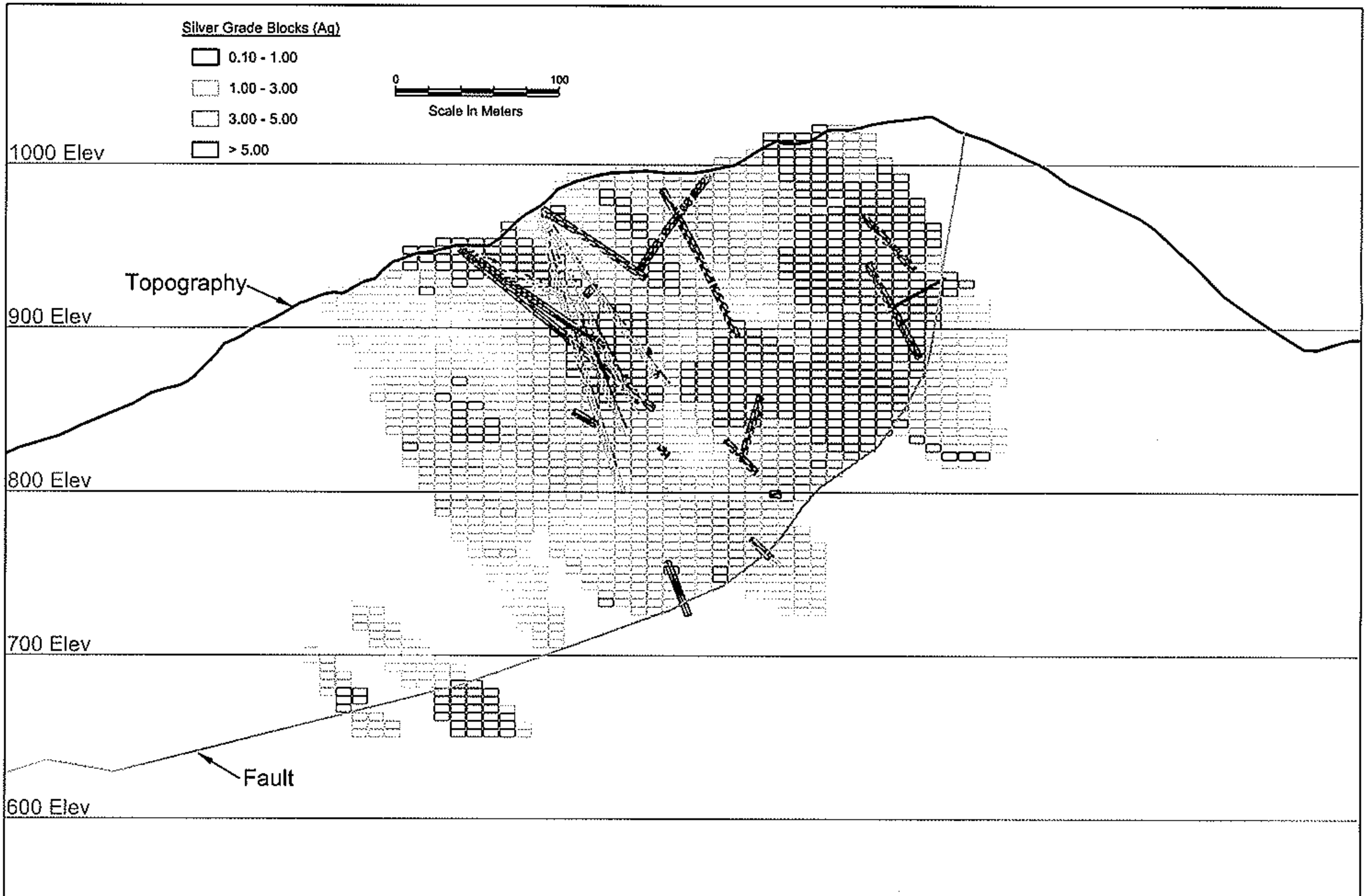
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-36_39.dwg	Figure 17-36 East-West Section 6,217,965 N Showing Resource Classification (Looking North)
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/28/2009	



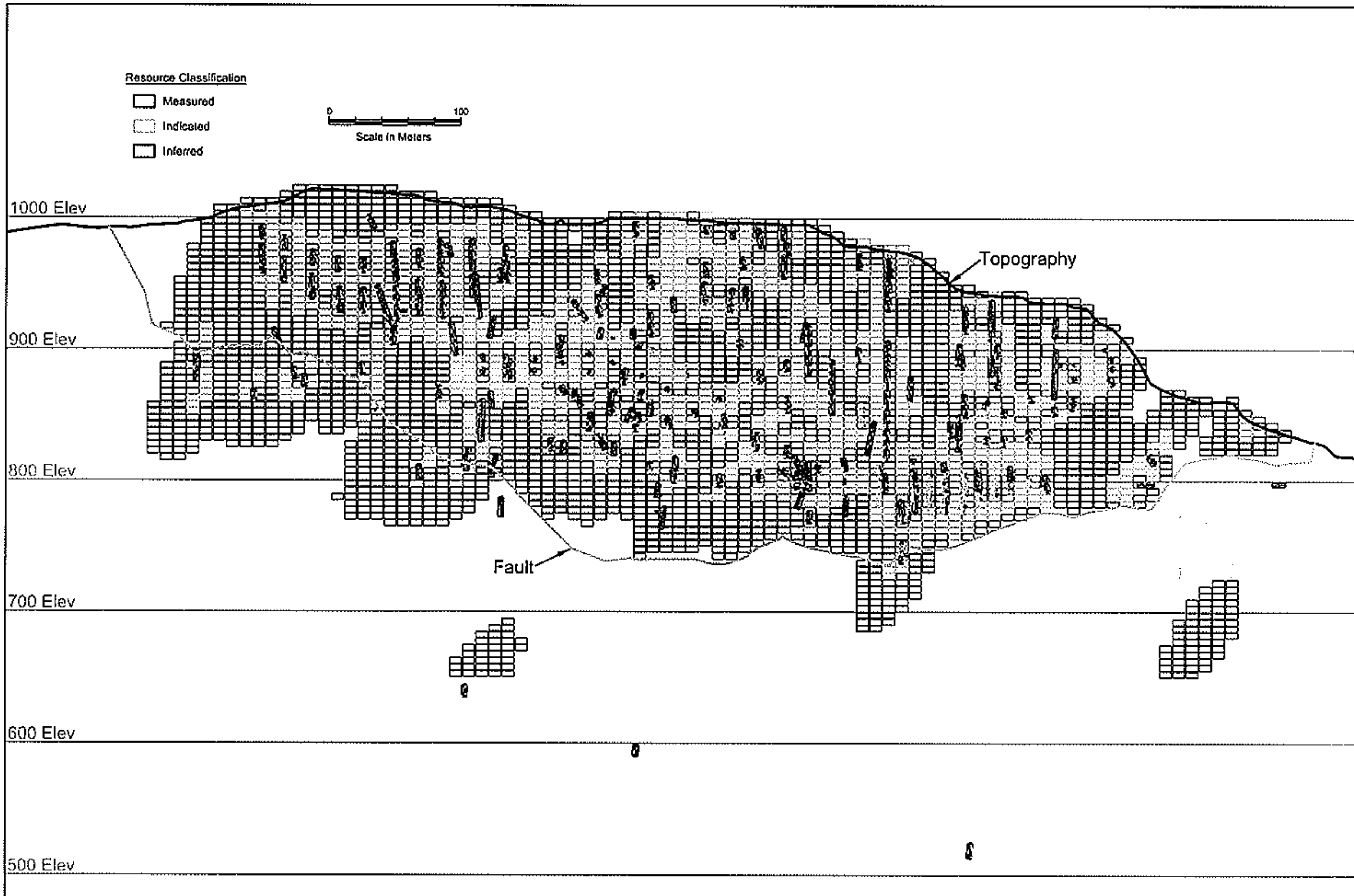
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-36_39.dwg	Figure 17-37 East-West Section 6,217,965 N Showing Lithologic Designations (Looking North)
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/29/2009	



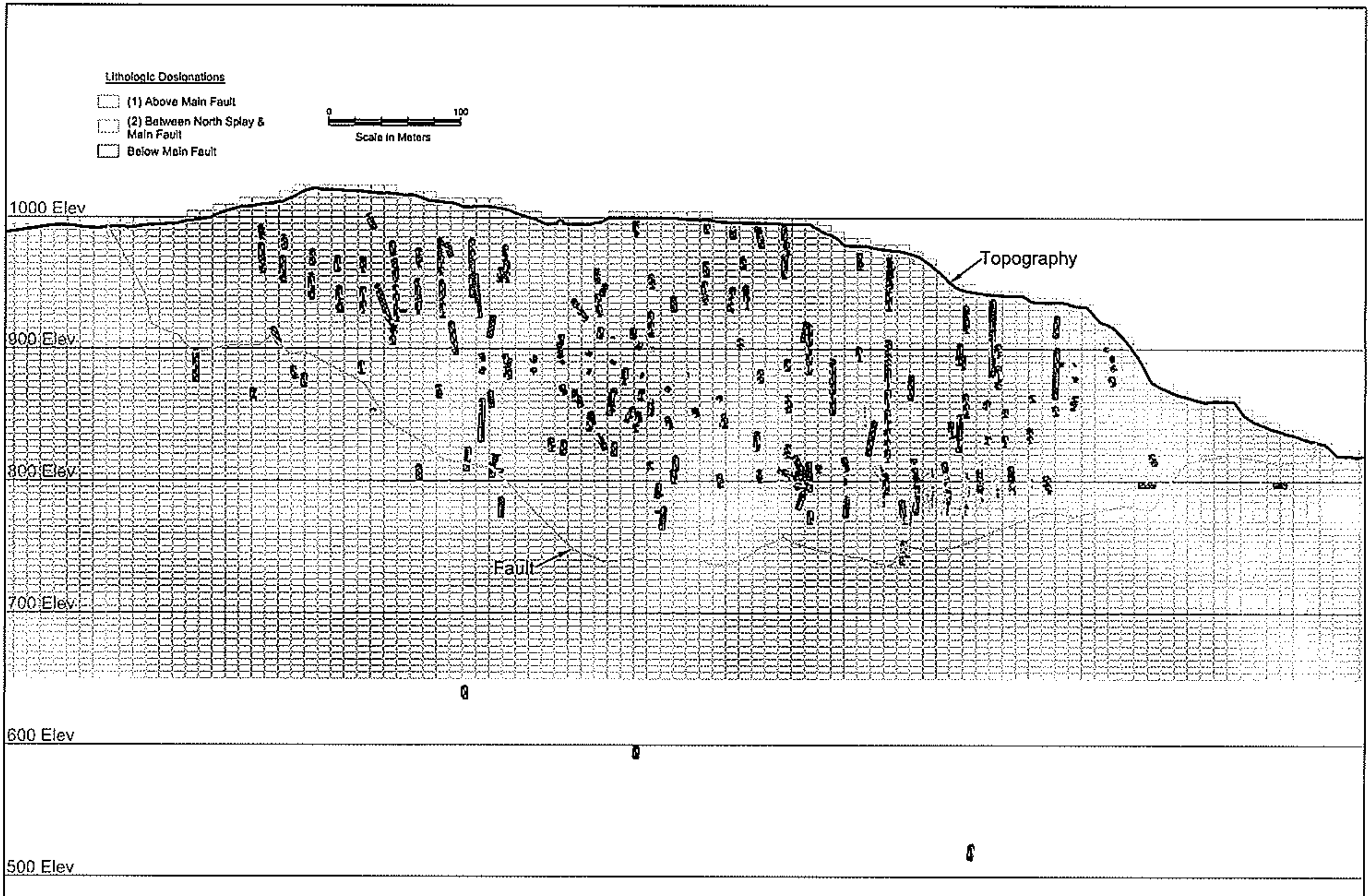
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Pinnacle Mines Ltd.	Fig17-36_39.dwg	Figure 17-38 **East-West Section 6,217,965 N** **Showing Gold Distribution by Grade** **(Looking North)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/29/2009	



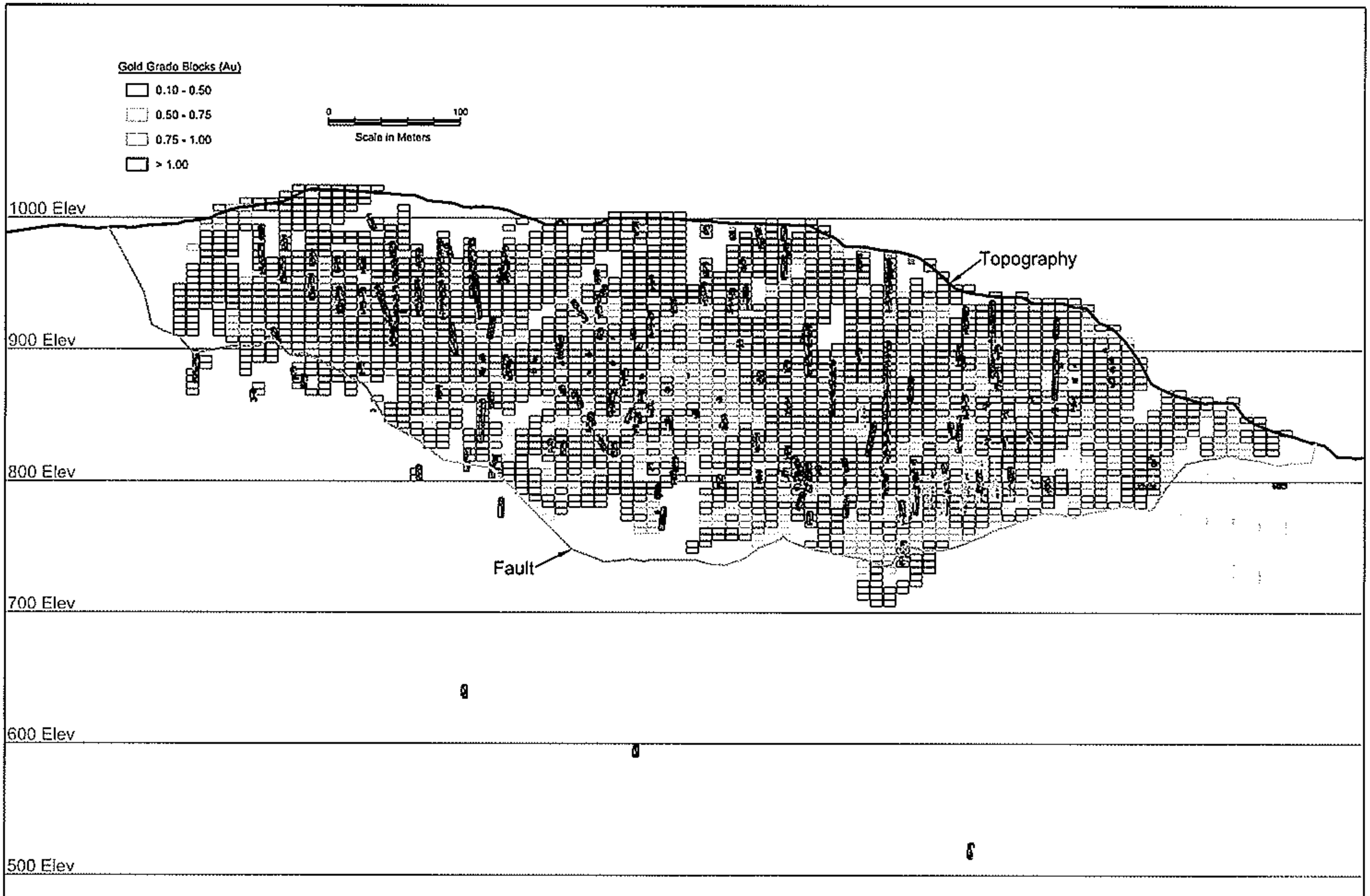
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-36_39.dwg	Figure 17-39
	Project: Silver Coin Gold Project	Project Number: 114-311007	East-West Section 6,217,965 N Showing Silver Distribution by Grade
	Project Location: Stewart, British Columbia	Date of Issue: 12/29/2009	(Looking North)



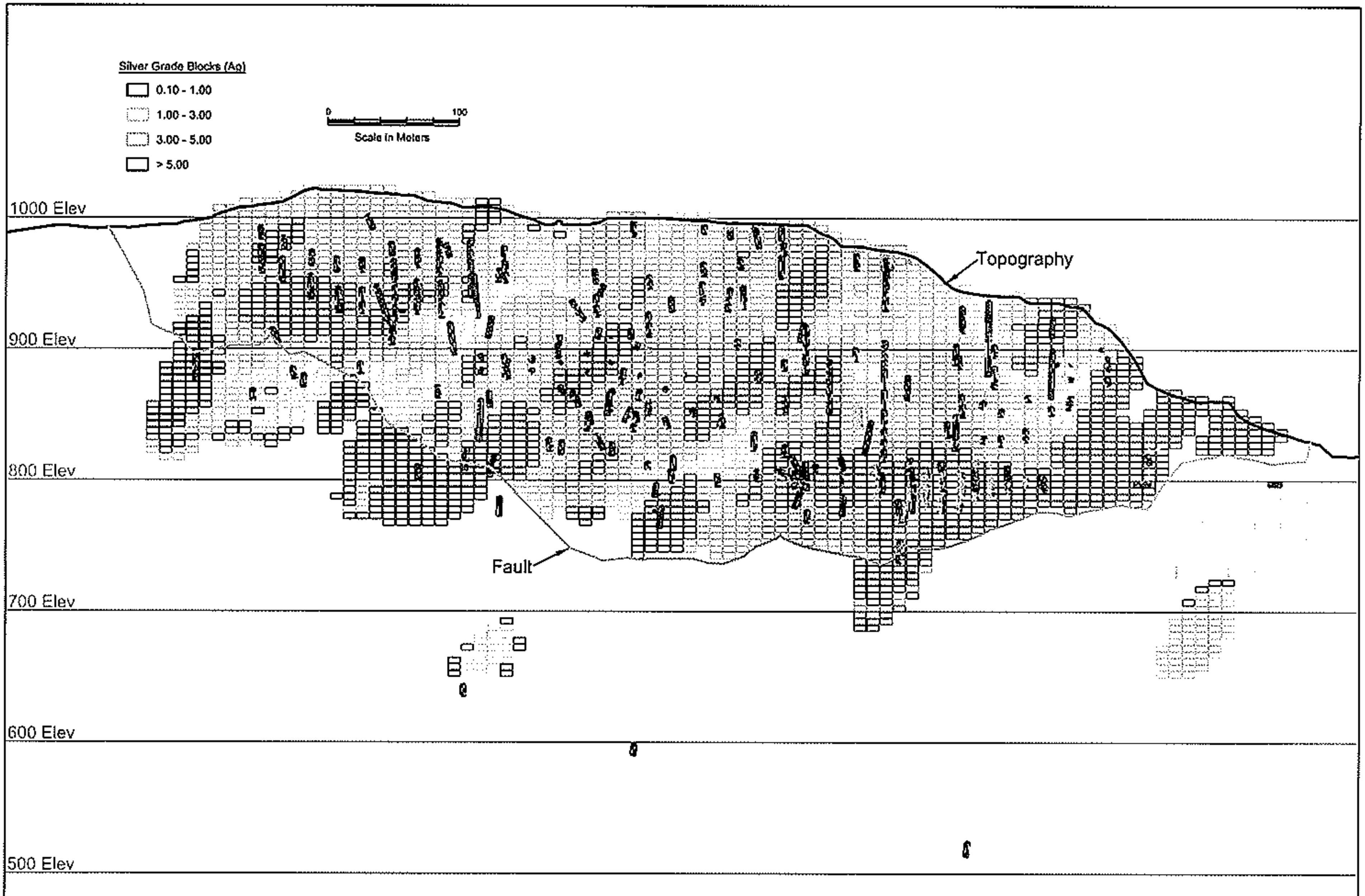
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-40_43.dwg	**Figure 17-40** **East-West Section 435,845 E** **Showing Resource Classification** **(Looking West)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/28/2009	



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-40_43.dwg	Figure 17-41 **East-West Section 435,845 E** **Showing Lithologic Designations** **(Looking West)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/28/2009	



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-40_43.dwg	Figure 17-42 **East-West Section 435,845 E** **Showing Gold Distribution by Grade** **(Looking West)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/28/2009	



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Pinnacle Mines Ltd.*	Fig17-40_43.dwg	Figure 17-43 **East-West Section 435,845 E** **Showing Silver Distribution by Grade** **(Looking West)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 12/28/2009	

TABLE 17-3: SILVER COIN TOTAL CLASSIFIED RESOURCES
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

MEASURED RESOURCES

ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	8,895	1.28	7.04	0.29	365	2,012	55,967
ALL	0.50	5,957	1.73	8.16	0.35	331	1,562	46,569
ALL	0.75	4,308	2.16	8.96	0.40	299	1,241	38,246
ALL	1.00	3,219	2.59	9.64	0.44	268	997	31,140
ALL	1.25	2,505	3.01	10.27	0.47	243	827	26,017
ALL	1.50	2,052	3.38	10.93	0.50	223	721	22,723

INDICATED RESOURCES

ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	18,385	1.02	5.99	0.20	602	3,544	82,522
ALL	0.50	11,811	1.38	6.92	0.25	526	2,627	65,174
ALL	0.75	8,009	1.75	7.54	0.28	451	1,942	49,527
ALL	1.00	5,608	2.13	8.13	0.30	384	1,465	37,511
ALL	1.25	4,073	2.51	8.56	0.32	329	1,121	28,949
ALL	1.50	3,048	2.90	9.17	0.35	284	898	23,297

MEASURED + INDICATED RESOURCES

ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	27,279	1.10	6.33	0.23	967	5,556	138,441
ALL	0.50	17,767	1.50	7.33	0.29	857	4,189	111,750
ALL	0.75	12,317	1.89	8.04	0.32	749	3,184	87,762
ALL	1.00	8,827	2.30	8.68	0.35	652	2,462	68,635
ALL	1.25	6,578	2.70	9.21	0.38	572	1,949	54,962
ALL	1.50	5,101	3.09	9.88	0.41	507	1,620	46,029

INFERRED RESOURCES								
ROCK TYPE	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)				Au (oz)	Ag (oz)	Zn (lb)
ALL	0.25	49,189	0.76	6.60	0.22	1,209	10,433	243,019
ALL	0.50	24,861	1.17	8.50	0.28	937	6,792	154,999
ALL	0.75	15,343	1.52	8.43	0.30	750	4,158	99,920
ALL	1.00	10,380	1.84	9.47	0.33	612	3,160	76,363
ALL	1.25	6,787	2.22	10.89	0.38	484	2,375	57,217
ALL	1.50	5,031	2.51	12.04	0.41	407	1,948	45,508

Virtually the entire known resource is located on 22 claims of the total 26 claims that make up the Silver Coin project. Pinnacle owns 70% (TABLE 17-4) and Mountain Boy Minerals owns 30% of these 22 claims and the known resource. Pinnacle has an option to acquire an additional 10% of the 22 claims (for a total of 80%) by spending CDN$2,000,000 on exploration expenses on or before June 30, 2014. The remaining four INDI claims lie on the eastern edge of the resource and Pinnacle owns 28.05% of these four claims with Mountain Boy owning an additional 26.95% for a total of 55%. Nanika Resources Inc. owning the balance of 45%.

TABLE 17-4: SILVER COIN CLASSIFIED RESOURCES CONTROLLED BY PINNACLE
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

MEASURED RESOURCES								
CLAIM	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
Missouri	0.75	937	2.37	12.01	0.68	71	362	14,142
Kansas	0.75	1,987	2.08	7.55	0.27	133	483	12,031
INDI 9	0.75	67	1.68	8.36	0.33	4	18	436
Total	0.75	2,991	2.16	8.97	0.44	208	863	26,609

INDICATED RESOURCES								
CLAIM	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
Missouri	0.75	1,678	1.84	9.66	0.50	99	521	18,403
Kansas	0.75	3,715	1.71	6.49	0.18	204	775	15,004
INDI 9	0.75	156	1.85	9.27	0.30	9	46	919
TOTAL	0.75	5,549	1.75	7.52	0.31	312	1,342	34,326

MEASURED + INDICATED RESOURCES								
CLAIM	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
Missouri	0.75	2,615	2.03	10.50	0.56	170	883	32,546
Kansas	0.75	5,702	1.84	6.86	0.22	337	1,258	27,040
INDI 9	0.75	222	1.80	9.01	0.30	13	64	1,346
Total	0.75	8,539	1.89	8.03	0.36	520	2,205	60,932

INFERRED RESOURCES								
CLAIM	Cutoff Grade Au (g/t)	TONNES	Avg. Grade			Contained Metal ('000)		
		(000)	Au (g/t)	Ag (g/t)	Zn (%)	Au (oz)	Ag (oz)	Zn (lb)
Missouri	0.75	3,494	1.70	10.26	0.55	191	1,153	42,509
Kansas	0.75	4,890	1.45	6.12	0.12	228	963	13,259
INDI 9	0.75	1,717	1.40	10.49	0.30	78	579	10,329
Total	0.75	10,101	1.53	8.30	0.33	497	2,695	66,097

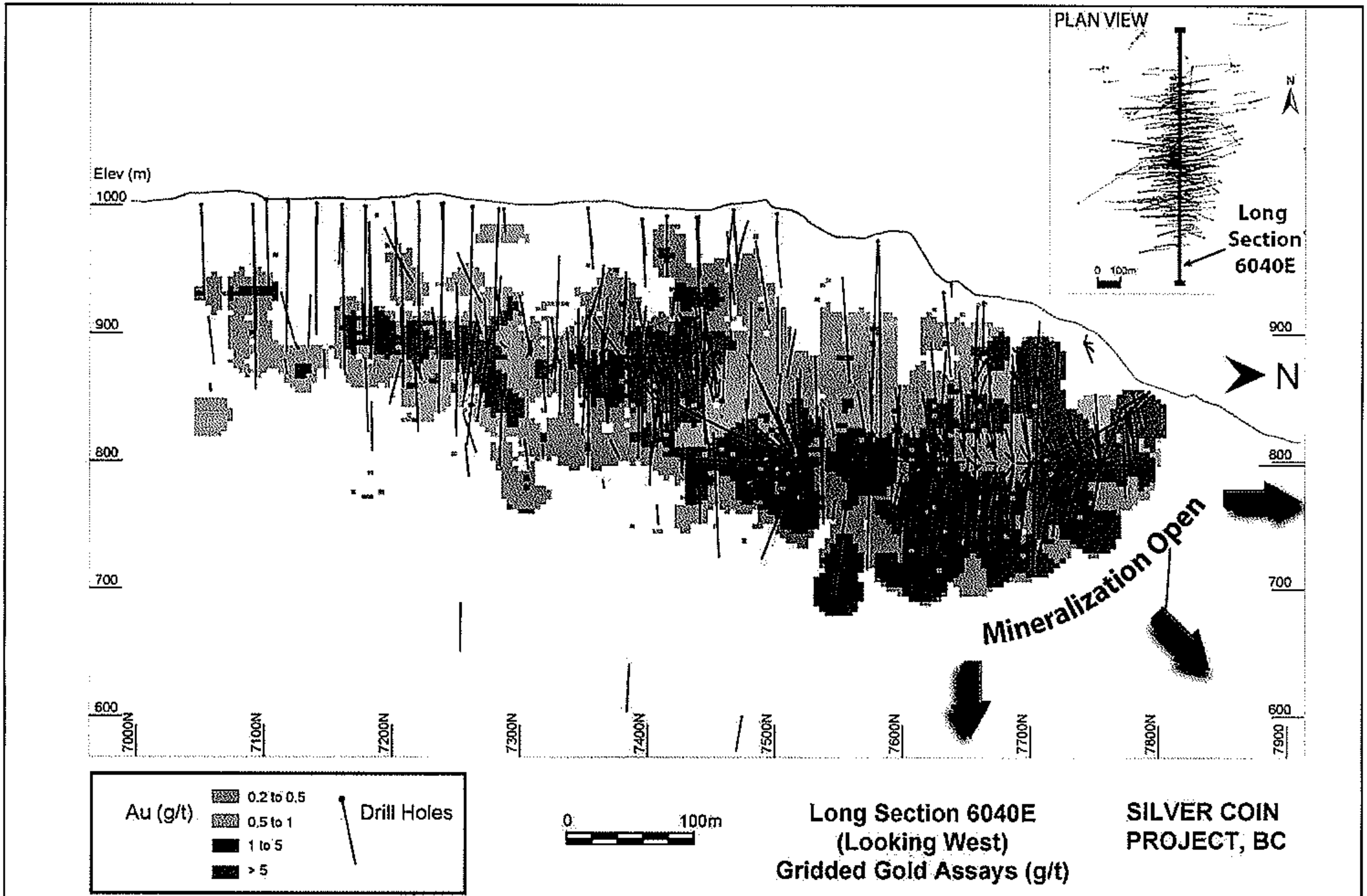
17.13 Resource Expansion Potential

As noted above, Pinnacle does not expect the identified mineralization at Silver Coin to project to depth immediately below the existing resource because it is believed to be related to a localized extensional basal fault within a shear couple that appears to limit the depth of mineralization. However, the package of favorable rocks extends both north and south of the resource and remains prospective and little tested. Additionally, there are a series of significant north-trending faults both east and west of the deposit and Pinnacle speculates that any of these environments might generate a similar dilational zone and potential for discovery of significant new mineralization.

Thick intervals of strong gold mineralization encountered in 2008 drilling near the south end of the deposit (for example, hole # SC08-243, 42.6m @ 4.14 gm/t Au) indicate additional potential. Although enough drilling exists in this area to limit a major expansion of mineralization to the immediate south of the deposit, when mined, this area may yield important zones of higher grade gold.

In the immediate vicinity of the Silver Coin deposit excellent potential exists to add resources to the north and northwest of the existing resource as shown in FIGURE 17-44. This is an area that is known to host good gold mineralization in trenches but has not been extensively explored due to precipitous topography and the fact that the bedrock is friable and very difficult to obtain core by means of diamond drilling. The gold mineralization apparently bends toward the west, following the trace of the Anomaly Creek fault.

Because the topography is dropping to the west, the amount of overburden remains low, and the grade of gold appears to remain at or better than the deposit average. The westward bend in the Anomaly Creek fault may have influenced the stress field and potentially enhanced the environment for deposition of gold.



Issued by: TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Drawing Provided by / Prepared for: ***Pinnacle Mines Ltd.***	File Name: Fig17-44.cdr	**Figure 17-44** **Long Section 6040E (Looking West)**
	Project: Silver Coin Gold Project	Project Number: 114-311007	
	Project Location: Stewart, British Columbia	Date of Issue: 01/06/2010	

18.0 POTENTIALLY MINEABLE RESOURCES

Silver Coin contains no mineral reserves as defined by CIMM standards. All categories of the estimated mineral resources - Measured (M), Indicated (I), and Inferred (I), have been used in the determination of potentially mineable mineral resources. All categories have been used in developing production schedules and preliminary cash flow analyses.

The potentially mineable resources are developed from open pit mining scenarios. The potentially mineable resource estimates were derived from 3D grade and geologic block models developed by Tt as described in SECTION 17.

Tt's review of these resources includes assessment of a potential development of a 3.5 million tonne-per-year operation.

18.1 Whittle Pit Design Parameters

To guide design of an ultimate pit and to determine the best extraction sequence, Tt utilized floating cones and the Whittle algorithm to establish guides to mineable shapes within the mineral resource block model. The ordinary kriging estimate of total gold in the model was imported to Gemcom's® Whittle® mine optimization software. Estimates for mine and processing costs were prepared from a comparison of actual and handbook costs for similar operations.

For the Silver Coin Gold Project, two potential operations are being considered. One that involves creating a bulk sulfide flotation concentrate that is shipped to Asia for smelting and one that involves flotation followed by cyanidation on site that produces a precious metals dore. TABLES 18-1 and 18-2 list the input parameters used for the LG cone runs for these two potential development scenarios. No additional constraints such as property boundaries or existing infrastructure locations were applied to the preliminary Whittle run. The average achievable pit slope was estimated at 45°. Slope measurements on historical benches are approximately 45° in several areas of the existing pit. The gold price is based on the 3-year trailing average gold price.

TABLE 18-1: WHITTLE LG PARAMETERS – ALL FLOTATION SCENARIO
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

Parameter	Units	Value
Average Pit Slopes	Degrees	45
Metal Price (3-year average)		
Gold	US$/t. oz	833
Silver	US$/t. oz	14.24
Metal Produced		
Gold	%	95
Silver	%	88
Mining Cost	$US/tonne mined	2.55
Processing Cost	$US/tonne milled	7.42
Freight & Refining	$US/ounce gold	25
General & Administrative Costs	$US/tonne milled	1.33
Environmental & regulatory Costs	US$/tonne milled	0.25

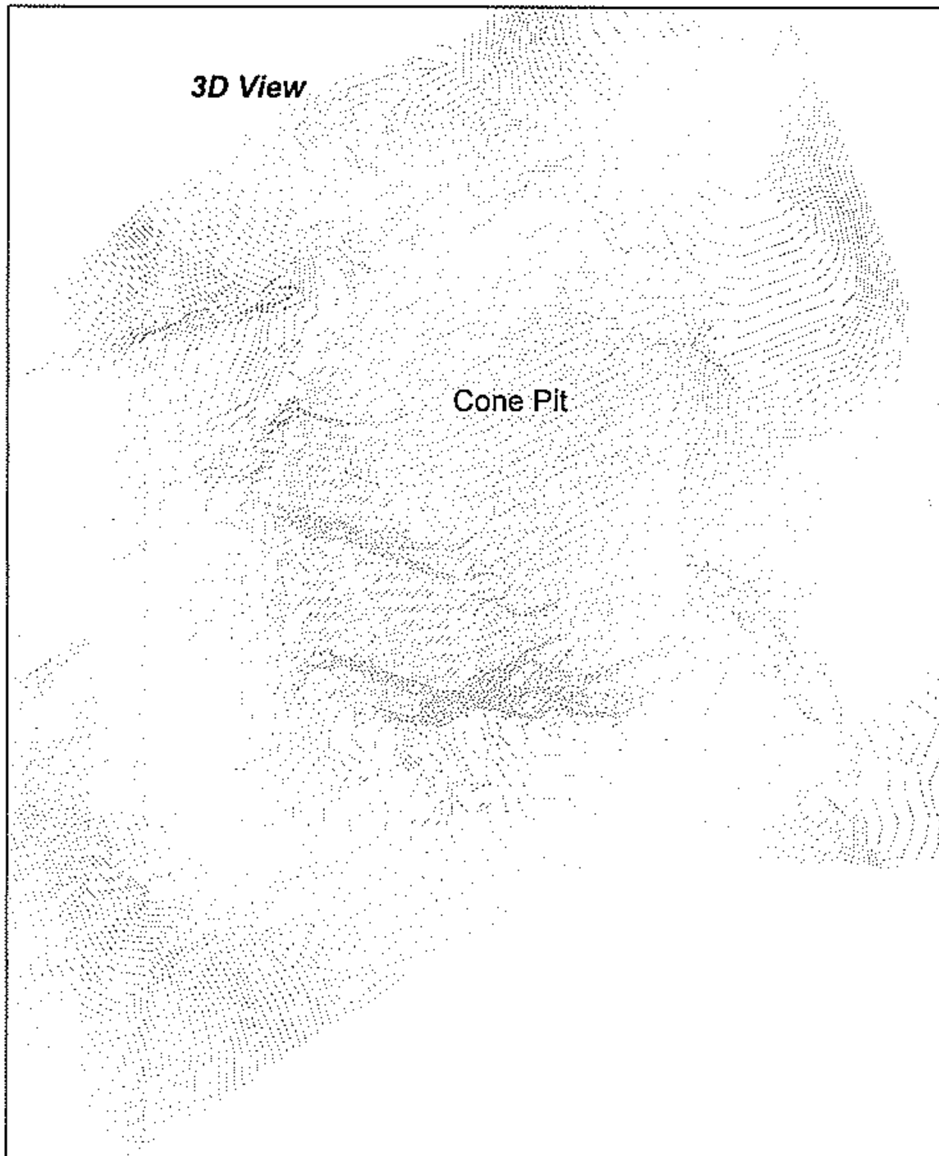
TABLE 18-2: WHITTLE LG PARAMETERS – FLOTATION - CYANIDATION SCENARIO PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009		
Parameter	**Units**	**Value**
Average Pit Slopes	Degrees	45
Metal Price (3-year average)		
Gold	US$/t. oz	833
Silver	US$/t. oz	14.24
Metal Produced		
Gold	%	88
Silver	%	60
Mining Cost	$US/tonne mined	2.55
Processing Cost	$US/tonne milled	8.42
Freight & Refining	$US/ounce gold	10.00
General & Administrative Costs	$US/tonne milled	1.33
Environmental & regulatory Costs	US$/tonne milled	0.25

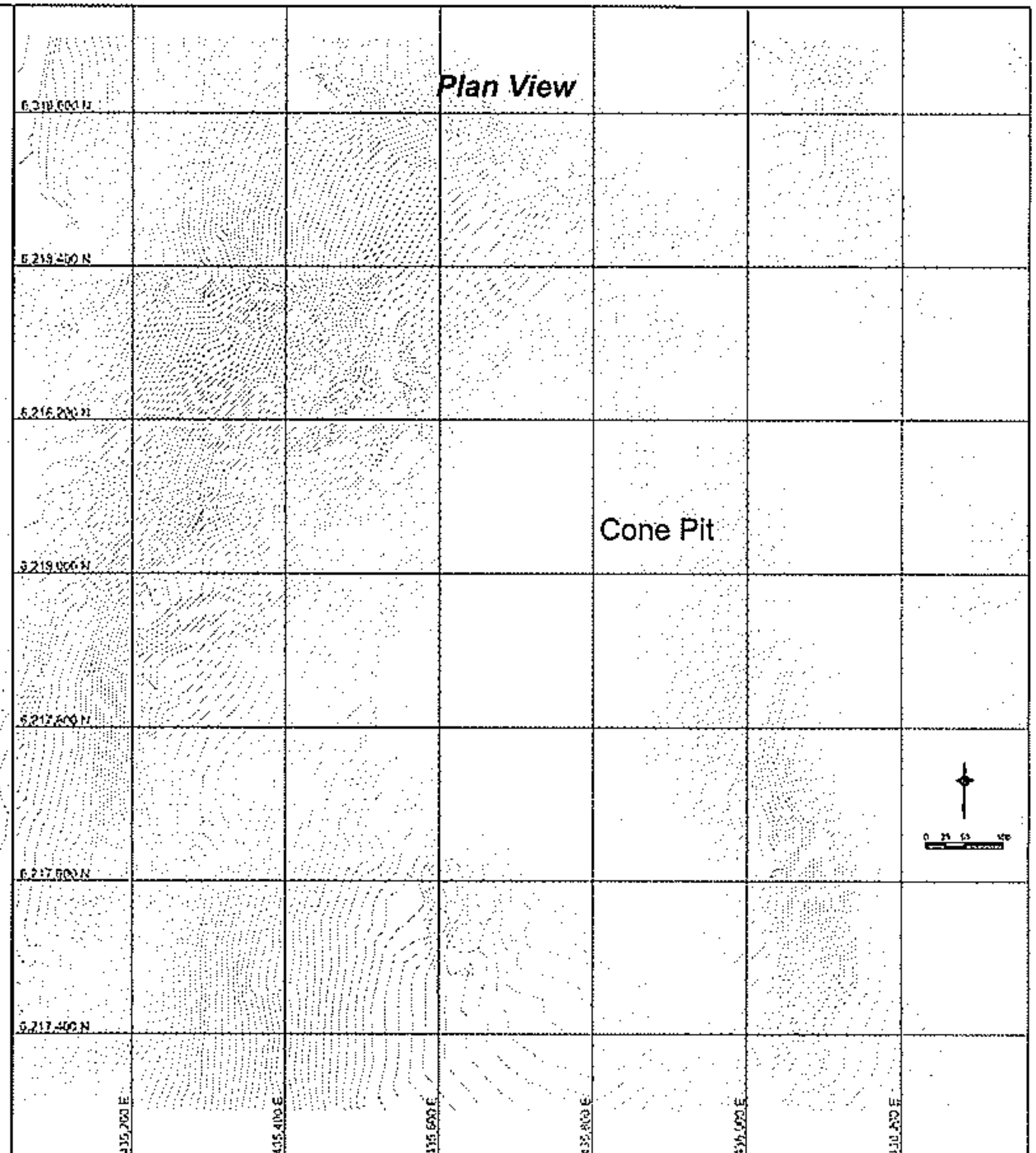
18.1 Potentially Mineable Resources and Production Scheduling

TABLES 18-3 and 18-4 summarize the results of the two Whittle LG scenarios. FIGURES 18-1 and 18-2 show the Whittle pit shapes for each scenario. For this PEA, the ore production rate was set at 3,500,000 ore tonnes per year or approximately 10,000 ore-tonnes per day. A one-year build up is expected with Year one ore production set at 3,500,000 tonnes and 6,354,000 tonnes of waste. Subsequent years will continue to produce 3,500,000 ore tonnes through year 15 and have waste tonnes dropping to approximately 2,000,000 tonnes in year 15.

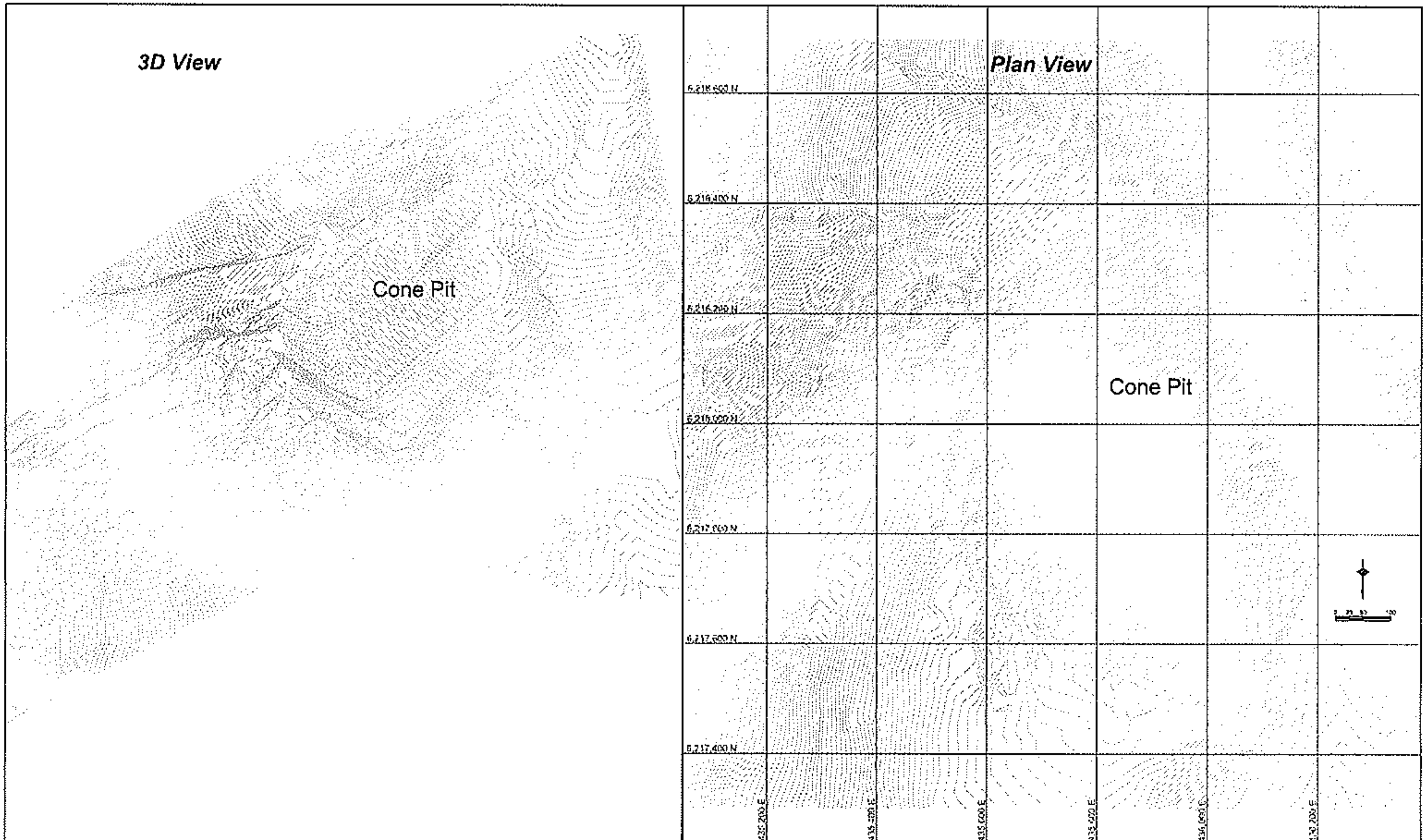
TABLE 18-3: POTENTIALLY MINEABLE RESOURCES – ALL FLOTATION SCENARIO PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009						
Ore Tonnes	**Avg. Metal Grades**			**Waste Tonnes**	**Total Tonnes**	**Stripping Ratio**
('000)	**Au (g/t)**	**Ag (g/t)**	**Zn (%)**	**('000)**	**('000)**	**(W:O)**
54,173	0.99	7.23	0.27	65,786	119,959	1.21:1

TABLE 18-4: POTENTIALLY MINEABLE RESOURCES – FLOTATION - CYANIDATION SCENARIO PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009						
Ore Tonnes	**Avg. Metal Grades**			**Waste Tonnes**	**Total Tonnes**	**Stripping Ratio**
('000)	**Au (g/t)**	**Ag (g/t)**	**Zn (%)**	**('000)**	**('000)**	**(W:O)**
42,840	1.13	7.82	.30	55,808	98,649	1.3:1





Issued by: TETRA TECH	Prepared for: *Pinnacle Mines Ltd.*	File Name: Fig18-1.dwg	Figure 18-1
	Project: Silver Coin Project	Project Number: 114-311007	Whittle® LG Gold-Silver Pit Shape
	Project Location: Stewart, British Columbia	Date of Issue: 01/06/2010	All Flotation Process



TETRA TECH

Prepared for: *Pinnacle Mines Ltd.*

Project: Silver Coin Project

Project Location: Stewart, British Columbia

File Name: Fig18-2.dwg

Project Number: 114-311007

Date of Issue: 01/05/2010

Figure 18-2
Whittle® LG Gold-Silver Pit Shape
Flotation - Cyanidation Mill

19.0 OTHER RELEVANT DATA AND INFORMATION

Neither Mr. Perry nor Tt is aware of any additional information that would have any material impacts and/or changes to the data presented, recommendations made, or conclusions presented in this TR.

20.0 INTERPRETATION AND CONCLUSIONS

20.1 Interpretation

It is Tt's opinion that most of the past work and all of the current Pinnacle work meets and/or exceeds the current standards and those areas that do not meet current standards have been identified within the body of this report. The work has been completed by well-qualified technical professionals, reputable mining companies, and independent third-party contractors and laboratories according to standards that meet most of today's requirements. The results of the 2004 through 2008 drilling, and assay geostatistical study provide strong support that the current geologic model and resource estimates are truly indicative of the mineralization at Silver Coin.

20.2 Conclusions

It is Tt's opinion that the data used in support of and for the estimation of the geologic resources quoted in this Preliminary Economic Assessment Report are compliant with CIMM definitions and that the geologic resources presented meet the requirements of measured, indicated, and inferred resources under current CIMM definitions. The capital and operating cost estimates are within the normal levels of accuracy for a Preliminary Economic Assessment (+/- 35 to 50 percent), with many of the costs exceeding these limits as Tt had access to data from other similar mines that are currently operating.

21.0 RECOMMENDATIONS AND WORK PLAN

To continue evaluation of the Silver Coin Gold Project, Tt and Mr. Perry recommend that Pinnacle undertake several additional investigations. These investigations will be required to allow the project to proceed toward feasibility level evaluation.

21.1 Recommended Additional Investigations

Studies are anticipated in eight key areas of investigation as follows:

Geology and Resources: additional drilling will be needed for

- Confirmation of the geology, mineralogy, and ore types
- Development of geotechnical parameters for pit slopes
- Determination of the hydrology of the area around the planned open pits
- Collection of additional metallurgical samples for testing

Mine Planning: development of an updated mine plan based on current gold pricing and updated costs. This work would include:

- Calculation of cut off grade
- New pit designs with haulage access
- End-of-year period plans
- End of six month plans for the first two years
- End of year plans through Year 5
- End of mine life plan
- Waste rock facilities for life of mine ultimate foot print
- Mine production schedule
- Mine equipment requirements
- Manpower requirements
- Capex and Opex

Metallurgical Testwork: Tt recommends additional testing including:

- Bond crusher impact and abrasion testing
- Bond Ball mill work indices on drill core and variability samples.
- JkTech drop weight (DWi) testing and/or SMC SAG mill testing
- Gravity pre-concentration studies
- Flotation Studies on representative and variability composites to evaluate
    - Grind
    - Collector types and dosages
    - Flotation kinetics
    - Regrind and cleaner flotation
    - Locked-cycle flotation

- Cyanidation studies on flotation cleaner concentrates to evaluate
    - Retention time
    - Cyanide Consumption
    - Lime Consumption
    - Preg-robbing characteristics
    - Regrind fineness
    - Slurry density
    - Cyanide destruction of cyanidation leach residue
- Thickening Studies on:
    - Flotation and cyanidation tailings
    - Flotation concentrates

Process Design: Tt recommends that additional work to further advance components of the proposed process design. Investigations should include:

- Plant design capacity should be reviewed based on revisions to ore reserves
- Review and validation of design criteria parameters.
- Major equipment list and associated power requirements.
- Development of design drawings.
- Development of process design flowsheets and drawings to include the following:
    - Process Flowsheets and P&IDs
        - Crushing Plant
        - Grinding Circuit
        - Flotation Circuit
        - Concentrate Cyanidation
        - Thickening and Filtration
        - Utilities
    - Site Plans
        - Overall
        - Crushing Plant
        - Flotation-Cyanidation Plant
    - General Arrangements
        - Crushing plant plans and sections
        - Grinding circuit plans and sections
        - Flotation circuit plans and sections
        - Cyanidation circuit plans and sections
        - Truck shop plans and section

- Fueling station plan and section

Tailings, Ponds and Waste Rock Facility: Tailing Dam and pond design studies must be completed as part of future feasibility analysis. Key elements should include:

- Geotechnical site investigation and creation of a test-pit plan for the area of the tailing dam.
- Liner evaluation
- Large scale direct shear laboratory tests for slope stability and liner design system
- Clay borrow source site investigation
- Hydrologic study – develop storm events using local weather stations (if more than one available) for use in the heap water balance, pond sizing, and stormwater control.
- Acid rock drainage control (if necessary)

Infrastructure: Investigations should be completed to assess major infrastructure requirements including:

- Power supply
- Water supply wells or wellfield including drilling, testing, construction, and pipeline distribution
- Sanitary waste disposal facilities

Environmental Permitting-related Studies: Several environmental permits (Section 4.3) will be required for development of the project. To advance the project towards permitting, it is recommended that baseline environmental studies are initiated as soon as possible since long-term (seasonal-based) data collection may be required. Studies will include:

- Land use
- Air quality
- Geologic resources and rock characterization (e.g. potential for ARD generation)
- Paleontologic resources
- Surface water and groundwater resources
- Soils
- Vegetation
- Wildlife and fisheries, including special status species
- Range resources
- Recreation
- Auditory resources
- Visual resources
- Cultural resources
- Native cultural values
- Hazardous materials

- Socio-economics
- Environmental justice

These studies will be directed towards development of an Environmental Assessment (EA) and/or Environmental Impact Study (EIS), but will also support development of other required environmental and operational permits.

Closure Studies: A reclamation and closure plan will need to be developed to assure long-term environmental stability.

21.2 Work Plan

Pinnacle's future plans include reducing drillhole spacing, preliminary metallurgical testwork, initiating mine planning and baseline environmental studies, continued surface geologic mapping, and securing adequate supplies of water and power. These items are required for the project to proceed toward feasibility.

TABLE 21-1 details the anticipated work plan and major categories of expenditure.

TABLE 21-1: PROPOSED BUDGET FOR PLAN OF WORK
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

Task	Estimated Completion Date*	Estimated Cost (US$) to Complete*	Notes
Development Drilling	October 2010	$500,000	
Exploration Drilling		$500,000	
Gold distribution and Assay Study		$25,000	
Hydrologic Study		$40,000	
Environmental Studies		$150,000	
Site Geology		$40,000	
Metallurgical Testing		$50,000	
Power Study		$15,000	
Water Supply Study		$10,000	
Pit Slope Geotechnical Study		$30,000	
Revised Economic Study		$100,000	
Total – Overall Budget		**1,460,000**	

* Completion dates and expenditures represent minimum programs based on depressed economic and market conditions and are subject to the availability of funding.

Tt and Mr. Perry have reviewed these costs and timelines and believes that they represent the next logical progression in the redevelopment of the Silver Coin Gold Project and that they reflect realistic estimates of the costs to complete the work plan identified.

22.0 REFERENCES CITED

Alldrick, D.J. (1988), Title: Detailed Stratigraphy of the Stewart Mining Camp; *in* Precious Metal Deposits of the Stewart Mining Camp, Geological Association of Canada, Field Trip Guidebook C, 15 pages.

Alldrick, 1993, Title: Geology and Metallogeny of the Stewart Mining Camp, Northwestern British Columbia, B.C Ministry of Energy, Mines and Petroleum Resources, Bulletin 85, (Bulletin, ISSN 0226-7497 ; 85), 113 pages.

Armstrong, 1988, Title: Mesozoic and Early Cenozoic Magmatic Evolution of the Canadian Cordillera; Geological Society of America, Special Paper 218, pages 55-91.

Bitterroot Group LLC, 2009, Pre-scoping Study, Silver Coin Project, Stewart British Columbia, Internal Unpublished Report, 93pages.

British Columbia Ministry of Energy, Mines and Petroleum Resources, 2009, "Appendix E Deposit Type/Mineral Deposit Profiles", http://www.empr.gov.bc.ca/Mining/Geoscience/MineralDepositProfiles/Pages/default.aspx

Britten, 1988, Surface geology map of SB property

Cambria Gordon, 2009, Preliminary Environmental Baseline Report, Silver Coin Project, Unpublished internal report, 26 pages plus appendix.

Cambria Gordon, 2009, Letter regarding Consideration in the Use of Cyanide for Gold Extraction; Exploration Activity Reclamation Liabilities, Silver Coin Property, Stewart, B.C., 9 pages plus attachments.

Greig, C.J., Anderson, R.G., Daubeny P.H., Bull K.F., and Hinderman T.K., 1994a, Title: Geology of the Cambria Icefield: regional setting for Red Mountain Gold Deposit, Northern British Columbia: in Current Research 1994-A: Geological Survey of Canada, p. 45-56.

Greig, C.J., Anderson, R.G., Daubeny P.H., Bull K.F., and Hinderman T.K., 1994b, Title: Geology of the Cambria Icefield: Stewart (103P/13), Bear River (104A/4), and parts of Meziadin Lake (104A/3 and Paw Lake (103P/14) map areas, northwestern British Columbia; Geological Survey of Canada, Open File 2931, 4 sheets (3 maps, 1 table).

Grove, E.W. (1971), Title: Geology and Mineral Deposits of the Stewart Area, British Columbia; B.C. Ministry of Energy, Mines and Petroleum Resources, Bulletin 58, 229 pages.

Grove, E.W., 1981, Title: Geological report on The Silver Coin Group-Salmon River District

Grove, E.W. (1986), Title: Geology and Mineral Deposits of the Unuk River - Salmon River - Anyox Area; B.C. Ministry of Energy, Mines and Petroleum Resources, Bulletin 63, 152 pages.

Lhotka P. G., Johnston R., and Rockingham C. J., 1991, Title: Kansas/West Kansas Geological Inventory

Kruchkowski, 2005, 2004 Exploration Summary on Silver Coin Property

Kruchkowski, 2006, 2005 Exploration Summary on Silver Coin Property

Kruchkowski, 2007, Assessment Report on Drilling, Trenching, and Geochemical Program on Silver Coin Property, Stewart, British Columbia, for Pinnacle Mines, Ltd., Vancouver, B.C., and Mountain Boy Minerals, Ltd., Stewart, B.C., 947 pages (206 report pages with figures and appendices)

Mahoney, J. B., Gordee, S., Haggart, J. W., Friedman, R. M.,Diakow, L. and Woodsworth, G. J., 2006, Jurassic-Tertiary evolution of the central Coast Plutonic Complex: Controls on over 120 million years of magmatism: Geol. Soc. America, Abstracts with Programs, April 2006.

Malott, M.L., 1988, BCMEMPR-Exploration in British Columbia, pages B145 – B152

Mazur, S., 2006, Title: Report on the Structural Mapping of the Silver Coin Property, Stewart, British Columbia, for Pinnacle Mines, Ltd., Vancouver, B.C., and Mountain Boy Minerals, Ltd., Stewart, B.C., 11 pages.

Melnyk and Britten, 1989, Referenced in Stone and Godden, 2007. Mentioned in JFC structure section, Kansas/West Kansas section, probably taken directly from Stone and Godden, 2007.

Payne, 1988, Petrographic study of thin sections from SB property

Pennstrom, William, April 2009, Review of the Metallurgical Study of the Silver Coin Gold Deposit, Memorandum to Patrick Highsmith after reviewing a report by Frank Wright, 8 pages

Snowden, 2008, DRAFT Technical Report – Pinnacle Mines Ltd: Silver Coin Project, Stewart, British Columbia, Canada, Chapter 17 (Full Report Not Available), 15 Pages

Stone, D.M.R. and Godden, S.J., 2007, Title: Updated Technical Report and Preliminary Economic Assessment on the Silver Coin Property, Stewart, British Columbia, Minefill Services, Inc. for Pinnacle Mines, Ltd., Vancouver, B.C., 160 pages.

Walus, A., 2009, Preliminary Feasibility Study for the Silver Coin Project, uncompleted DRAFT Report, March 2009, 67 pages.

Wright, F.W., 2009, Metallurgical Study on the Silver Coin Gold Project, 247p.

23.0 DATE AND SIGNATURE PAGE

Robert Perry

Independent Consultant

16622 West 56th Dr.
Golden, Colorado 80403
Telephone: 303-717-2213

Email: bperrygeo@gmail.com

CERTIFICATE of AUTHOR

I, Robert Perry, P.G., do hereby certify that:

1. I am currently an independent geological consultant and that I reside at:

   16622 West 56th Dr
   Golden, Colorado 80403

2. I graduated with a degree in Geology (BA.) from the University of Colorado, in Boulder, Colorado in 1973. In addition, I graduated from the University of Colorado, Boulder, Colorado, with a graduate degree in Geology (M.Sc.) in 1976.
3. I am a Member of the American Institute of Professional Geologists (CPG-11074), and the Society of Economic Geologists.
4. I have worked as a geologist for a total of thirty years since my graduation from university; as a graduate student, as an employee of both public and private mining and exploration companies and as a consultant.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of the majority of the sections of this technical report titled "*SILVER COIN GOLD PROJECT, PRELIMINARY ECONOMIC ASSESSMENT REPORT, STEWART, BRITISH COLUMBIA, CANADA.*" and dated 30 December 2009 (the "Preliminary Economic Assessment Report"). I have visited the subject property on April 20-22, 2009, and August 17-19, 2009.
7. I have either supervised the data collection, preparation, and analysis and/or personally completed an independent review and analysis of the data and written information contained in this Preliminary Economic Assessment.
8. I have worked as an independent consultant to Pinnacle Mines on the Silver Coin Gold Project that is the subject of this Technical Report.
9. I am not aware of any material fact or material change with respect to the subject matter of the Preliminary Economic Assessment that is not reflected in the Preliminary Economic Assessment, the omission to disclose which makes the Preliminary Economic Assessment misleading.
10. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to

the best of my knowledge do I have any interest in any securities of any corporate entity with property within a two Km distance of any of the subject properties.

11. I have read National Instrument 43-101 and Form 43-101F, and the Preliminary Economic Assessment has been prepared in compliance with that instrument and form.
12. I consent to the filing of the NI 43-101 Preliminary Economic Assessment with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Updated Resource NI 43-101 Technical Report.

Dated this 30th Day of December, 2009

Signature of Qualified Person

"Robert Perry"

Print name of Qualified Person

John W Rozelle, P.G.

Principal Geologist
TETRA TECH MM, INC.

350 Indiana Street, Suite 350
Golden, Colorado 80401
Telephone: 303-217-5700
Facsimile: 303-217-5705
Email: john.rozelle@tetratech.com

CERTIFICATE of AUTHOR

I, John W. Rozelle, P.G., do hereby certify that:

13. I am currently employed by Tetra Tech MM, Inc. at:

    350 Indiana Street
    Suite 350
    Golden, Colorado 80401

14. I graduated with a degree in Geology (BA.) from the State University of New York at Plattsburg, New York, in 1976. In addition, I graduated from the Colorado School of Mines, Golden, Colorado, with a graduate degree in Geochemistry (M.Sc.) in 1978.
15. I am a Member of the American Institute of Professional Geologists (CPG-07216), a register Geologist in the State of Wyoming (PG-337), a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) and the Society of Economic Geologists.
16. I have worked as a geologist for a total of thirty-one years since my graduation from university; as a graduate student, as an employee of a major mining company, and as a consultant for more than 30 years.
17. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
18. I am responsible for and/or have supervised the preparation of specific sections (Sections 1.0, 16.0, 17.0, 18.0, and 24.0) of the technical report titled "*SILVER COIN GOLD PROJECT, PRELIMINARY ECONOMIC ASSESSMENT REPORT, STEWART, BRITISH COLUMBIA, CANADA*" and dated 30 December 2009 (the "Preliminary Economic Assessment Report"). I have visited never visited the subject property.
19. I have either supervised the data collection, preparation, and analysis and/or personally completed an independent review and analysis of the data and written information contained in this Preliminary Economic Assessment.
20. I have had no prior involvement with Silver Coin Gold Project and/or Pinnacle Mines Ltd. on the property that is the subject of this Technical Report.
21. I am not aware of any material fact or material change with respect to the subject matter of the Preliminary Economic Assessment that is not reflected in the Preliminary Economic Assessment, the omission to disclose which makes the Preliminary Economic Assessment misleading.
22. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of

this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a two (2) Km distance of any of the subject properties.

23. I have read National Instrument 43-101 and Form 43-101F, and the Preliminary Economic Assessment has been prepared in compliance with that instrument and form.
24. I consent to the filing of the Updated Resource NI 43-101 Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Updated Resource NI 43-101 Technical Report.

Dated this 30th Day of December, 2009

________________________.
Signature of Qualified Person

"John W. Rozelle" .
Print name of Qualified Person

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

24.1 Base Case Mining Operation

The Silver Coin Gold Project will be mined using conventional open pit methods utilizing off-highway trucks and front-end loaders. The mine pit designs were based on the floating cone algorithm using Whittle's software.

Mining at Silver Coin will utilize conventional open pit practices, encompassing the drill, blast, load, and haul functions to remove both overburden waste and mineralized material. Waste is to be hauled from the pit to an outside disposal site which will have been previously prepared to accept the barren tonnage. Economic tests have been performed on the deposit to gauge the largest expected pit, and from this the waste location has been sited well beyond the hypothetical pit rim. Excess ground pressure from stacking the waste is not expected to impact pit wall stability.

Overburden removal (along with any occasional ore uncovered) is planned to begin in Year -1, the first year prior to official start-up. Any mineralized material will be taken to a stockpile area near the processing facility, and depending on grade, may be processed in the initial year of operations or may be delivered to the circuit in the final production year when pit deliveries will be declining. By Year 1 (the initial year of production) a full complement of mining equipment and related facilities will need to be in place as the anticipated production of mined material will approach its maximum for the project.

Production levels are expected to be in the range of 12 million tonnes annually as a maximum, with 3.5 million tonnes of ore delivered to the process plant each year. Waste production is reasonably uniform through Year 4 of operations at +/- 6 - 8 million tonnes per year, at which point the waste movement gradually declines to 0.6 million tonnes in the 13th year of production. The mine life extends for the one pre-production year during which the pit is being pre-stripped, and then encompasses ore production for the following 13 years. Material quantities handled during the mining phase totals approximately 43 million ore tonnes, and just under 56 million waste tonnes, for an overall 1.30:1 stripping ratio. TABLE 24-1 presents the annual quantities of material mined from the pit.

24.1.1 Pit Parameters and Design

Material quantities were developed for the pre-production year, and the following productions years. Truck speeds and round trip travel times for the haulers were estimated without benefit of defined haulage profiles so that an annual assessment could be made for projecting equipment hours, the number of units required, and both capital and operating costs.

24.1.2 Equipment Requirements

This schedule of material quantities served as the base starting point from which to calculate the primary equipment requirements. Primary equipment represents those units which are dependent upon production, as opposed to secondary items that are subordinate to the production equipment and are estimated based on historical practice.

The drills for Silver Coin are scheduled to operate around the clock, and thereby need four crews to cover all shifts. Different production rates have been incorporated because of varying parameters between the ore and waste rock, and allowances have been provided to account for employee breaks during each shift, travel time between hole locations, mechanical availability, and utilization of the equipment. TABLE 24-2 presents the summary data on drill requirements

TABLE 24-1: SILVER COIN GOLD PROJECT – PRODUCTION SCHEDULE PINNACLE MINE, LTD. – SILVER COIN GOLD PROJECT December 2009						
Year	Ore Tonnes	Waste Tonnes	W:O Ratio	Ag Grade (g/t)	Au Grade (g/t)	Zn Grade (%)
ALL FLOTATION PRODUCTION SCHEDULE						
-1	0	6,354,588	3.86	0.00	0.00	0.00
1	3,500,000	7,821,267	3.59	6.91	0.74	0.20
2	3,500,000	8,615,404	2.55	5.79	0.72	0.29
3	3,500,000	5,839,198	1.67	5.53	0.78	0.20
4	3,500,000	4,680,584	1.34	5.36	0.83	0.13
5	3,500,000	4,135,327	1.18	5.83	0.85	0.13
6	3,500,000	3,559,583	1.02	6.30	0.97	0.15
7	3,500,000	2,993,320	0.86	6.48	1.00	0.18
8	3,500,000	2,547,391	0.73	6.93	1.07	0.21
9	3,500,000	2,363,730	0.68	7.68	1.03	0.26
10	3,500,000	2,452,669	0.7	8.50	0.98	0.32
11	3,500,000	2,561,077	0.73	9.20	0.93	0.39
12	3,500,000	2,719,803	0.78	8.60	0.90	0.41
13	3,500,000	3,314,511	0.95	7.74	0.96	0.35
14	3,500,000	3,041,459	0.87	9.08	1.26	0.42
15	3,500,000	2,133,876	0.61	8.05	1.70	0.37
16	1,672,809	652,602	0.45	8.22	1.24	0.25
TOTAL	**54,172,809**	**65,786,389**	**1.21**	**7.23**	**0.99**	**0.27**
FLOTATION – CYANIDATION PRODUCTION SCHEDULE						
-1	0	6,562,839	4.57	8.07	0.84	0.20
1	3,500,000	9,500,000	4.21	6.22	0.83	0.29
2	3,500,000	8,500,000	2.56	5.50	0.90	0.20
3	3,500,000	7,000,000	1.52	5.56	0.95	0.13
4	3,500,000	4,834,690	1.21	6.15	1.02	0.14
5	3,500,000	3,143,011	0.9	6.61	1.14	0.18
6	3,500,000	2,362,709	0.68	7.24	1.22	0.22
7	3,500,000	1,959,284	0.56	8.33	1.17	0.29
8	3,500,000	2,027,618	0.58	9.52	1.07	0.38
9	3,500,000	2,252,378	0.64	10.13	1.00	0.47
10	3,500,000	2,756,895	0.79	8.79	1.04	0.44
11	3,500,000	2,857,434	0.82	10.14	1.38	0.49
12	3,500,000	1,982,629	0.57	9.16	1.79	0.40
13	840,302	68,991	0.32	9.17	1.05	0.15
TOTAL	**43,840,302**	**55,808,478**	**1.3**	**7.83**	**1.12**	**0.30**

TABLE 24-2: DRILL REQUIREMENTS
PINNACLE MINE, LTD. – SILVER COIN GOLD PROJECT
December 2009

7

Explosives Price = US1,060/tonne

ORE				WASTE			
PRIMARY DRILL				PRIMARY DRILL			
Expected Annual Tonnage	=	4000000	tonnes	Expected Annual Tonnage	=	8000000	tonnes
In Situ Density	=	2.6	dry tonnes/cu m	In Situ Density	=	2.6	dry tonnes/cu m
Powder Factor	=	0.2	kg/tonne	Powder Factor	=	0.2	kg/tonne
Annual Powder Requirements	=	800000	kg	Annual Powder Requirements	=	1600000	kg
Drill Bit Diameter	=	150	mm	Drill Bit Diameter	=	180	mm
Powder Density (ANFO)	=	800	kg/cu m	Powder Density (ANFO)	=	800	kg/cu m
Volume of Hole Required	=	1000	cu m	Volume of Hole Required	=	2000	cu m
Length of Hole Required, Loaded	=	56617	m	Length of Hole Required, Loaded	=	78635	m
Bench Height	=	6	m	Bench Height	=	12	m
Bench Height + Subgrade	=	6.5	m	Bench Height + Subgrade	=	13	m
Powder Column in Hole	=	4	m	Powder Column in Hole	=	10	m
Annual No. of Holes	=	14154		Annual No. of Holes	=	7863	
Drillhole Pattern	=	4.3	m square	Drillhole Pattern	=	5.7	m square
Length of Hole Required, Drilled	=	92003	m	Length of Hole Required, Drilled	=	102225	m
Scheduled Days/Year	=	350		Scheduled Days/Year	=	350	
Scheduled Hours/Day	=	24		Scheduled Hours/Day	=	24	
Scheduled Hours/Year	=	8400	hours	Scheduled Hours/Year	=	8400	hours
No. of 12-hour Shifts	=	700	shifts	No. of 12-hour Shifts	=	700	shifts
Effective Time/Shift*	=	630	minutes	Effective Time/Shift*	=	630	minutes
Penetration Rate, Instantaneous	=	16	m/hour	Penetration Rate, Instantaneous	=	20	m/hour
Drill Rig Move Time	=	10	%	Drill Rig Move Time	=	10	%
Altitude Derate Factor	=	1		Altitude Derate Factor	=	1	
Length of Hole/Operating Shift	=	151	m/shift	Length of Hole/Operating Shift	=	189	m/shift
Length of Hole/Operating Year	=	105840	m/year	Length of Hole/Operating Year	=	132300	m/year
Mechanical availability	=	90	%	Mechanical availability	=	90	%
Maximum Use of Available Time	=	90	%	Maximum Use of Available Time	=	90	%
Nominal Drilling Time/Shift	=	510	minutes	Nominal Drilling Time/Shift	=	510	minutes
Expected Tonnage/Year	=	3727294	tonnes	Expected Tonnage/Year	=	8386412	tonnes
Operating Hours/Year	=	6389	hours/year	Operating Hours/Year	=	5679	hours/year
Number of Drills Required	=	1.07		Number of Drills Required	=	0.95	
			2.0				
			2	Atlas Copco DM45			

* Allows for 15 minutes at each end of the shift plus two 15-minute breaks, plus a 30-minute lunch period.

such as expected operating hours per year, the number of drills required, and the purchase schedule. Drill replacement is expected in the ninth and tenth year of operations.

Blasting in this report is forecast to be performed by the owner, although many operations employ specific blasting contractors to conduct this activity. The powder factors of ore and waste are somewhat different, and this consideration has been incorporated for projecting annual quantities and costs of explosives.

Loading and hauling are dependent activities which are best analyzed in concert with each other. Hauling typically accounts for 40 percent or more of mine operating costs, and significant attention is given to maximizing the utilization and life of the units. The required number of shovels and trucks on site is seen on the table, along with the purchase schedule for the equipment. A 60,000-hour truck life is believed attainable at Silver Coin, but no units need to be replaced at the property because of the early purchases and declining tonnage. Similarly, the loaders are expected to last for the full extent of mining, without replacement.

24.1.3 Mine Equipment and Facilities Capital

Mine equipment requirements and capital over the project life is presented in TABLE 24-3. This table shows the initial timing of equipment purchases, the replacement timing, unit costs for the items, and annual expected outlay. Specific manufacturers are identified, which may or may not be the ultimate equipment suppliers on the project. Costing data were obtained from Info Mine USA, Inc., and EquipmentWatch for 2009. It is seen that initial mine equipment will be in the range of US$29.3 million, to which has been added an 8-percent allowance for initial spare parts. Total equipment capital over the project life is projected at $38.5 million.

Mine facilities include those structures specifically related to the mining function, such as the mine dry, explosives storage, shop/warehouse, and so forth. These items are estimated to cost $3.6 million, which will be expended in Year -2 in preparation for pre-stripping the following year.

24.1.4 Mine Operating Costs

Unit operating costs for the mining equipment are given in TABLE 24-4; it will be noted that in addition to expected day-to-day field expenses, and allowance for overhaul parts has been included in the total hourly operating cost estimate. This information is then incorporated with expected annual equipment hours to arrive at an annual material and supply cost for the mining portion of the project. The variation in US$/tonne mined and US$/tonne processed during the mine life ranges between an average of $1.29 and $2.92/tonne, respectively.

Labor charges for hourly and salaried personnel have been developed in TABLE 24-5, based on the scheduled operating hours for the various equipment classes, maintenance requirements, and the managerial personnel needed to oversee the operation. Roughly 128 people will be needed during peak production years. Labor costs (in US$) average $1.17/tonne of material moved, and $2.65/tonne processed.

Total direct mine operating costs are therefore seen to be: US$ 2.46/tonne of material

US$ 5.57/tonne processed

All costing has been presented on a constant-dollar basis without allowance for inflationary factors.

TABLE 24-3: MINE CAPITAL SUMMARY PINNACLE MINE, LTD. – SILVER COIN GOLD PROJECT December 2009				
Item	**Make**	**Number**	**Unit Cost (US$)**	**Total Costs (US$)**
Primary Equipment				
Drills	Atlas Copco DM45	4	600,000	2,400,000
Loaders	Cat 992G	2	2,100,000	4,200,000
Trucks	Cat 777F	11	1,250,000	13,750,000
Secondary Equipment				
Dozers	Cat D9T	3	900,000	2,700,000
	Cat D10T	4	1,100,000	4,400,000
	Cat 834H	2	860,000	1,720,000
Graders	Cat 14M	1	450,000	450,000
	Cat 16M	1	670,000	670,000
Water Trucks	10,000 gal	2	650,000	1,300,000
Other Equipment				
Skid Steer	Cat 246C	6	40,000	240,000
Tool Carrier	Cat IT38H	6	170,000	1,020,000
BH Loader	Cat 430E	6	220,000	1,320,000
Light Plant		12	20,000	240,000
Lube/Fuel Truck		1	80,000	80,000
Service Truck		1	70,000	70,000
Powder Truck		1	80,000	80,000
Tire Truck		1	160,000	160,000
Pickup		52	30,000	1,560,000
TOTAL				**38,528,000**

TABLE 24-4: MINE UNIT OPERATING COSTS
PINNACLE MINE, LTD. – SILVER COIN GOLD PROJECT
December 2009

			Field Repair						Major Overhaul		TOTAL	Diesel
	Manufacturer	Model	Parts	Elec/Fuel	Lube	Tires	GEC	Subtotal	Parts	Subtotal	US$	Burn Rate gal/hr
Drills												
	Atlas Copco	*DM45*	21.59	34.56	9.49	-	2.16	67.80	12.36	12.36	80.16	16.3
Excavators												
	Hitachi	*EX1900-6*	39.83	83.53	24.94	-	6.28	154.58	35.89	35.89	190.47	39.4
Loaders												
	Caterpillar	*992G*	20.29	53.64	16.58	29.10	2.65	122.26	18.39	18.39	140.65	25.3
Haul Trucks												
	Caterpillar	*777F*	15.16	42.61	17.30	23.12	-	98.19	24.56	24.56	122.75	20.1
Dozers												
	Caterpillar	*D9T*	23.40	38.16	8.95	-	3.90	74.41	24.02	24.02	98.43	18.0
	Caterpillar	*D10T*	32.67	47.28	12.56	-	5.45	97.96	33.55	33.55	131.51	22.3
Graders												
	Caterpillar	*140H*	5.58	13.36	3.33	3.17	0.45	25.89	6.33	6.33	32.22	6.3
	Caterpillar	*163H*	6.76	15.05	3.94	4.50	0.57	30.82	7.29	7.29	38.11	7.1
	Caterpillar	*834H*	8.17	34.34	7.80	9.98	-	60.29	10.66	10.66	70.95	16.2
Water Trucks												
	Caterpillar	*10,000 gal*	12.30	32.65	7.38	10.41	-	62.74	6.37	6.37	69.11	15.4
Light Plants												
	Onan, other	30-ft tower	0.12	1.27	0.24	0.04	-	1.67	0.11	0.11	1.78	0.6
Service Trucks, Other												
	Lube/Fuel		0.73	9.96	0.65	0.30	-	11.64	0.39	0.39	12.03	4.7
	Service		0.88	9.96	0.72	0.30	-	11.86	0.47	0.47	12.33	4.7
	Tire		2.07	9.96	1.24	0.30	-	13.57	1.11	1.11	14.68	4.7
	Caterpillar	*IT38H*	2.54	11.66	2.57	2.50	0.34	19.61	2.60	2.60	22.21	5.5
	Caterpillar	*430E*	1.89	8.48	1.53	1.45	0.26	13.61	2.00	2.00	15.61	4.0
	Caterpillar	*246C*	0.97	5.72	0.83	0.89	0.08	8.49	1.37	1.37	9.86	2.7
	Pickups		0.57	2.54	0.59	0.44	-	4.14	0.66	0.66	4.80	1.2

* Taken from InfoMine USA, Inc., Oct-2009, or EquipmentWatch, 2nd half, 2009

TABLE 24-5: HOURLY AND SALARIED LABOR SUMMARY PINNACLE MINE, LTD. – SILVER COIN GOLD PROJECT December 2009					
			Cost (US$) per Year		
Personnel	Make	Number (over life)	Labor Rate (US$)	Benefits (40%)	Total Rate (US$)
Hourly Labor					
Drillers	DM45	4-8	58,000	23,200	81,200
Drillers Helpers		4-8	48,000	19,200	67,200
Blasters		2	58,000	23,200	81,200
Blasters Helpers		2	48,000	19,200	67,200
Loader Operators	Cat 992G	4-8	64,000	25,600	89,600
Truck Drivers	Cat 777F	28-12	54,000	21,600	75,600
Dozer Operators	D9	2	58,000	23,200	81,200
	D10	4	58,000	23,200	81,200
	834H	2	58,000	23,200	81,200
Grader Operators	14M	2	58,000	23,200	81,200
	16M	2	58,000	23,200	81,200
Water Truck Operator		4	54,000	21,600	75,600
Misc. Hourly @ 15%		8-11	46,000	18,400	64,400
Maintenance @ 35%		65-112	62,000	24,800	86,800
Salaried Labor					
Gen. Superintendent		1	120,000	48,000	168,000
Mine Manager		1	110,000	44,000	154,000
Maintenance Manager		1	110,000	44,000	154,000
Mine Shift Boss		2	90,000	36,000	126,000
Maintenance Shift Boss		2	90,000	36,000	126,000
Chief Engineer		1	110,000	44,000	154,000
Mine Engineer		1	90,000	36,000	126,000
Chief Geologist		1	80,000	32,000	112,000
Geologist		2	60,000	24,000	84,000
Surveyor		2	50,000	20,000	70,000
Surveyor Assistant		2	40,000	16,000	56,000
TOTAL					

24.1.5 Process Facilities Capital Cost

Preliminary capital cost estimates has been prepared for both a 10,000 tpd all-flotation process facility and a 10,000 tpd flotation-cyanidation process facility. Both process facilities are similar in that they include primary crushing, grinding and bulk sulfide flotation. The two process facilities differ in that the flotation-cyanidation process includes cyanidation of the bulk sulfide concentrate to recover the contained gold values as a marketable product at site, whereas the all-flotation process produces a bulk sulfide concentrate that must be shipped off site to a smelter. These capital cost estimates were developed from information provided by Mine Cost Services (2009) and from in-house data, and are judged to be at a +/- 50 percent level of accuracy. The capital cost for the all-flotation process facilities is estimated at US$101 million and is summarized in TABLE 24-6. The capital cost for the flotation-cyanidation process facilities is estimated at almost US$116 million (TABLE 24-7), and is based on the assumption

TABLE 24-6: CAPITAL COST ESTIMATE FOR A 10,000 TPD ALL-FLOTATION PROCESS FACILITY PINNACLE MINE, LTD. – SILVER COIN GOLD PROJECT December 2009	
Cost Area	**US$**
Equipment	23,850,000
Equipment Installation	15,610,000
Concrete	2,000,000
Piping	6,780,000
Structural Steel	2,200,000
Instrumentation	1,500,000
Insulation	1,200,000
Electrical	2,900,000
Mill Building	3,500,000
Total Process Plant	**59,540,000**
Infrastructure and Site Services	12,000,000
Tailing Disposal	15,000,000
Non Process Buildings (office, warehouse, truckshop, lab)	5,000,000
Total Direct and Indirect	**91,540,000**
EPCM @ 10% of Direct and Indirect	9,154,000
Total Process Capital	**100,694,000**

TABLE 24-7: CAPITAL COST ESTIMATE FOR A 10,000 TPD FLOTATION-CYANIDATION PROCESS FACILITY PINNACLE MINE, LTD. – SILVER COIN GOLD PROJECT December 2009	
Cost Area	**US$**
Equipment	23,850,000
Equipment Installation	15,610,000
Concrete	2,000,000
Piping	6,780,000
Structural Steel	2,200,000
Instrumentation	1,500,000
Insulation	1,200,000
Electrical	2,900,000
Mill Building	3,500,000
Cyanidation Circuit (1)	13,300,000
Cyanide Destruct Circuit	750,000
Total Process Plant	**73,590,000**
Infrastructure and Site Services	12,000,000
Tailing Disposal	15,000,000
Non Process Buildings (office, warehouse, truckshop, lab)	5,000,000
Total Direct and Indirect	**105,590,000**
EPCM @ 10% of Direct and Indirect	10,559,000
Total Process Capital	**116,149,000**

Note:

1. Assume 5% of Flotation Plant feed goes to cyanidation at 500 tpd

that bulk sulfide concentrates will be produced at the rate of up to 500 tpd as feed to the cyanidation circuit.

24.1.6 Other Capital Costs

Startup capital for other project components are summarized (TABLE 24-8) and discussed elsewhere in this report.

TABLE 24-8: OTHER CAPITAL COSTS
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

Item	Start-up Cost (US$ 1000)
Access and Site Preparation	5,000
Permitting & Bonding	5,000
Reclamation and Closure	15,000
Working Capital (1)	12,616
Total Other	**37,616**
Total Estimated Closure Costs(2)	15,000

(1) Estimated at 1/3 of Year 1 total operating costs.

(2) Not part of initial startup capital costs as this charge occurs at the end of the project.

24.1.7 Capital Cost Summary

Total initial capital for the pre-production period is estimated at US$145.8 million for the project based on an all all-flotation process and US$161.3 million for the project based on a flotation-cyanidation process. A capital cost summary for both alternatives is provided in TABLE 24-9.

TABLE 24-9: INITIAL CAPITAL COST SUMMARY, $1,000S PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009		
AREA	**All-Flotation (Base-Case)**	**Flotation-Cyanidation**
Mine		
Access and Site Prep	5,000	5,000
Mine Facilities	3,600	3,600
Mine Equipment and Spares	29,268	29,268
Subtotal - Mine	**37,868**	**37,868**
Process		
Infrastructure and Site Services	12,000	12,000
Nonprocess Buildings	5,000	5,000
Process Plant	59,540	73,590
Tailing Disposal	15,000	15,000
Subtotal - Process	**91,540**	**105,590**
Other		
EPCM	9,154	10,559
General Surface Mobile Equipment	2,250	2,250
Permitting and Bonding	5,000	5,000
Subtotal - Other	**16,404**	**17,809**
Total Initial Capital	**145,812**	**161,267**

24.2 Operating Costs Estimates

24.2.1 Plant Operating Costs

Process plant manpower schedule and labor costs, including a 40% burden, are presented in TABLES 24-10 and 24-11 for both the all-flotation and flotation-cyanidation process alternatives. Plant operating costs for both process facilities are summarized in TABLES 24-12. Operating costs are based on the following:

- Manpower schedule and 2009 labor rates typical of western Canada.
- Electrical power estimated at US$0.07/KWh
- Power consumption is based on typical plants of similar capacity and ore hardness.
- Reagent costs are based on unit consumption rates identified in the preliminary metallurgical studies and typical unit reagent costs.
- Wear materials are based on typical consumption rates and current unit pricing.

TABLE 24-10: ALL FLOTATION PROCESS PLANT MANPOWER SCHEDULE PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009					
Position	**Workers/Day**	**Base, $/hr**	**Base, $/year**	**Burden, %**	**Total, $/year**
Control Room	3	23		40%	282,072
Crushing Plant					
Lead Operator	3	20		40%	245,280
Operator	3	19		40%	233,016
Helper	3	16		40%	196,224
Grinding					
Lead Operator	3	20		40%	245,280
Operator	3	19		40%	233,016
Flotation					
Lead Operator	3	22			192,720
Operator	3	19			166,440
Filters					
Operator	3	19		40%	233,016
Laborers	12	14		40%	686,784
Maintenance					
Mechanics	8	21		40%	686,784
Electrician/Instrumentation	8	22		40%	719,488
Laboratory					
Assayer	3	20		40%	245,280
Technicians	6	18		40%	441,504
Total Hourly	**64**				**4,806,904**
Salaried Personnel					
Process Superintendent	1		130,000	40%	182,000
Metallurgist	1		110,000	40%	154,000
Chief Chemist	1		65,000	40%	91,000
General Foreman	1		98,000	40%	137,200
Shift Foreman	4		90,000	40%	504,000
Maintenance Superintendent	1		98,000	40%	137,200
Maintenance Supervisor	2		90,000	40%	252,000
Total Salaried	**11**				**1,457,400**
Total Process Labor	**75**				**6,264,304**

TABLE 24-11: FLOTATION-CYANIDATION PROCESS PLANT MANPOWER SCHEDULE
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

Position	Workers/Day	Base, $/hr	Base, $/year	Burden, %	Total, $/year
Control Room	3	23		40%	282,072
		Crushing Plant			
Lead Operator	3	20		40%	245,280
Operator	3	19		40%	233,016
Helper	3	16		40%	196,224
		Grinding			
Lead Operator	3	20		40%	245,280
Operator	3	19		40%	233,016
Flotation					
Lead Operator	3	22			192,720
Operator	3	19			166,440
		Cyanidation/Refining/Detox			
Operator	6	19		40%	466,032
Laborers	12	14		40%	686,784
		Maintenance			
Mechanics	8	21		40%	686,784
Electrician/Instrumentation	8	22		40%	719,488
		Laboratory			
Assayer	3	20		40%	245,280
Technicians	6	18		40%	441,504
Total Hourly	**67**				**5,039,920**
		Salaried Personnel			
Process Superintendent	1		130,000	40%	182,000
Metallurgist	2		110,000	40%	308,000
Chief Chemist	1		65,000	40%	91,000
General Foreman	1		98,000	40%	137,200
Shift Foreman	4		90,000	40%	504,000
Maintenance Superintendent	1		98,000	40%	137,200
Maintenance Supervisor	2		90,000	40%	252,000
Total Salaried	**12**				**1,611,400**
Total Process Labor	**79**				**6,651,320**

TABLE 24-12: ESTIMATED PROCESS PLANT OPERATING COST PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT December 2009						
					All Flotation	Flotation-Cyanidation
PRODUCTION RATE						
	Ore Tons				3,500,000	3,500,000
LABOR						
	Payroll (1)				6,264,304	6,651,320
ELECTRICAL POWER		$/kWh	kWhr/ton (2)	$/ton		
	Total Electricity Consumption	0.07	26.7	$1.87	6,541,500	6,541,500
CONSUMABLES						
		Unit Cost	Usage			
	Chemicals and Reagents	$/kg	kg/tonne	$/tonne		
	Flotation Circuit					
	Collector - SIPX	3.00	0.035	0.105	367,500	367,500
	Collector - A208	3.00	0.055	0.165	577,500	577,500
	Lime	0.40	1.5	0.600	2,100,000	2,100,000
	Frother	2.50	0.035	0.088	306,250	306,250
	Flocculant	5.50	0.006	0.033	115,500	115,500
	CYANIDATION CIRCUIT (FEED TO CYANIDATION)					
	Cyanide (3)	2.70	6.0	0.810		2,835,000
	Lime (3)	0.40	2.00	0.040		140,000
	Sodium metabisulphite (3)	2.70	0.55	0.074		259,875
	WEAR MATERIALS					
	Grinding Media	1.45	0.50	0.725	2,537,500	2,537,500
	Mill Liners			0.150	525,000	525,000
	Primary Crusher Liners	1.20	0.013	0.030	105,000	105,000
	Cone Crusher Liners	1.30	0.030	0.060	210,000	210,000
	Subtotal			$2.88	6,844,250	10,079,125
OPERATING AND MAINTENANCE SUPPLIES						
	Minor Operating Supplies and Consumables				500,000	700,000
	Major Maintenance Items				1,000,000	1,250,000
	Minor Maintenance Materials & Services				300,000	400,000
	Subtotal				1,800,000	2,350,000
OTHER PLANT OPERATING COSTS						
	Insurances etc				100,000	100,000
	Staff Travel				100,000	100,000
	Consultants & Services				100,000	150,000
	Other				200,000	200,000
	Subtotal				500,000	550,000
Total Process Operating Cost ($/Year)					$21,950,054	$26,171,945
Total Process Operating Cost ($/ton)					6.27	7.48

NOTES:
1. Based on Process Manpower Schedule and includes 40% burden
2. Based on in-house data for comparable plant
3. Kg/tonne of concentrate = 5 wt% of new ore

24.2.2 General and Administrative Costs

TABLE 24-13 summarizes general operation staffing, salary costs, and estimated expenses for general and administrative services The general and administrative payroll is estimated at approximately US$2.7 million per year, and the operating expenses are estimated at US$2.0 million per year, for a total G&A cost of US$ 4.7 million, which is equivalent to US$1.33 per ton of ore processed at full production.

24.3 Environmental Considerations—Bonding, Reclamation and Closure

A reclamation and closure plan will be required to protect groundwater and surface water resources, meet post-mining land use objectives and satisfy the regulatory commitments. The primary reclamation elements will include:

- Regrading and contouring tailings and waste rock facilities
- Facilities demolition
- Regrading facilities and roads
- Application of top soil or other suitable growth medium
- Revegetation

Post-mining reclamation and, to the extent possible, concurrent reclamation will be conducted in accordance with applicable Provincial and Federal regulations. A US$15 million allowance for the cost of reclamation and closure of the site has been included in the cash flow projection based on the physical requirements for closure and closure costs from similar gold milling operations. A US$0.25 per tonne million allowance has been allocated for environmental management and concurrent reclamation during the life of mine operations. Additionally, US$5.0 million is included in the cash flow for permitting and bonding.

24.4 Cash Flow Analysis

Cash flow analyses was developed for the mining and processing the measured, indicated and inferred resources currently defined at Silver Coin. Both the all-flotation and flotation-cyanidation process alternatives were evaluated and included the following input parameters:

- Gold price at US$850 per ounce and silver price at US$14.25 per ounce
- All-flotation process gold recovery at 95 percent and silver recovery at 88 percent
- Flotation-cyanidation process gold recovery at 88 percent and silver recovery at 60 percent
- Mine operating cost at $2.31per tonne mined
- Process operating cost at $6.27 per tonne ore for the all-flotation process alternative and US$7.48 per tonne processed for the flotation-cyanidation process alternative.
- G & A at US$1.33 per tonne ore processed
- Concentrate transport and smelting costs were based on the following:
    - Trucking and port handling – US$5.00 per tonne of concentrate
    - Ocean freight – US$60.00 per tonne of concentrate
    - Smelter treatment charge – US$200 per tonne of concentrate

TABLE 24-13: ESTIMATED GENERAL AND ADMINISTRATION COSTS
PINNACLE MINES LTD. – SILVER COIN GOLD PROJECT
December 2009

		No. Employee	Base, $/year	Burden	Total, $/year
		Management			
	General Manager	1	150,000	0.4	210,000
	Administrative Assistant	1	70,000	0.4	98,000
		Accounting			
	Accounting Manager	1	90,000	0.4	126,000
	Accountant	1	75,000	0.4	105,000
	Payroll	1	50,000	0.4	70,000
	Accounts Receivable	1	50,000	0.4	70,000
		Purchasing			
	Purchasing Manager	1	90,000	0.4	126,000
	Warehouseman	3	75,000	0.4	315,000
	Inventory Control	1	60,000	0.4	84,000
	Buyer	1	60,000	0.4	84,000
		Human Resources			
	HR Manager	1	90,000	0.4	126,000
	Clerk	1	50,000	0.4	70,000
		Safety, Security & Environmental			
	Safety/Security Manager	1	90,000	0.4	126,000
	Safety Trainer/Inspector	1	60,000	0.4	84,000
	Security	8	60,000	0.4	672,000
	Environmental Manager	1	90,000	0.4	126,000
	Environmental Technicians	2	60,000	0.4	168,000
Total G & A Payroll					2,660,000
General Expenses					2,000,000
Total G & A Cost					4,660,000
Total G & A Cost ($/tonne processed)					1.33

- Gold refining charge – US$6.00 per oz.
- Silver refining charge – US$0.50 per oz.

TABLE 24-14 provides a cash flow summary for the project based on processing the ore by the flotation-cyanidation process alternative. This cash flow indicates a before tax net present value (NPV) of US$58.3 million for the project at a 10 percent discount rate, and assumes 100 percent equity and a constant 2009 US dollar. TABLE 24-15 summarizes the sensitivity of the Net Present Value ("NPV") of the projected cash flows at discount rates of 5%, 8%, 10% and 12% for variation in Capex, Opex and gold price.

TABLE 24-16 provides a cash flow summary for the project based on processing the ore by the all-flotation-process alternative. This cash flow indicates a before tax net present value (NPV) of a negative US$82 million for the project. This economics of the all-flotation alternative is negatively impacted by concentrate transport and smelting charges. TABLE 24-17 summarizes the sensitivity of the Net Present Value ("NPV") of the projected cash flows at various discount rates.

TABLE 24-14: CASH FLOW FOR FLOTATION - CYANIDATION PROCESS ALTERNATIVE

PINNACLE MINES LTD - SILVER COIN PROJECT

December 2009

			PP 2	PP 1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Totals
PRICE SCHEDULE	$/oz Au				$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00	$850.00		
	$/oz Ag				$14.25	$14.25	$14.25	$14.25	$14.25	$14.25	$14.25	$14.25	$14.25	$14.25	$14.25	$14.25	$14.25		
MINE PRODUCTION SCHEDULE																			
ORE	tons (000s)				3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	840		42,840
	Au	g/t			0.83	0.80	0.85	1.03	1.14	1.22	1.17	1.07	1.00	1.04	1.38	1.79	1.05		
	Ag	g/t			6.23	5.50	5.54	6.15	6.61	7.25	6.35	6.52	10.13	6.79	10.14	9.16	9.17		
WASTE	tons (000s)			6,563	9,500	8,500	7,000	4,835	3,143	2,363	1,958	3,026	3,253	2,757	2,857	1,983	69		55,806
	TOTAL			6,563	13,000	12,000	10,500	8,335	6,643	5,863	5,458	6,526	6,753	6,257	6,357	5,483	909		98,645
	Stripping Ratio	1.30			2.71	2.43	2.00	1.38	0.90	0.68	0.56	0.86	0.93	0.79	0.82	0.57	0.08		
MILL PRODUCTION SCHEDULE																			
MILL FEED	tons (000s)				3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	840		42,840
	Au	g/t			0.83	0.80	0.85	1.03	1.14	1.22	1.17	1.07	1.00	1.04	1.38	1.79	1.05		
	Ag	g/t			6.23	5.50	5.54	6.15	6.61	7.25	6.33	6.52	10.13	6.78	10.14	9.16	9.17		
MILL RECOVERY	% of Au in feed	85%			88%	86%	84%	84%	84%	84%	85%	85%	85%	84%	84%	88%	85%		
	% of Ag in feed	60%			60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%		
Au PRODUCED		oz			83,576	89,825	84,313	101,314	112,474	120,734	115,873	106,087	99,134	103,155	156,411	177,373	25,035		1,362,864
Ag PRODUCED		oz			420,287	371,351	375,535	415,130	446,605	489,212	542,577	643,025	684,094	583,557	684,647	618,501	146,943		6,433,380
REVENUE ($000)																			
	Au Revenue	1.0			70,189	75,501	80,080	86,032	95,603	102,637	98,322	90,157	84,264	87,682	115,950	150,764	21,280		1,154,434
	Ag Revenue				5,989	5,295	5,351	5,916	6,364	6,971	8,017	9,163	9,748	8,454	9,756	8,814	2,118		91,961
	TOTAL	1.0			76,178	80,796	85,432	91,948	101,967	109,578	106,339	99,320	94,012	96,140	125,706	159,580	23,398		1,250,395
MINE OPERATING COSTS ($000s)																			
Mining Materials and Supply				7,427	11,844	13,133	12,532	11,246	10,131	9,102	8,416	7,885	7,885	7,885	7,714	7,457	2,499		125,157
Mining Labor				1,583	6,909	10,618	10,618	10,616	10,097	10,007	9,338	6,594	6,107	6,065	5,757	4,998	3,510		103,003
Total Mining		$2.31		9,009	18,753	23,749	23,148	21,862	20,228	19,109	17,754	14,379	13,993	13,950	13,471	12,456	6,008		228,160
Processing	$/tonne processed	$7.43			26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	26,180	6,265		
G&A	$/tonne processed	$1.33			4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	1,118		
Environmental	$/tonne processed	$0.25			875	875	875	875	875	875	875	875	875	875	875	875	215		
	TOTAL	1.0		9,009	50,463	55,464	54,861	53,572	51,939	50,909	49,464	46,289	45,782	45,660	45,181	44,169	13,832		
FREIGHT & MARKETING COST ($000s)																			
Refining, Freight and Insurance	$/oz Au recovered	$35			2,064	2,221	2,355	3,530	3,812	3,018	2,892	2,652	2,470	2,579	3,410	4,434	636		
Marketing	% of revenue	0.0%			0	0	0	0	0	0	0	0	0	0	0	0	0		
	TOTAL				2,064	2,221	2,358	3,530	3,812	3,018	2,892	2,652	2,470	2,578	3,410	4,434	536		
ROYALTY ($000s)		0%			0	0	0	0	0	0	0	0	0	0	0	0	0		
	TOTAL				0	0	0	0	0	0	0	0	0	0					
TOTAL COSTS ($000s)					53,529	57,679	57,313	56,102	54,750	53,827	52,356	48,941	48,180	48,238	48,581	48,600	14,248		641,356
TOTAL COSTS PER TON MILLED					15.01	16.46	16.35	16.03	15.64	15.41	14.99	13.98	13.77	13.78	13.88	13.89	16.96		
NET BEFORE TAX ($000s)				(9,009)	23,649	23,117	28,219	35,845	47,216	51,651	53,983	50,379	45,832	47,901	77,115	110,880	9,150		600,030
[illegible]				[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		600,030
CAPITAL COST SUMMARY ($000s)																			
Road Access and Site Prep			5,000																5,000
Surface Buildings			1,600	2,000															3,600
Mining Equipment				15,346	13,821	120	120	1,060	120	4,260	1,020	1,060	720	725	120	120			38,527
Site Infrastructure, power, water			6,000	6,000															12,000
Process Plant			36,795	36,795															73,590
Office, warehouse, lab				5,000															5,000
Tailings			7,500	7,500				2,500					2,500						20,000
EPCM			10,559																10,559
Working Capital		3 Month Opex			12,818												(12,818)		0
Permitting, Reclamation, and Closure			5,000														5,000	10,000	20,000
TOTAL CAPITAL		1.0	72,454	73,641	26,637	120	120	3,560	120	4,480	1,020	1,060	3,220	725	120	120	-7,818	10,000	188,279
Contingency		20%	14,491	14,828	5,327	24	24	712	24	816	204	212	644	144	24	24	-1,823	2,000	
TOTAL CAPITAL WITH CONTINGENCY		1.0	86,945	87,469	31,965	144	144	4,272	144	4,896	1,224	1,272	3,864	864	144	144	-9,140	12,000	225,931
NET PRETAX CASH FLOW ($000s)			[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

NPV (10%) $58,348

IRR 15%

TABLE 24-15: FLOTATION–CYANIDATION ALTERNATIVE SENSITIVITY ANALYSIS
PINNACLE MINES LTD. - SILVER COIN GOLD PROJECT
December 2009

Net Present Value Calculations ($000s) Capital Sensitivity			
Discount %	**Base**	**CAPEX-20%**	**CAPEX+20%**
0	374,099	419,286	328,913
5	170,130	210,485	129,776
8	95,428	133,442	57,415
10	58,349	94,970	21,728
12	28,861	64,201	-6,479
Net Present Value Calculations ($000s) Au Price Sensitivity, US$/oz			
Discount %	**850**	**900**	**800**
0	374,099	442,242	305,956
5	170,130	214,352	125,908
8	95,428	130,353	60,503
10	58,349	88.457	28,242
12	28,861	54,990	2,733
Net Present Value Calculations ($000s) Operating Cost Sensitivity			
Discount %	**Base**	**Op Cost-20%**	**Op Cost+20%**
0	374,099	497,358	280,841
5	170,130	254,687	85,574
8	95,428	164,340	36,516
10	58,349	118,962	-2,263
12	28,861	82,494	-24,771

TABLE 24-18: CASH FLOW FOR ALL - FLOTATION PROCESS ALTERNATIVE
PINNACLE MINES LTD - SILVER COIN PROJECT
December 2009

			PP 2	PP 1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Totals
PRICE SCHEDULE	$/oz Au				$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	$ 850.00	
	$/oz Ag				$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	$ 14.25	
MINE PRODUCTION SCHEDULE																					
ORE	tons (000s)		0	0	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	1,673	54,173
	Au	g/t	0.00	0.00	0.74	0.72	0.78	0.83	0.85	0.87	1.00	1.07	1.03	0.88	0.83	0.80	0.96	1.26	1.79	1.24	
	Ag	g/t	0.00	0.00	8.81	5.79	5.50	5.36	5.83	6.30	6.44	6.83	7.88	8.50	8.20	8.60	7.74	9.08	8.05	8.22	
WASTE	tons (000s)		0	8,351	7,831	8,615	5,835	4,681	4,135	3,560	3,053	2,547	2,384	2,453	2,541	2,720	3,315	3,041	2,134	651	65,786
	TOTAL		0	8,351	11,331	12,115	9,335	8,181	7,635	7,060	6,553	6,047	5,884	5,953	6,041	6,220	6,815	6,541	5,634	2,324	119,959
	Stripping Ratio	1.21			2.23	2.46	1.67	1.34	1.18	1.02	0.87	0.73	0.68	0.70	0.73	0.78	0.95	0.87	0.61	0.39	
MILL PRODUCTION SCHEDULE																					
MILL FEED	tons (000s)				3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	1,673	54,173
	Au	g/t			0.74	0.72	0.78	0.83	0.85	0.87	1.00	1.07	1.03	0.88	0.83	0.80	0.96	1.26	1.79	1.24	
	Ag	g/t			8.81	5.79	5.53	5.36	5.83	6.30	6.48	6.83	7.88	8.50	8.20	8.60	7.74	9.08	8.05	8.22	
MILL RECOVERY	% of Au in feed	85%			85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	
	% of Ag in feed	84%			84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	
Au PRODUCED		oz			79,255	76,583	83,253	88,289	90,737	103,384	106,475	114,803	110,441	104,443	98,860	98,243	102,444	135,117	181,720	63,430	1,635,547
Ag PRODUCED		oz			684,535	575,712	547,843	530,633	577,378	623,933	642,046	688,216	760,583	841,405	919,630	851,855	764,435	898,746	798,877	388,129	11,061,354
CONCENTRATE PRODUCED	tonnes	Wt% 2.50%			87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	87,500	41,820	
CONCENTRATE GRADE	Au	oz/tonne			0.91	0.88	0.95	1.01	1.04	1.18	1.22	1.31	1.26	1.19	1.13	1.16	1.17	1.54	2.08	1.52	
	Ag	oz/tonne			7.83	6.56	6.26	6.06	6.60	7.13	7.34	7.86	8.69	9.62	10.41	9.73	8.76	10.27	9.11	9.30	
Au PAYABLE	Deduct .05 oz Au and pay for 95%				71,136	68,587	74,854	79,718	82,644	94,039	96,895	104,807	100,783	95,083	88,875	87,275	93,164	124,205	168,478	58,281	
Ag PAYABLE	Deduct 1 oz Ag and pay for 85%				547,171	461,801	437,440	420,875	465,342	509,672	520,819	568,780	639,437	718,300	761,874	725,847	644,868	770,884	674,003	329,843	
REVENUE ($000)	Au Revenue	1.0			60,465	58,307	63,654	67,760	69,737	79,933	82,448	89,171	85,643	80,805	78,394	74,184	79,181	105,574	143,208	49,536	1,266,059
	Ag Revenue				8,083	6,582	6,234	5,998	6,633	7,262	7,507	8,105	9,113	10,206	11,143	10,345	9,181	10,983	9,605	4,702	131,684
	TOTAL	1.0			68,548	64,889	69,888	73,755	76,369	87,195	89,953	97,276	94,761	91,015	89,537	84,529	88,363	116,556	152,811	54,248	1,397,743
MINE OPERATING COSTS ($000s)																					
Mining Materials and Supply				7,427	11,846	13,132	12,532	11,246	10,131	8,102	8,418	7,883	7,845	7,885	7,714	7,457	2,490				125,157
Mining Labor				1,582	6,909	10,616	10,616	10,616	10,097	10,007	9,336	6,494	6,107	6,065	5,757	4,999	3,510				123,001
Total Mining		32.31		9,009	18,753	23,748	23,148	21,862	20,228	18,109	17,754	14,378	13,952	13,950	13,471	12,456	6,000	15,111	13,014	5,372	281,657
Processing	$/ tonne processed	$6.27			21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	21,945	10,490	
G&A	$/ tonne processed	$1.33			4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	4,655	2,225	
Environmental	$/ tonne processed	$0.25			875	875	875	875	875	875	875	875	875	875	875	875	875	875	875	418	
	TOTAL	1.0		9,009	46,230	51,223	50,633	49,337	47,703	46,674	45,229	42,684	41,447	41,425	40,946	39,931	33,484	42,586	40,433	18,563	
FREIGHT & MARKETING COST ($000s)																					
Trucking and Port Handling	$/tonne of Conc	$5.00			875	875	875	875	875	875	875	875	875	875	875	875	875	875	875	418	
Ocean Freight	$/tonne of Conc	$60			5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	5,250	2,509	
Smelter Treatment Charge	$/tonne of Conc	$200			17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	17,500	8,364	
Au Refining Charge	$/oz Au	$6.00			427	412	450	478	493	564	582	629	605	570	530	524	559	745	1,011	350	
Ag Refining Charge	$/oz Ag	$0.50			274	231	218	215	233	255	263	284	320	356	381	363	337	385	337	165	
	TOTAL				24,326	24,268	24,293	24,314	24,350	24,444	24,470	24,635	24,645	24,681	24,616	24,612	24,506	24,754	24,973	11,804	
ROYALTY ($000s)		0%			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
	TOTAL				0	0	0	0	0	0	0	0	0	0							
TOTAL COSTS ($000s)				9,009	70,565	75,491	74,916	73,651	72,053	71,118	69,699	66,583	66,016	65,976	65,501	64,443	57,990	67,341	65,462	30,308	1,060,137
TOTAL COSTS PER TON MILLED					20.16	21.57	21.40	21.04	20.58	20.32	19.91	19.03	18.86	18.85	18.71	18.41	16.57	19.24	18.70	18.12	
NET BEFORE TAX ($000s)				(9,009)	(2,016)	(10,601)	(4,987)	109	4,316	16,077	20,253	30,683	28,744	25,033	22,036	20,086	30,331	49,215	87,349	23,931	337,606
[illegible]				[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	337,606
CAPITAL COST SUMMARY ($000s)																					
Road Access and Site Prep			5,000																		5,000
Surface Buildings			1,500	2,000																	3,500
Mining Equipment				15,346	13,821	120	120	1,060	120	4,080	1,020	1,060	720	720	120	120					38,527
Site Infrastructure, power, water			6,000	6,000																	12,000
Process Plant			29,770	29,770																	59,540
Office, warehouse, lab				3,000																	3,000
Tailings			7,500	7,500				2,500					2,500								20,000
EPCM			9,154																		9,154
Working Capital		3 Month Opex			11,554															(11,554)	0
Permitting, Reclamation, and Closure			3,000															5,000	5,000	5,000	18,000
TOTAL CAPITAL		1.0	61,924	65,616	25,375	120	120	3,560	120	4,080	1,020	1,060	3,220	720	120	120	-			-6,554	150,821
Contingency		20%	12,385	13,123	5,075	24	24	712	24	816	204	212	644	144	24	24	-				
TOTAL CAPITAL WITH CONTINGENCY		1.0	74,309	78,739	30,450	144	144	4,272	144	4,896	1,224	1,272	3,864	864	144	144	-	0	0	-6,554	184,287
NET PRE-TAX CASH FLOW ($000s)			[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

NPV (10%) ($52,013)
IRR 4%

TABLE 24-17: ALL-FLOTATION ALTERNATIVE SENSITIVITY ANALYSIS PINNACLE MINES LTD. - SILVER COIN GOLD PROJECT December 2009			
Net Present Value Calculations ($000s) Capital Sensitivity			
Discount %	Base	CAPEX-20%	CAPEX+20%
0	137,310	178,169	98,450
5	-16,112	19,345	-51,569
10	-82,013	-49,572	-114,454
12	-96,000	-64,648	-127,352
Net Present Value Calculations ($000s) Au Price Sensitivity, US$/oz			
Discount %	850	900	800
0	137,310	211,783	62,836
5	-16,112	28,288	-60,512
10	-82,013	-53,666	-110,371
12	-96,000	-71,890	-120,110
Net Present Value Calculations ($000s) Operating Cost Sensitivity			
Discount %	Base	Op Cost-20%	Op Cost+20%
0	137,310	275,154	-535
5	-16,112	72,795	-105,019
10	-82,013	-20,980	-143,047
12	-96,000	-42,703	-149,297

25.0 ILLUSTRATIONS

All of the illustrations used in the preparation of this report appear in each of their respective sections.

APPENDIX A

"Metallurgical Study on the Silver Coin Gold Project"
by F. Wright Consulting Inc., January 8, 2009

Metallurgical Study

on the

Silver Coin
Gold Project

Prepared for:

Pinnacle Mines Ltd.

350 – 885 Dunsmuir Street
Vancouver BC Canada
V6C 1N5

January 8, 2009

F. Wright Consulting Inc.
427 Fairway Dr., North Vancouver, BC, Canada V7G 1L4
Phone: 604 802-4449 / email: fwright@telus.net

TABLE OF CONTENTS

Page No.

APPENDICES

1.0 SUMMARY

Pinnacle Mines Ltd. provided eight mineral composite samples for undertaking a preliminary metallurgical study on the Silver Coin Gold Project. The gold (Au) and silver (Ag) head grade, along with the total sulfur content (S_T) are provided in the following table for each composite and on a blended master composite (MC1).

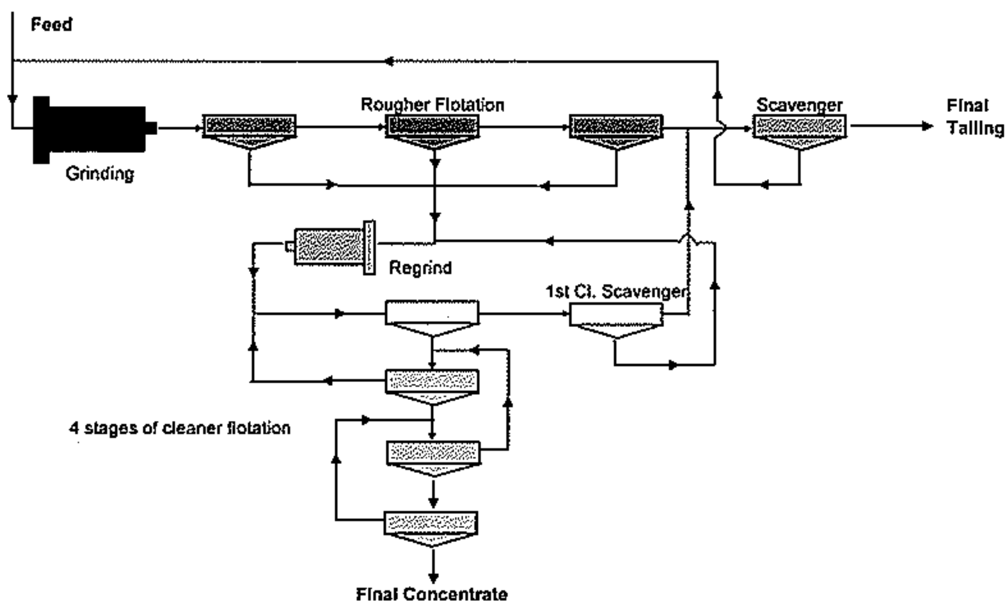
Head Assays

Comp. #	Au (g/t)	Ag (g/t)	$\%S_T$
08-1	0.41	2.3	2.20
08-2	1.35	7.6	4.11
08-3	1.45	8.3	4.62
08-4	1.69	8.9	8.44
08-5	2.88	22.7	5.46
08-6	0.38	5.5	2.30
08-7	1.85	3.5	2.45
08-8	1.96	5.2	5.27
MC1	1.87	7.1	4.55

The laboratory test program investigated flotation and cyanide leaching procedures. Depending on the sample and procedures that were used the gold recoveries varied from 85% to 95%, for cyanidation. Results were marginally improved using carbon in leach (CIL) procedures. Gravity recovery methods prior to cyanidation were also beneficial in reducing both gold losses and the required leach retention time. Cyanide silver recovery ranged from 62% to 83% depending on the sample tested. The preliminary work indicates that cyanidation of either the flotation concentrate or the whole ore can be considered as a potential process procedure. However, due to anticipated permitting considerations in using cyanidation on site, most of the process evaluation focused on flotation.

Initial open cycle flotation tests used a xanthate collector at a natural pH. These studies indicated the material was relatively insensitive to grind in relation to precious metal recovery, and provided an excellent bulk yield of over 95% gold into a rougher concentrate. However, this concentrate did not upgrade satisfactorily due to the high pyrite content, resulting in grades of generally less than 30 g/t Au.

Cleaning studies showed pyrite could be rejected without high gold losses. Consequently, two alternate flotation procedures were evaluated to reject pyrite in order to improve the concentrate grade. These procedures consisted of using a more gold selective collector, and secondly the use of elevated pH. Following open cycle evaluation both procedures were separately tested with locked cycle studies on the master composite MC1 sample. Based on the locked cycle flotation results a conceptual flowsheet using elevated pH was established, as provided in a simplified schematic below.



The flowsheet is relatively conventional producing a bulk rougher concentrate that was reground and cleaned in 3 to 4 stages. The rougher and 1st cleaner tailing were scavenged and recycled. Sodium hydroxide was used to increase pH in order to reject pyrite in both roughing and cleaning. The results from this locked cycle test provided gold and silver recoveries of approximately 90% on the MC1 composite. The corresponding bulk tailing losses averaged 0.13 g/t for gold, and were generally below detection (<0.5 g/t) for silver. The cleaned concentrate grade averaged 110 g/t Au and 259 g/t Ag during the last three cycles.

Further optimization, as well as sample variability testing to adequately cover the metallurgical response of the resource is required, but the initial test results are considered encouraging. In summary, the laboratory program showed the composite samples that were tested responded well to both cyanidation and flotation. Based on the current understanding of the project and the process response it is recommended that future metallurgical test work continue to optimize flotation procedures for rejecting pyrite to produce a cleaned bulk gold/silver concentrate.

2.0 INTRODUCTION

Pinnacle Mines Ltd., provided mineral samples from the Silver Coin Project, located in northwest British Columbia, Canada. The samples were used for a preliminary evaluation of the mineral processing response and to develop a conceptual flowsheet as part of the scoping study relating to the Project.

The mineral processing testwork focused on froth flotation for recovery of gold, with some investigation into gravity, and cyanidation procedures. Laboratory process tests were primarily performed by Process Research Associates Ltd. (PRA) of Richmond, BC, based in part on an earlier study. Some related environmental studies were undertaken by ALS Chemex of North Vancouver BC. Corresponding chemical analyses was primarily performed by the iPL Laboratory, in Richmond BC. This report provides a summary of the generated data, and an interpretation of the resulting information.

3.0 PROCEDURES

3.1 SAMPLE PREPARATION AND ANALYSES

Process Research Associates (PRA) Ltd., of Richmond, BC, received 95 samples of split drill core in July 2008, as provided in Appendix 1. As part of the geological quality control program, that was not directly related to the metallurgical test work each of the 95 samples was crushed to -10 Tyler mesh and a sub-sample split out for individual head analyses consisting of gold by fire assay, total sulfur, and multi-element analysis by induced coupled plasma spectrophotometry (ICP).

The 95 samples were then segregated and blended into eight composite samples. Sample compositing was done in consultation with the client's project geologist to represent the expected grade range, as well as the primary lithologies and mineralogy of the resource. Each composite sample represented a continuous interval of drill core from various spatial areas and depths of the resource. No mineralogical studies were specifically performed as part of this test program, but a historical mineralogical report dated January 4, 2007 provided by the client was referred to.

The composites were separated into two groups representing high and low sulfide material, where the primary sulfide is reported as pyrite. Each composite head analyses included precious metal assays, total and sulfide sulfur, total and organic carbon, zinc, solid specific gravity (SG), whole rock, and multi-element analysis by induced coupled plasma spectrophotometry (ICP).

Analytical work was performed by iPL Laboratories, which has ISO 9001 accreditation using government certified assayers. Gold analyses was undertaken by standard fire assay procedures and completed with either a gravimetric or an atomic absorption (AA) finish. A one ton (~30g) fire assay was done on head samples. A metallics assay procedure for gold and silver was also undertaken in which pulverized sample weighing close to 300 g was screened at

150 Tyler mesh and the precious metal content of the coarse and fine sieve fractions compared. Head analyses and various product samples were also submitted for induced coupled plasma spectrophotometry (ICP) scans to provide quantitative multi-element metal species determinations, which included silver. Individual metals of interest were typically finished with ICP or atomic absorption (AA) spectrometry. Total sulphur was measured using a Leco furnace, and sulphide sulphur assays were based on a wet chemical gravimetric procedure. Solids specific gravity was taken using the pycnometric method on finely ground sample.

The quality control and assurance procedures included submission of laboratory standards with each batch of samples analyzed. This information is included in Appendix 2.

3.2 PROCESS TESTING

Laboratory studies and procedures were specified by the report author in consultation with the client. Most of the test work focused on flotation in order to produce a sulfide concentrate for cyanidation or alternately for direct sale to smelter refiners.

Primary grinding was performed in a stainless steel laboratory rod mill. Test grinds were used to calculate the time requirements to meet specified targeted particle size distribution. A standard charge to the mill was slurried to ~65% by weight solids content, and a particle size analysis is performed on the ground product. Particle size analyses was undertaken for each ground sample using a Rotap™ equipped with 20 cm (8") diameter test sieves, stacked in ascending mesh sizes. Each sample was initially wet screened at 37 microns (400 Tyler™ mesh). The +37 micron fraction was then dried and re-screened through the stacked sieves. Each sieved fraction was collected, weighed, and the individual and cumulative percent retained calculated. A standard Bond Ball Mill Work Index using a closing screen size of 105 microns (150 Tyler mesh) was performed on the composites, as detailed in Appendix 2.

For gravity recovery the sample was ground to the specified target particle size, re-pulped to approximately 20% solids by weight, and subjected to a single pass through a Falcon® or Knelson® centrifugal laboratory concentrator. The resulting concentrate was hand-panned to simulate a plant gravity upgrading circuit (typically by tabling), and the entire pan concentrate was fire assayed for gold and silver. The gravity tailing was submitted for further processing by either cyanidation or flotation. Some procedures used the Falcon concentrator to scavenge gold from the flotation tailing. The gravity procedures are outlined in the details for the related tests in Appendix 3, 4 and 5.

Bench scale flotation tests were undertaken in a Denver D12 laboratory machine, to produce a specific pulp density (usually 33% solids by weight), with typically 1 or 2 kg of feed. The D12 impeller speed was set at the required rate according to cell size, and the airflow was controlled manually to maintain the froth level. Various collectors were tested singularly or in combination for recovering sulfide minerals and associated metals, or for specifically targeting liberated gold. Methyl iso-butyl carbinol (MIBC) was used as the frother. Initially kinetic testing was performed without cleaning. Flotation cleaning following regrinding of the rougher concentrate was also undertaken using different methods. All of the flotation tests used municipal potable water at ambient temperature. The initial program consisted of open cycle procedures (see Appendix 3). Based on these results two locked cycle tests were performed each consisting of 6 cycles each as provided in Appendix 4.

Baseline cyanidation tests were performed to determine silver and gold dissolution typically by standard bottle roll procedures, carried out at a pulp density of 40 wt% solids for periods of up to 48 hours. Intermediate solution samples were taken to evaluate gold leaching verses time. Sodium cyanide (NaCN) level was maintained at a pre-selected concentration of typically 2 g/L. Prior to adding NaCN, the alkalinity was adjusted and maintained to pH 10 to 11 with hydrated lime. The dissolved oxygen concentration was monitored for the testing period. Variations to the test included using carbon in leach (CIL) procedures with the addition of activated carbon. At termination of the leach the pregnant leachate solution (PLS) was recovered by filtration and the filter cake washed with hot cyanide solution, followed by two hot water displacement washes. For standard procedures intermediate solution samples were typically removed to determine silver and gold dissolution with time. The leachate and the final residue were analyzed for gold and silver.

Reagent concentrations used in cyanidation were determined using standard titration methods. The sodium cyanide concentration was titrated against 0.1N silver nitrate with para-dimethylamino rhodanine as an indicator. Lime concentration was determined by titrating with oxalic acid with phenolphthalein as the indicator. The reducing power of the final leachate against 0.1N potassium permanganate was used as an indication of potential solution fouling characteristics. The detailed cyanidation procedures are provided with results in Appendix 6.

Settling tests were undertaken on tailing as detailed in Appendix 6. The tailing were slurried in municipal water at the discharge pH. Beaker scoping studies were initially performed on the two composites in order to evaluate several flocculants, and included evaluating increasing the pH with hydrated lime. Following this settling studies using the selected flocculent were performed in 2 L cylinder and the settling interface recorded with time.

4.0 RESULTS

4.1 HEAD CHARACTERIZATION

The samples used for process testing were blended into eight composite samples originating from 95 sub-samples each originating from contiguous drill core from various areas of the resource. The original sample weights and blending ratios are provided in Appendix 1. A summary of assays of the original 95 samples is provided in Appendix 2. The gold content shows a moderate trend with zinc, while there is a generally poor correlation evident between gold and other elements.

A total of eight composite samples were evaluated for the test program, along with a blended master composite (MC1) that was used primarily for the flotation locked cycle evaluation. The head assays of the composites are provided in Appendix 2, and summarized in Table 4.1.

Table 4.1: Head Assays

Comp. #	Au (g/t)	Ag (g/t)	%Pb	%Zn	$\%S_T$
08-1	0.41	2.3	0.06	0.11	2.20
08-2	1.35	7.6	0.32	0.57	4.11
08-3	1.45	8.3	0.11	0.73	4.62
08-4	1.69	8.9	0.31	1.11	8.44
08-5	2.88	22.7	0.53	1.40	5.46
08-6	0.38	5.5	0.02	0.04	2.30
08-7	1.85	3.5	0.07	0.25	2.45
08-8	1.96	5.2	0.02	0.03	5.27
MC1	1.87	7.1	0.07	0.57	4.55

There appears to be no strong correlations between the gold content to that of the silver, sulfur or other major elements, as shown in graphed data provided in Appendix 2. If composites 08-7 (Comp. 7) and 08-8 (Comp. 8) are eliminated there is a relationship between gold and zinc content in the first six head composite samples as shown in Figure 4-1, below.

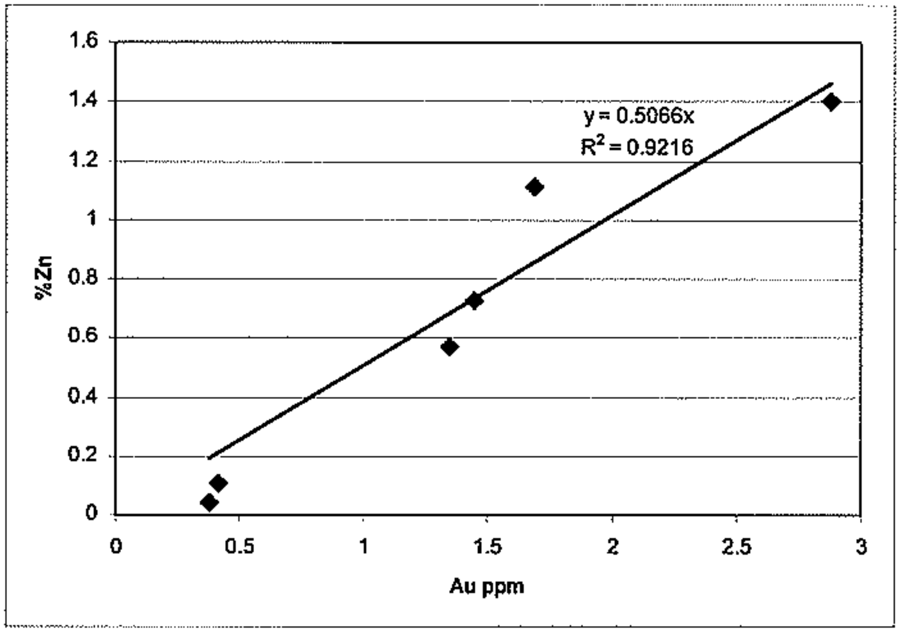


Figure 4-1: Zinc vs Gold Content in Comp. 08-1 to 08-6

Metallic assay procedures (see appendix 2) showed only minor coarse gold or silver was present. The total sulfur content for the composites ranged from 2.2% to 8.4%, virtually all of which analyzed as sulfide sulfur.

There are some deleterious elements present in the eight composites that have potential to negatively impact the process, or which could be penalty items in flotation products. These elements included organic carbon (C_{org}), that ranged from 0.18% to 0.46%, and which is a potential Au preg robber during cyanidation. Arsenic, which can contain refractory gold, was present in a range of 52 to 132 ppm. Mercury measured by cold vapor procedures was variable ranging from <5 ppb to 2089 ppb in the samples. Two other potentially deleterious elements were below detection limits assaying ≤0.001% antimony (Sb), and <0.01% selenium (Se).

Mineralogy is described in an historical petrographic report (dated January 4, 2007) as "dominated by quartz with lesser sercite and minor pyrite; sporadically there is also minor sphalerite, galena and chalcopyrite, and a trace of native gold". "Gold occurs in intergrowth with sulphides, and less as inclusions in pyrite and along pyrite grain borders". Gold silver electrum was also noted, with a color that suggested a silver content of 10% to 25%. No other distinctive silver bearing minerals were noted. In a few gold samples, the gold was reported to be intimately associated with a graphitic substance.

Bond Ball Mill Work Index tests were performed on two composite blends representing a high sulfur and low sulfur material. The work was done at a closing screen size of 105 microns (150 Tyler mesh). This resulted in a similar work index of both materials at ~18.5 kWh/tonne, indicating a moderately hard ore. The solids specific gravity (SG) of the composites ranged from 2.7 to 2.9.

4.2 FLOTATION

4.2.1 Kinetic Flotation

Open cycle, scoping flotation studies using standard sulfide flotation procedures that included the use of xanthate collectors were performed on two composite blends representing high sulfur and low sulfur material. The initial kinetic testing was done at various grinds to determine a grind verses recovery relationship. Tests F1 to F5 were done on the lower sulfur composite blend 08-1 & 08-2, which had a head grade of 0.72 g/t Au and 3.4% S. Test F6 to F10 were performed on the higher sulfur composite of 08-5 and 08-6 with a head grade of 2.19 g/t Au and 4.1% S.

Procedures were similar except for the primary grind size and collectors used. The primary collectors selected for investigation were SIPX and A208 in equal ratios. Modifications were for tests F1 and F6 done at the finest targeted grind size (80% passing 53 microns), which used a gold specific collector A6697 in the 1st stage only. As well tests F5 and F10 were done at the targeted mid grind size (80% passing 105 microns) and replaced SIPX and A208 with PAX and CuSO4. All of the kinetic tests were performed at natural pH, which for the various composites ranged from pH 8.2 to 8.9. The detailed test results are provided in Appendix 3 and summarized in Tables 4.2 a&b, below.

Table 4.2a: Kinetic Flotation Response – Low Sulfur Comp.

Test No.	Comment	Grind P80 u	Calc Hd Au (g/t)	Tail Grade		Bulk Recovery (%)		
				Au,g/t	%S	Mass	Au	S
F1	V. fine grind, 1st stg A6697	51	1.26	0.04	0.05	37.8	98.0	98.9
F2	Fine, SIPX/A208 only	71	0.91	0.08	0.09	30.1	93.9	97.7
F3	Mid grind, SIPX/A208 only	113	0.75	0.07	0.08	28.1	93.8	98.1
F4	Coarse with SIPX/A208	170	0.85	0.07	0.11	24.6	93.8	97.5
F5	As F3 but PAX CuSO4	115	0.91	0.10	0.11	26.7	91.9	92.5

Table 4.2b: Scoping Flotation Response – High Sulfur Comp.

Test No.	Comment	Grind P80 u	Calc Hd Au (g/t)	Tail Grade		Bulk Recovery (%)		
				Au,g/t	%S	Mass	Au	S
F6	V. fine grind,1st stg A6697	53	1.97	0.02	0.04	32.2	99.3	99.3
F7	Fine, SIPX/A208 only	70	2.23	0.04	0.04	29.6	98.7	99.4
F8	Mid grind, SIPX/A208 only	113	1.91	0.04	0.05	26.3	98.5	99.0
F9	Coarse with SIPX/A208	183	1.96	0.08	0.10	23.2	96.9	98.0
F10	As F3 but PAX CuSO4	~115	2.02	0.04	0.03	28.0	98.8	99.5

Bulk gold recoveries were very good at >94% over a wide range of primary grinds, with the higher grade composite blend showing slightly higher recoveries of >96%. The results showed similar gold tailing losses for various grinds ranging from approximately P80 of 50 to 180 microns. The high mass pull and sulfur recovery indicates that upgrading the gold grade during flotation cleaning would be difficult unless gold bearing minerals can be selectively floated or alternately the bulk of the sulfides can be depressed without incurring significant gold losses.

The use of a gold specific collector (Cytec A6697) in tests F1 and F6 indicated gold to some extent floated preferentially to pyrite. As a result additional work was performed using this collector, as discussed below.

Tests F5 and F10 used a less selective collector PAX, along with CuSO4 as an activator at a moderate grind. Comparing respectively to the SIPX/A208 collector combination in F3 and F8 did not show an improvement to gold recovery or decrease in tailing losses. Consequently no further work with these PAX and CuSO4 was undertaken.

Following the kinetic studies the test program investigated cleaning flotation with two separate goals. One was to produce a lower grade concentrate with the highest possible recovery for feed to on-site cyanidation. A second procedure looked at producing a higher grade concentrate suitable for shipment to off-site treatment.

4.2.2 Open Cycle Cleaning Flotation

Most of the earlier cleaning tests were performed on a composite with higher sulfur content (Comp. 08-4) and which also had the most sample weight available. The initial variability work included changing the primary grind, investigating the use of the gold specific collector identified in kinetic testing, and implementing a higher pH with the SIPX and A208 collectors in order to depress the pyrite. Sodium hydroxide instead of lime was used as a pH modifier as the latter can have a tendency to depress gold. The summary of these cleaner testing results performed on Comp 08-4 is provided in Table 4.3 below.

Table 4.3: Comp 08-4 - Response to Float Modifications

Test No.	Collector Types Types	pH	Grind	Gold Grade (g/t)			Bulk Recovery	
			P_{80} (µ)	Hea	Tail	Conc	%S	%Au
F12	A6697	natura	100	1.90	0.60	123	16	73
F14	SIPX & A208	natura	99	1.91	0.15	42	98	95
F15	A6697	natura	55	1.80	0.40	94	28	85
F16	A6697 / SIPX & A208	nat./10	50	1.70	0.40	80	90	86
F20	SIPX & A208	10	100	1.73	0.20	131	88	91

The results for Comp. 08-4 confirmed the earlier studies that the highest bulk precious metal recoveries are performed by maximizing sulfide recovery (F14), but that this would not allow for high upgrading of the concentrate. This procedure is therefore considered more appropriate if on-site cyanidation of the concentrate is to be considered. A number of related cyanidation procedures on concentrate produced using these flotation procedures on other composites is discussed in the following section of this report.

The results in tabulated above, as well as two tests F11 and F21 (Appendix 3) performed on a lower sulfur composite (Comp. 08-7) suggested two procedures show promise for further upgrading of the concentrate. The first procedure used the gold selective collector A6697 (F12, F15), and the second procedure increases the pH during flotation to depress pyrite (F20). On this particular composite the test F20 procedure appeared to allow for better recovery while permitting production of a similar grade of concentrate. Finer grinding in F15 as compared to F12 reduced gold tailing losses from 0.6 g/t to 0.4 g/t but with a somewhat lower concentrate grade. Scavenging with centrifugal concentration in test F15 reduced gold losses further to 0.2 g/t, which was similar to the losses in test F20. Combining the two procedures in test F16 and scavenging with the less selective collector at high pH did not reduce losses.

Consequently, these two procedures were forwarded as methods for further investigation. This included testing on other composites as well as on a master composite (MC1). MC1 was a blend of the remaining material from the 8 original composites (see Appendix 2 for blend ratio and head assay).

A description of the two flotation procedures for pursuing production of a higher grade concentrate for off site sale is as follows;

Method 1. The elevated pH procedure used a pH 10 during roughing, with a combination of sodium isopropal xanthate (SIPX) and A208 as the collector. A moderate primary grind size targeting a P80 of 74 microns was used. While a coarser grind can be considered for liberation of sulfides from gangue, the finer grind was chosen in relation to gold liberation within sulfides (although neither the primary or regrinding particle size has been fully optimized during this program). The combined rougher concentrate is reground and cleaned in three stages increasing the pH stage wise to 11.5.

Method 2. The gold specific collector A6697 was used at a natural pH cleaning in three stages. Similar grind and rougher flotation times were employed as were used in method 1. A final scavenging step was undertaken on the flotation tailing using a three stage Falcon centrifugal concentrator.

The two methods were performed on composites 08-1 08-3, 08-8 and MC1. The results for Method 1 are summarized in Table 4.4 below. Also included in Table 4.4 are two earlier tests (F20, F21) run on different composites at a coarser primary grind size (-100 to 122 μ) and with only two stages of cleaning.

Table 4.4: Open Cycle Flotation Cleaning at Elevated pH (Method 1)

Test No.	Comp. ID / Types	Calc. Head		Tailing	2nd or 3rd Cleaner Concentrate				Bulk Recovery	
		Au, g/t	%S	Au, g/t	%Mass	%S	Au, g/t	Ag, g/t	%S	%Au
F20	08-4	1.73	7.75	0.20	0.9	38.9	131	387	88	91
F21	08-7	2.55	2.29	0.37	2.7	44.3	76.0	135	91	88
F22	08-1	0.41	2.08	0.03	1.3	47.0	24.7	176	96	94
F23	08-3	1.70	4.28	0.10	1.3	40.9	103	388	96	95
F24	08-8	2.27	4.59	0.42	0.8	49.6	203	545	95	85
F25	MC1	1.77	4.11	0.19	2.3	45.1	62.8	213	94	92
F26* MC	MC1 (grav.)	1.89	3.89	0.23	0.9	41.0	135	454	94	90

*F26 is the same as F25 but used one stage Falcon gravity recovery prior to flotation

The data shows differences in recovery and final concentrate grade depending on the sample. Lower gold head grades and higher sulfur content in the feed tend to lower a final concentrate grade, which confirms earlier work. Test F22 was performed on a low grade sample assaying ~0.4 g/t gold which resulted in low concentrate grades, but recovery was maintained. Check analyses were performed on some tailing streams which altered moderately the recoveries so more confirmation work is required as the program develops. Continuing work on the lower grade samples will assist in evaluating the economic cut-off grade for ore going to the mill, or alternately to low grade stockpiles, or waste dumps.

The F24 results show a higher final concentrate grade, as well as having the highest tailing losses. Examination of the detailed results may explain some of the reasons. This composite (08-8) had a higher calculated head and a slightly coarser primary grind size. Importantly based on the technicians' observations the final cleaning stage time was reduced from 4 minutes to 3 minutes allowing less mass and therefore less sulfides into the final conc. The corresponding recovery into the final bulk concentrate was lower due to higher final tailing grade losses. This may be a result of variations in mineralogy, higher head grade, and slightly coarser grind.

Examining data from tests F25 and F26, both of which were performed on composite MC1, show that the F26 procedure produced a higher concentrate grade. This is unexpected as both test procedures were the same, except that

F26 used gravity recovery prior to flotation, which should result in a lower final float concentrate grade. Examination of the detailed results may offer an explanation in that it shows that the amount of NaOH used was considerably higher for F26. While the resulting pH of both tests were similar, the high alkalinity can make pH measurement more difficult and differences in the pH electrodes more pronounced. The higher caustic addition may further assist pyrite rejection without gold losses and suggests this may be a parameter to test in future optimization studies.

The F26 data provided a cleaned gravity concentrate of 251 g/t Au, recovering approximately 12.6% of the gold. However, with no improvement in reducing tailing losses (as compared to F25) and with a relatively low grade for the cleaned gravity concentrate, it would suggest that gravity procedures prior to flotation will not improve overall recovery, at least on this particular composite. This agrees with earlier studies performed on other samples.

Alternate use of gravity recovery was evaluated by incorporating a three stage centrifugal procedure to scavenge the flotation tailing. A gold specific collector (A6697) was used at natural pH during flotation. These results are summarized in Table 4.5.

Table 4.5: Gold Specific Float Collector with Gravity Scavenging (Method 2)

Test No.	Comp. ID / Types	Head	Tail Au (g/t)		3rd Cleaner Float Concentrate				% Ro. Au Rec.	
		Au, g/t	Float	&Grav.	%Mass	%S	Au, g/t	Ag, g/t	Float	&Grav.
F27	08-1	0.45	0.02	0.04	1.2	46.1	28.9	215	~95	~99
F28	08-3	1.53	0.30	0.12	1.3	36.8	91.2	419	83	94
F29	08-8	2.22	0.48	0.15	0.5	46.2	318	673	81	88
F30	MC1	1.91	0.40	0.14	1.1	38.7	137	476	82	90

*Grav. = data after 3 stage Falcon gravity procedure scavenging flotation tailing

While the number of samples is limited, the data suggests using flotation method 2 trends to a slightly higher open cycle concentrate grade, and lower recovery as compared to method 1 summarized in Table 4.4. The use of gravity scavenging allows for a similar recovery for both the methods.

The results for method 2 in Table 4.5 show considerable variation between samples using the same procedure. This is likely due to variation of the gold head grade as well as the gold to sulfur ratio, with higher gold grades and Au:S ratio's favoring higher concentrate grades. Despite a low head grade test F27 shows a higher recovery due to a very low tailing grade after flotation. The gold tailing grade after flotation is actually higher, than after a gravity scavenging step. Although the 0.02 g/t difference is within the margin of analytical error it would indicate the flotation recovery is actually lower than reported for test F27. It appears that a bulk recovery in the low eighty percent range can be expected in using the elevated pH flotation approach and that this recovery is relatively independent of the head grade.

Using only the gold specific collector the tailing losses remained higher than desired ranging from 0.3 g/t to 0.5 g/t for the majority of the samples. This may be improved by finer primary grinding (see earlier tests F12 vs. F15 on Comp. 08-4). Scavenging with gravity procedures also reduces these tailing losses. Following three stage gravity testing to simulate a Falcon continuous centrifugal concentrator, the tailing losses dropped to ≤ 15 g/t Au, which were lower than method 1. This resulted in modestly improved open cycle bulk recoveries for method 2, similar to those achieved by method 1. Locked cycle testing was required to better evaluate the two procedures and is discussed in the following section.

4.2.3 Locked Cycle Flotation

The two methods tested for locked cycle study were based on the open cycle test program. Each method was subjected to a 6 cycle locked cycle test on blended composite MC1. The detailed procedure and results are provided in Appendix 4.

The high pH method (test FLC1) was similar to test F25 but used an additional fourth cleaning stage. The rougher concentrate and 1st cleaner scavenger concentrate were recycled to the regrind mill. The rougher scavenger concentrate was recycled back to primary grinding.

A flowsheet of circuit is provided in Figure 4.2, below.

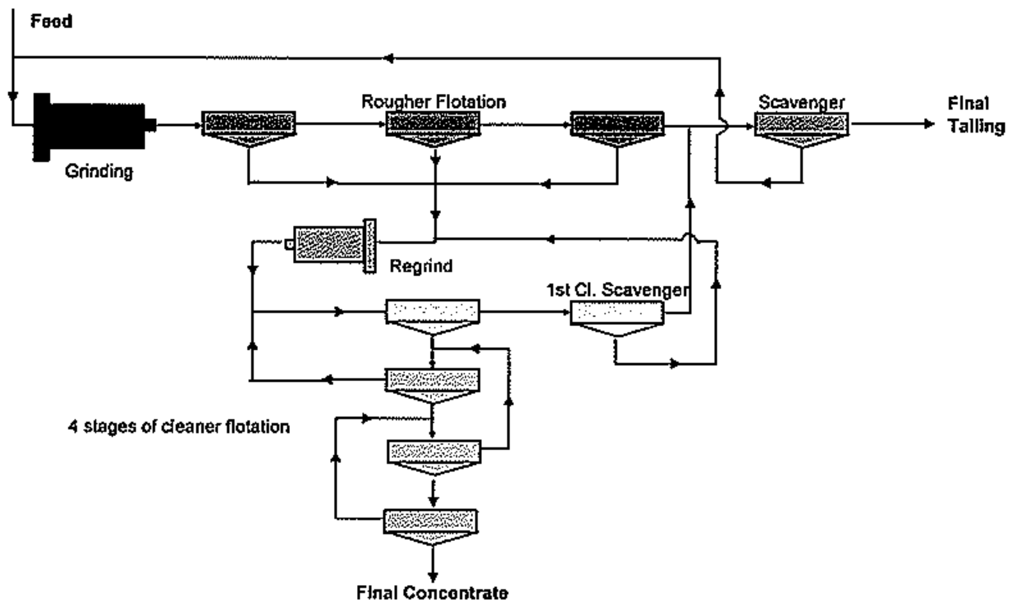


Figure 4-2: Locked Cycle Flotation Circuit

The FLC1 results for the full 6 cycles not including recycle streams gave a gold recovery of 87% and a concentrate grade of 110 g/t Au. The tailing grades for the six cycles fluctuated between 0.07 g/t to 0.17 g/t and did not show an escalating trend for mass or grade. Averaging the last three cycles and assuming recycle streams were stabilized indicated a gold recovery of 93% maintaining the concentrate grade of ~110 g/t Au, with a tailing grade of ~0.13 g/t Au. The calculated gold head grade was 2.06 g/t. Silver grades in the final float tailing were below detection limit, which prevents calculating an accurate silver recovery, but which is greater than 87% and which would likely exceed the gold recovery. The last three cycles showed an average silver concentrate grade of 259 g/t. Deleterious elements in the final cycle concentrate were 0.11% As, 0.02% Sb, and 34 ppm Hg (by cold vapor).

The FLC1 results show a good response for precious metals recovery and close agreement with the open cycle testing. The high pH procedure provides confidence that acceptable precious metal recoveries and concentrate grades can be achieved using this procedure. Future test work might consider discharging the first cleaner tailing directly, since scavenging this circuit results in only minor improvements to recovery and direct discharge would allow for faster stabilization of the circuit. An actual operating circuit might consider flexibility in design to allow for either option, with regard to the sulfide content related to environmental (acid rock drainage) considerations.

The second locked cycle program (test FLC2) used a gold specific collector at natural pH and with gravity scavenging, similar to open cycle test F30. The flotation circuit for FLC2 is the same as shown in Figure 4-2, above for FLC1. The principal difference is the reagents used and incorporation of the Falcon gravity circuit scavenging the float tailing (see flowsheet in Appendix 4). The rougher Falcon concentrate was cleaned by hand panning to simulate tabling. The gravity cleaner tailing was then recycled to the regrind mill to join the bulk rougher concentrate.

Examination of the results shows a high recirculation load from the gravity tailing being sent to the regrind mill with the FLC2 procedure. Consequently, the cleaning circuit mass did not stabilize, and the recovery is difficult to predict. Future test work might consider floating the gravity tailing separately to alleviate this issue. The gold grades in the float concentrate were lower than the open cycle work, and gravity concentrate grades did not upgrade well by panning. The FLC2 is a less conventional procedure and is shown to be more problematic to operate, with no apparent benefit as compared to FLC1 procedure.

4.3 CYANIDATION

4.3.1 Whole Ore Cyanidation

Whole ore cyanidation was tested on two composite blends using two separate procedures. The two composite blends that were tested for whole ore cyanidation consisted of a 1:1 weight ratio of Comp. 08-1 & 08-2, and on a 1:1 ratio of Comp. 08-5 & 08-6. Both procedures used a targeted grind of P_{80} 74 microns (200 Tyler mesh) with a cyanide concentration of 2 g/L NaCN, at pH ~10.5, for 96 hours. The second method included a modification in which carbon in leach (CIL) procedures in which 20 g/L of activated carbon is added to absorb cyanide soluble gold. CIL was tried to counter the presence of natural organic carbon that may absorb gold dissolved during cyanidation thereby lowering recovery. The CIL procedure included addition of sodium lauryl sulphate (SLS), as a blinding agent on the natural carbon, before adding the activated carbon.

Two additional CIL tests (CIL3, CIL4) were conducted on each of the two composite blends at a slightly finer grind following gravity pretreatment. Gravity pretreatment was conducted to observe if it might reduce losses of precious metals during CIL.

The cyanidation results are provided in Appendix 5, and summarized for the whole ore procedures in Table 4.6.

Table 4.6: Whole Ore Cyanidation Results

Comp.	Test	Grind	Calc. Head		Tailing Grade		Recovery	
ID	No	P_{80} µ	Au, g/t	Ag, g/t	Au, g/t	Ag, g/t	Au	Ag
08-1&2	C1	70	1.40	6.8	0.15	2.2	89.3	68.3
08-1&2	CIL1	71	0.84	5.3	0.11	1.5	86.8	71.5
08-1&2	CIL3	64	0.99	4.7	0.09	0.5	~92	~91
08-5&6	C2	67	2.08	17.1	0.52	6.5	75.2	61.9
08-5&6	CIL2	67	1.79	14.8	0.27	5.5	84.9	62.9
08-5&6	CIL4	52	1.97	13.2	0.14	2.5	~94	~83

The addition of activated carbon (CIL) assisted in reducing a minor preg robbing effect of the ore. Based on the tailing assays the results suggest that optimized cyanidation procedures include prior gravity concentration to minimize losses, which would likely also assist in reducing the required leach retention time. Incorporating these procedures (CIL 3 & 4) resulted in cyanide tailing losses of less than 0.15 g/t Au, which approximately correlates to the gold losses experienced with the optimized flotation procedures. Silver tailing losses are significantly higher in cyanidation than with flotation.

4.3.2 Cyanidation of Flotation Concentrate

Several of the flotation concentrates produced from the composites were evaluated for cyanide leaching of precious metals. These were cleaned to produce a low grade concentrate in order to maximize flotation recovery. Coarser gold in the resulting concentrates was removed by hand panning and the pan tailing subjected to 96 hour CIL cyanide procedures. The detailed results are included with the corresponding flotation tests (F17 to F21) in Appendix 3, and summarized in Table 4.7 below.

Table 4.7: Float Concentrate Cyanidation Results

Comp. ID	Test No.	Grind P_{80} µ	Assayed Head		Tailing Grade		% CN Recovery	
			Au, g/t	Ag, g/t	Au, g/t	Ag, g/t	Au	Ag
08-2	CILF17	55	19.5	55.8	1.52	24.6	92	56
08-3	CILF18	63	19.7	86.3	1.84	41.1	90	52
08-5	CILF19	55	40.1	224	2.42	41.5	94	81
08-4	CILF20	n/a	117.8	382	4.00	139	96	64
08-7	CILF21	40	77.0	98.3	4.72	11.2	93	88

Due to the elevated grade of feed, the gold recoveries are high, although losses in the residue vary between 1.5 to 4.7 g/t Au. As with the whole ore cyanidation silver recoveries are significantly lower than gold. Overall losses in flotation and to a lesser extent in the gravity circuit need to be included. This would best interpreted with locked cycle testing but can be expected to be in the high eighties percent range. The results show that cyanidation of flotation concentrate can be considered as a process option.

4.4 TAILING AND WASTE CHARECTERIZATION

Preliminary settling tests and acid base accounting was undertaken on composite MC1 tailing generated from locked cycle testing (see Appendix 6). The tailing without settling aids showed poor settling rates and high observed turbidity in the supernatant. The use of settling aids significantly improved these characteristics. After beaker scoping evaluation a flocculent was selected for 2L cylinder settling testing. The flocculent dosage requirement was shown to be reduced if the tailing pH was increased with hydrated lime to pH 11. NaOH had been used to increase the slurry to pH 10 during bulk flotation, but the tailing dropped to pH ~9 prior to the settling test. Lime is a preferred as a pH modifier but was not used during flotation testing, as it has potential to be a depressant for gold. The pH modifier type and dosage should be further evaluated in tailing treatment in regard to both water quality and potential to depress gold since water from the tailing will be re-circulated for use as process makeup water. As the site is in an area of high precipitation the need for fresh makeup water is anticipated to be low or not required.

In the 2 L settling testing the use of 30 mg/L Percol 368 at pH 11 (using hydrated lime) significantly improved the supernatant clarity and settling rate as compared to no flocculent addition. The flocculated settling rate was provided at 8.2 m/day or 0.34 m^2/tpd as determined by the Modified Coe and Clevenger calculation method (see Appendix 6). The floc size remained small indicating further improvements can likely be made.

Observations during flotation testing indicated the rougher concentrate settling and filtering characteristics were poor for some composites. Future work will need to include settling and filtering tests of concentrate once grinding requirements become better understood.

Acid base accounting (ABA) testing was performed on tailing generated from the two locked cycle tests showing similar results using the modified Solbek procedure. Results from FLC1 tailing which is considered to currently most closely represent the process circuit provided a net neutralization potential of 24.6 kg CaCO3 equivalent. The neutralization potential to acid potential ratio was 1.4.

Additional ABA's were conducted on assay rejects provided to ALS Chemex Laboratories. The samples were selected as a scoping study to evaluate waste rock and low grade material that could potentially be stockpiled in dumps during operations. These results are summarized in Appendix 6 and indicate that some of the waste rock and low grade material has a negative neutralization potential. Therefore ABA and other environmental studies relating to acid rock generating potential will be a critical component of further laboratory studies as the project advances.

5.0 RECOMMENDATIONS

Further laboratory testing is recommended when proceeding to pre-feasibility assessment. Based on the results of this test program and discussions with the client the next phase of metallurgical testing should focus on flotation in order to evaluate producing a gold silver concentrate for sale to smelters or other custom treatment facilities. This is recommended based on the encouraging results achieved with the preliminary flotation work, as well as to the potential permitting and social issues relating to use of cyanidation on site. The project is located in an area of high precipitation (both rainfall and snowfall), near the international boundary with Alaska, and in the watershed of salmon bearing streams and rivers. These issues would be likely to complicate and delay an application of an operating circuit that uses cyanidation.

Flotation, in addition to being a more benign process than cyanidation, is seen to have several advantages including more flexibility in dealing with variable feed (such as higher base metals content) and the presence of preg robbing organic carbon. Flotation also produces a higher silver recovery and removes some sulfide minerals from the system where there is ARD potential. Concentrate characteristics on samples tested to date are considered encouraging. If further testing indicates an average concentrate grade that is too low in precious metals, or that contains deleterious elements, then alternate processing options may need to be considered including a more detailed look at cyanidation. If the future resource modeling shows zones of high base metals such as lead and zinc, then modifications to the flotation circuit may be required.

Based on the current resource the anticipated a milling throughput is reported to be approximately 6,000 tonnes per day from open pit mining. The next phase of the metallurgical test program should include more comminution testing. Comminution is currently anticipated to consist of three stages of crushing, and one stage of grinding. The third stage of crushing may incorporate use of either gyratory or high pressure grinding rolls (HPGR). Grinding is anticipated to be by ball milling in closed circuit, although it is possible rod milling or semi-autogenous grinding (SAG) would be considered, depending on further metallurgical data and preliminary engineering studies. The conceptual flotation flowsheet would generally follow that of locked cycle test FLC1 discussed previously, which uses elevated pH to depress pyrite. The combined rougher concentrate would be reground and cleaned. The final concentrate would be thickened and filtered for sale to smelter(s). The rougher scavenger and cleaner scavenger concentrates would be recycled and bulk and 1st cleaner scavenger tailings would be disposed either separately or together depending on environmental and logistical considerations. In this context the following test work is recommended for the next phase of the laboratory testing program;

COMMINUTION / MATERIAL HANDLING

- Bond Crusher Impact Testing
- Bond Ball Mill Work Index (vary with resource and mineralogy)
- Bond Abrasion Tests
- HPGR evaluation
- Engineering Data (angle of repose, bulk density, comprehensive strength, particle size distributions)
- Bond Rod Mill Work Index (optional to be determined)
- Breakage Characterization - JK Tech Drop (optional to be determined)
- SMC SAG Mill (optional to be determined)

FLOTATION

a) Kinetic Flotation (on blended composites)

- pH variation in bulk float with lime and separately with soda ash and caustic (vary between natural and pH 11).
- primary grind size (vary at both natural and elevated pH).
- Collector type
- Collector dosage
- Gravity pre-concentration at optimized conditions (compare with and without)

b) Open Cycle cleaning (using optimized bulk float)

- Regrind (size including ultrafine grinding)

- Investigate pH modifiers (moderate pH with lime and caustic) and aggressive pH (caustic only) for pyrite depression
- Investigate various other pyrite depressants
- Flotation time and number of cleaning stages, optional to include column flotation investigation
- Vary collector, modifier, frother dose and type

c) Variability Testing

A large number of open cycle cleaning flotation tests will be required using the established optimum conditions. The number of the tests required will depend on the resource model representing tonnage and resource variation, which is still being developed. These tests will represent the various spatial and depth profiles of the deposit, as well as investigating variations in grade, lithology, and mineralogy profiles. Some allocation for basic mineralogy (petrographic, XRD etc.) is included but additional budget for SEM characterization of feed and flotation products may be required.

d) Locked Cycle Flotation Testing

These would be performed on representative composite samples under the developed optimized conditions. It would include detailed concentrate and tailing characterization, including settling and filtration characteristics. Response of the concentrate to cyanidation would also be briefly investigated.

e) Confirmation Testing

A second process testing laboratory will be selected for confirmation studies using the developed optimized float conditions.

TAILING CHARECTERIZATION

- Settling Tests
- Investigate separate or combined bulk and 1st cleaner tailing disposal
- Geotechnical Tests (particle size distribution, rheology, SG, etc)
- ABA, humidity cells, recycle water quality

The cost for the above mentioned laboratory work is estimated at $200,000. These costs do not include charges for sample collection or related engineering studies. Samples should be obtained from fresh or recently archived drill core. Some samples for comminution testing will require large diameter drill core (or less preferably might be obtained from trenched surface samples).

The recommended test program assumes the principal product for the Silver Coin Project is gold, with silver as a minor contributing by-product. Ongoing exploration may show by-product potential for zinc, lead, or other metals, which has not been included in these recommendations. Some studies relating to acid rock drainage potential of flotation tailing have been included, but other environmental tests including for waste rock and low grade stockpile material is not included. This work may require ARD kinetic testing for feasibility evaluation, which is a long lead time item and should be discussed in context with the selected environmental consultant.

6.0 CONCLUSIONS

Composite samples from the Silver Coin Project were evaluated for metallurgical testing to develop a conceptual process flowsheet to recover gold, with silver as a byproduct. Head assays on eight composites that were tested showed a head grade range of 0.41 g/t to 2.88 g/t Au; with 2.3 g/t to 22.7 g/t Ag. The sulfur content ranged from 2.2% to 8.4%, with 0.02% to 0.53%Pb, and 0.03% to 1.40% Zn.

The lead and zinc content have generally been too inconsistent and at average grades that are considered too low to include for byproduct credit. However, some zones of the resource have reported elevated zinc and lead which may offer byproduct potential and further consideration to this will be given with the developing resource model. Ore zones with consistently higher levels of base metals were not included as part of this test program. Their inclusion could modify the selected treatment flowsheet.

Froth flotation and tank cyanidation including the use of gravity treatment were evaluated in the laboratory testing. Procedures were developed to investigate precious metal recovery method using either flotation or cyanidation, including cyaniding a flotation concentrate. Both cyanide and flotation procedures provided similar gold recoveries varying from 85% to 95%, depending on the sample tested. Cyanidation resulted in lower silver recoveries than flotation, generally recovering between 62% to 83% Ag. Gravity pre-treatment is recommended prior to cyanidation, but is not necessarily required prior to flotation (based on the samples that were tested).

Depending on the treatment procedure selected the primary grind requirements are moderate to coarse. Regrinding of the bulk concentrate prior to flotation cleaning improves the concentrate grade but more work is required to establish the optimized grind. Bond Ball Mill Work Index testing using a closing screen size of 105 microns (150 Tyler mesh) indicated an ore hardness of 18 to 19 kWh/tonne, which is considered moderately hard. Further comminution studies including more grinding studies related to resource variability, along with abrasion testing, and crushing work indices will be required as part of any future pre-feasibility evaluation.

The highest cyanidation recovery on the whole ore included the use of gravity pretreatment and carbon in leach (CIL) procedures. Resulting gold recoveries are approximately 90%, with corresponding losses of less than 0.15 g/t Au in the tailing. Silver losses by cyanidation were considerably higher ranging 0.5 g/t to 5.0 g/t. Improvements to the cyanide precious metals recovery did not appear to benefit from fine grinding, but further evaluation on grind sensitivity is recommended if this treatment method is pursued. Cyanidation of the flotation concentrate was also shown to achieve recoveries in the low nineties percent range. Depending on technical and environmental issues developed during future prefeasibility testing, the cyanide leaching of a flotation concentrate may be an option for further evaluation.

Standard bulk flotation provided excellent precious metal recoveries of greater than 95%, but had poor concentrate upgrading characteristics due to high pyrite content. Scoping procedures indicated the pyrite could be partially rejected with low gold losses. Consequently, two flotation methods were selected for more detailed evaluation to reject pyrite in order to improve the concentrate upgrading characteristics.

The first float method used for more detailed evaluation incorporated sodium hydroxide to increase pH in order to depress pyrite. Lime was not used as a pH modifier as it can have a tendency to depress gold. Future testing should include evaluating lime as a pH modifier during rougher flotation as it is less costly and improves settling characteristics of the bulk tailing. The open cycle results using elevated pH showed a final concentrate grades varying between 63 g/t to 203 g/t Au, with 88% to 95% bulk float gold recovery for the various samples. These numbers held up well with the locked cycle test on the composite blend (MC1), which resulted in a final concentrate grading 110 g/t Au and a recovery approaching 90%. Silver in the concentrate was 269 g/t and was less than the assay detection in the final tailing (< 0.5 g/t), which corresponds to a recovery of >89% in the locked cycle test. When averaging the final three cycles and proportionally distributing the recycle streams to the tailing and product; a gold recovery of 93% is projected on a calculated head of 2.06 g/t Au. Recovery will decrease with decreasing head grade. The second locked cycle flotation procedure used a gold selective collector to improve precious metal grades in the concentrate. Despite encouraging results during open cycle testing, it was considered less beneficial than those of the first locked cycle procedure and is not recommended for further evaluation.

A decision on pursuing a flotation only process route would need to include evaluating the transportation issues and contracts with toll treatment facilities. The precious metal grade requirements, and limits on deleterious elements will be critical parameters to establish. Mercury needs to be evaluated by cold vapor assay methods, and along with arsenic was shown to vary considerably in the feed composite samples used in this test program. The concentrate produced from locked cycle testing contained approximately 0.11% As, 0.02% Sb, and 34 ppm Hg.

Continued variability and locked cycle studies should be performed as the project progresses focusing on samples that represent the majority of the expected resource grade and geology. Based on the current understanding of the project it is recommended the next phase of test work focus on bulk flotation and cleaning at an elevated pH with continued reagent optimization. This would include better defining the primary grind and regrind requirements.

Observations during flotation and scoping solid / liquid separation studies showed poor settling and filtering qualities for some of the composites. Separate handling of the 1st cleaner scavenger tailing may offer potential environmental advantages for disposing higher sulfide tailing subaqueous or placed beneath the lower sulfide bulk tailing. Related environmental studies should also be undertaken. Acid base accounting (ABA) on low grade ore (possible storage dumps), and waste rock shows some zones have the potential to generate acid. Additional ABA and related kinetic studies will be required as the program advances.

Ongoing environmental and process testing is recommended that would be dependent on further project development factors. These factors include variability within the resource model (grade, mineralogy, lithology), as well as evaluation of site engineering issues (geotechnical, infrastructure, water balance, environmental, etc.), permitting and other items that can affect economics.

Based on the samples tested a conceptual treatment flowsheet was developed for the Silver Coin Project. This flowsheet consists of regrinding and cleaning a bulk flotation concentrate, at an elevated pH. The data indicates for ore similar to the MC1 blended composite with a head grade of approximately 2 g/t Au, a gold concentrate of ~110 g/t with ~90% recovery could be expected.

7.0 STATEMENT OF QUALIFICATIONS AND LIMITATIONS

I, **Frank R. Wright** do hereby certify:

I am a Consulting Metallurgical Engineer, practicing at 427 Fairway Dr., North Vancouver, BC, Canada

I graduated with a Bachelor of Science, in Metallurgical Engineering, obtained in 1979 from the University of Alberta., Edmonton Alberta. I also obtained a degree of Bachelor of Business Administration, Simon Fraser University, Burnaby BC, in 1984.

I have continuously practiced my profession for 25 years, the last 12 years as a self-employed consulting engineer developing process treatment circuits for the mineral industry.

I am a registered member in good standing with the Association of Professional Engineers and Geoscientists of British Columbia.

This report titled "Metallurgical Study on the Silver Coin Gold Project" and dated January 8, 2009 has been issued to Pinnacle Mines Ltd. (Pinnacle). This report relates to a preliminary process study of the Silver Coin mineral exploration project, and is intended for use by the professional management team of Pinnacle. Any other use of, or reliance on, this report by any third party is at that party's sole responsibility. The reported work is based on studies performed on mineral samples supplied by, and on laboratory results provided by, other parties, as specified in this report. Related technical evaluations have also been performed and supervised by other parties.

Signed this 8th day of January, 2009, at North Vancouver, BC

Frank Wright, P.Eng.

APPENDIX 1

SAMPLE RECEIVE AND COMPOSITING

SAMPLE RECEIVING LOG SHEET

Receiving Date: 21-Jul-08	**Project No:** 0805107
Carrier: Canadian Freightways	**Client:** Pinnacle Mines Ltd.
Receiver: Roxanne/Joe	**Page:** 1 of 5

Count	Sample Label		Container Type	Sample Type (C, R, P, Sl, S)	Wet /Dry	Top Size	Weight (kg)
1	2006-106	1086	Bag	C	Wet	6"	2.46
2	2006-106	1087	Bag	C	Wet	6"	2.34
3	2006-106	1088	Bag	C	Wet	5"	2.36
4	2006-106	1089	Bag	C	Wet	5"	2.34
5	2006-106	1090	Bag	C	Wet	5"	3.45
6	2006-106	1091	Bag	C	Wet	5"	1.38
7	2006-106	1092	Bag	C	Wet	5"	1.19
8	2006-109	1839	Bag	C	Wet	6"	1.25
9	2006-109	1840	Bag	C	Wet	5"	1.16
10	2006-109	1841	Bag	C	Wet	4"	1.18
11	2006-109	1842	Bag	C	Wet	4"	1.11
12	2006-109	1843	Bag	C	Wet	6"	1.24
13	2006-109	1844	Bag	C	Wet	6"	1.10
14	2006-109	1845	Bag	C	Wet	6"	1.17
15	2006-109	1846	Bag	C	Wet	6"	1.25
16	2006-109	1847	Bag	C	Wet	6"	1.35
17	2006-109	1848	Bag	C	Wet	6"	1.61
18	2006-109	1849	Bag	C	Wet	6"	0.88
19	2006-109	1850	Bag	C	Wet	6"	1.28
20	2006-109	1851	Bag	C	Wet	6"	1.16
							31.3

Note :

Core, Rock, Pulp, Slurry, Solution

SAMPLE RECEIVING LOG SHEET

Receiving Date: 21-Jul-08	**Project No:** 0805107
Carrier: Canadian Freightways	**Client:** Pinnacle Mines Ltd.
Receiver: Roxanne/Joe	**Page:** 2 of 5

Count	Sample Label		Container Type	Sample Type (C, R, P, Sl, S)	Wet /Dry	Top Size	Weight (kg)
21	2006-109	1852	Bag	C	Wet	6"	1.86
22	2006-115	3932	Bag	C	Dry	5"	1.54
23	2006-115	3933	Bag	C	Dry	6"	1.66
24	2006-115	3934	Bag	C	Dry	6"	1.58
25	2006-115	3935	Bag	C	Dry	5"	1.52
26	2006-115	3936	Bag	C	Dry	5"	1.49
27	2006-115	3937	Bag	C	Dry	5"	1.60
28	2006-115	3938	Bag	C	Dry	6"	1.64
29	2006-115	3939	Bag	C	Dry	7"	1.57
30	2006-115	3940	Bag	C	Dry	5"	1.26
31	2006-115	3941	Bag	C	Dry	6"	1.40
32	2006-115	3942	Bag	C	Dry	7"	1.59
33	2006-115	3943	Bag	C	Dry	7"	1.53
34	2006-115	3944	Bag	C	Dry	5"	1.37
35	2006-115	3945	Bag	C	Dry	6"	0.96
36	2006-115	3946	Bag	C	Dry	6"	1.51
37	2006-121	5389	Bag	C	Dry	4"	1.18
38	2006-121	5390	Bag	C	Dry	5"	1.41
39	2006-121	5391	Bag	C	Dry	6"	1.51
40	2006-121	5392	Bag	C	Dry	6"	1.42
							29.6

Note :

Core, Rock, Pulp, Slurry, Solution

SAMPLE RECEIVING LOG SHEET

Receiving Date: 21-Jul-08	**Project No:** 0805107
Carrier: Canadian Freightways	**Client:** Pinnacle Mines Ltd.
Receiver: Roxanne/Joe	**Page:** 3 of 5

Count	Sample Label		Container Type	Sample Type (C, R, P, Sl, S)	Wet /Dry	Top Size	Weight (kg)
41	2006-121	5393	Bag	C	Dry	7"	1.58
42	2006-121	5394	Bag	C	Dry	6"	1.33
43	2006-121	5395	Bag	C	Dry	7"	1.76
44	2006-121	5396	Bag	C	Dry	6"	1.57
45	2006-121	5397	Bag	C	Dry	6"	1.47
46	2006-121	5398	Bag	C	Dry	5"	1.55
47	2006-121	5399	Bag	C	Dry	7"	1.37
48	2006-121	5400	Bag	C	Dry	6"	1.60
49	2006-121	5401	Bag	C	Dry	5"	1.56
50	2006-121	5402	Bag	C	Dry	6"	1.73
51	2006-121	5403	Bag	C	Dry	7"	1.61
52	2006-121	5404	Bag	C	Dry	6"	1.69
53	2006-136	3066	Bag	C	Wet	5"	1.31
54	2006-136	3067	Bag	C	Wet	6"	1.32
55	2006-136	3068	Bag	C	Wet	5"	1.17
56	2006-136	3069	Bag	C	Wet	6"	1.29
57	2006-136	3070	Bag	C	Wet	5"	1.24
58	2006-136	3071	Bag	C	Wet	5"	1.60
59	2006-136	3072	Bag	C	Wet	7"	1.96
60	2006-136	3073	Bag	C	Wet	5"	1.37
							30.1

Note :

Core, Rock, Pulp, Slurry, Solution

SAMPLE RECEIVING LOG SHEET

Receiving Date: 21-Jul-08	Project No: 0805107
Carrier: Canadian Freightways	Client: Pinnacle Mines Ltd.
Receiver: Roxanne/Joe	Page: 4 of 5

Count	Sample Label		Container Type	Sample Type (C, R, P, Sl, S)	Wet /Dry	Top Size	Weight (kg)
61	2006-136	3074	Bag	C	Wet	5"	1.29
62	2006-136	3075	Bag	C	Wet	5"	1.37
63	2006-136	3076	Bag	C	Wet	5"	1.22
64	2006-136	3077	Bag	C	Wet	5"	1.13
65	2006-136	3078	Bag	C	Wet	6"	1.30
66	2006-136	3079	Bag	C	Wet	5"	1.18
67	2006-136	3080	Bag	C	Wet	4"	1.27
68	2006-136	3081	Bag	C	Wet	5"	1.39
69	2006-136	3082	Bag	C	Wet	5"	1.32
70	2006-136	3083	Bag	C	Wet	5"	1.24
71	2006-136	3084	Bag	C	Wet	6"	1.21
72	2006-146	9013	Bag	C	Dry	5"	1.89
73	2006-146	9014	Bag	C	Dry	7"	1.75
74	2006-146	9015	Bag	C	Dry	7"	1.77
75	2006-146	9016	Bag	C	Dry	6"	1.60
76	2006-146	9017	Bag	C	Dry	6"	1.18
77	2006-146	9018	Bag	C	Dry	7"	1.04
78	2006-146	9019	Bag	C	Dry	7"	1.51
79	2006-146	9020	Bag	C	Dry	6"	1.56
80	2006-146	9021	Bag	C	Dry	7"	1.49
							27.7

Note :

Core, Rock, Pulp, Slurry, Solution

SAMPLE RECEIVING LOG SHEET

Receiving Date: 21-Jul-08	**Project No:** 0805107
Carrier: Canadian Freightways	**Client:** Pinnacle Mines Ltd.
Receiver: Roxanne/Joe	**Page:** 5 of 5

Count	Sample Label		Container Type	Sample Type (C, R, P, Sl, S)	Wet /Dry	Top Size	Weight (kg)
81	2006-146	9022	Bag	C	Dry	7"	1.45
82	2006-146	9023	Bag	C	Dry	5"	1.42
83	2006-146	9024	Bag	C	Dry	6"	1.49
84	2006-146	9025	Bag	C	Dry	7"	1.59
85	2006-146	9026	Bag	C	Dry	6"	1.38
86	2006-146	9027	Bag	C	Dry	7"	1.18
87	2006-146	9028	Bag	C	Dry	6"	1.79
88	2006-146	9029	Bag	C	Dry	7"	1.58
89	2006-146	9030	Bag	C	Dry	7"	1.50
90	SC212	16972	Bag	C	Wet	6"	1.71
91	SC212	16973	Bag	C	Wet	6"	1.50
92	SC212	16974	Bag	C	Wet	6"	1.62
93	SC212	16975	Bag	C	Wet	6"	1.38
94	SC212	16976	Bag	C	Wet	6"	1.54
95	SC212	16977	Bag	C	Wet	6"	1.98
96							
97							
98							
99							
100							

Note : **23.1**

Core, Rock, Pulp, Slurry, Solution

Silver Coin Composite Blends

Composite 1 to 8 sample origin

Comp ID	Drill Hole # and Depth
Comp 08-1	DH 106 (106-126)
Comp 08-2	DH 109 (137-159)
Comp 08-3	DH 115 (110-132)
Comp 08-4	DH 121 (4.3-29)
Comp 08-5	DH 136 (71-88)
Comp 08-6	DH 136 (88-102)
Comp 08-7	DH 146 (116-143)
Comp 08-8	DH 212 (51-61)

Blend used for Master Composite MC1*

Comp ID	Wt kg	Distrubtion %
Comp 08-1	1.4	4.3
Comp 08-2	1.3	4.0
Comp 08-3	9.0	27.8
Comp 08-4	3.3	10.2
Comp 08-5	0.74	2.3
Comp 08-6	0.3	0.9
Comp 08-7	13	40.2
Comp 08-8	3.3	10.2
Total	32.3	100.0

*based on available weight of sample remaining
targeting ~2 g/t Au in feed grade

APPENDIX 2
HEAD CHARACTERIZATION

INTERNATIONAL PLASMA LABS LTD
ISO 9001:2000 CERTIFIED COMPANY

CERTIFICATE OF ANALYSIS
iPL 08H3745

200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5
Phone (604) 272-7818
Fax (604) 272-0851
Website www.ipl.ca

Process Research Associates Ltd
Project : 0805107
Shipper : Boja Grcic
Shipment: PO#: 10526
Comment:

95 Samples Print: Aug 27, 2008 In: Aug 12, 2008 [374513:31:24:80082708:001]

CODE	AMOUNT	TYPE	PREPARATION DESCRIPTION	PULP	REJECT
B31100	95	Pulp	Pulp received as it is, no sample prep.	12M/Dis	00M/Dis
B84100	5	Repeat	Repeat sample - no Charge	12M/Dis	00M/Dis
B82101	1	Blk iPL	Blank iPL - no charge.	00M/Dis	00M/Dis
B90026	1	Std iPL	Std iPL (Au Certified) - no charge		

NS=No Sample Rep=Replicate M=Month Dis=Discard

Document Distribution
1 Process Research Associates Ltd
11620 Horseshoe Way
Richmond
BC V7A 4V5

Att: Boja Grcic
Ph:604/272-8110
Em:bojagrcic@pralab.com

Analytical Summary
Analysis: Au(FA/AAS) S(T) / ICP(MuAc-Metal)30 in ppm

##	Code	Method	Units	Description	Element	Limit Low	Limit High
01	0368	FA/AAS	g/mt	Au (FA/AAS 30g) g/mt	Gold	0.01	5000.00
02	0135	Leco	%	S(tot) Assay by LECO in %	Sulfur (LECO)	0.01	20.00
03	0751	ICPM	ppm	Al ICP(Multi-Acid)	Aluminum	100	50000
04	0752	ICPM	ppm	Sb ICP(Multi-Acid) Depressed	Antimony	5	2000
05	0753	ICPM	ppm	As ICP(Multi-Acid) Depressed	Arsenic	5	10000
06	0754	ICPM	ppm	Ba ICP(Multi-Acid)	Barium	2	10000
07	0755	ICPM	ppm	Bi ICP(Multi-Acid)	Bismuth	2	2000
08	0757	ICPM	ppm	Cd ICP(Multi-Acid)	Cadmium	0.2	2000.0
09	0758	ICPM	ppm	Ca ICP(Multi-Acid)	Calcium	100	100000
10	0759	ICPM	ppm	Cr ICP(Multi-Acid)	Chromium	1	10000
11	0760	ICPM	ppm	Co ICP(Multi-Acid)	Cobalt	1	10000
12	0761	ICPM	ppm	Cu ICP(Multi-Acid)	Copper	1	20000
13	0762	ICPM	ppm	Fe ICP(Multi-Acid)	Iron	100	50000
14	0763	ICPM	ppm	La ICP(Multi-Acid)	Lanthanum	2	10000
15	0764	ICPM	ppm	Pb ICP(Multi-Acid) Depressed	Lead	2	10000
16	0765	ICPM	ppm	Mg ICP(Multi-Acid)	Magnesium	100	100000
17	0766	ICPM	ppm	Mn ICP(Multi-Acid)	Manganese	1	10000
18	0782	ICPM	ppm	Hg ICP(Multi-Acid)	Mercury	3	10000
19	0767	ICPM	ppm	Mo ICP(Multi-Acid)	Molydenum	1	1000
20	0768	ICPM	ppm	Ni ICP(Multi-Acid)	Nickel	1	10000
21	0769	ICPM	ppm	P ICP(Multi-Acid)	Phosphorus	100	50000
22	0770	ICPM	ppm	K ICP(Multi-Acid)	Potassium	100	100000
23	0786	ICPM	ppm	Sc ICP(Multi-Acid)	Scandium	1	10000
24	0771	ICPM	ppm	Ag ICP(Multi-Acid)	Silver	0.5	500.0
25	0772	ICPM	ppm	Na ICP(Multi-Acid)	Sodium	100	100000
26	0773	ICPM	ppm	Sr ICP(Multi-Acid)	Strontium	1	10000
27	0797	ICPM	ppm	Tl ICP(Multi-Acid)	Thallium	2	1000
28	0776	ICPM	ppm	Ti ICP(Multi-Acid)	Titanium	100	100000
29	0777	ICPM	ppm	W ICP(Multi-Acid)	Tungsten	5	1000
30	0779	ICPM	ppm	V ICP(Multi-Acid)	Vanadium	1	10000
31	0780	ICPM	ppm	Zn ICP(Multi-Acid)	Zinc	1	10000
32	0781	ICPM	ppm	Zr ICP(Multi-Acid)	Zirconium	1	10000

* Our liability is limited solely to the analytical cost of these analyses.
ID=C032725

BC Certified Assayer: David Chiu

Signature:

INTERNATIONAL PLASMA LABS LTD.
ISO 9001:2000 CERTIFIED COMPANY

CERTIFICATE OF ANALYSIS
iPL 08H3745

200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5
Phone (604) 272-7818
Fax (604) 272-0851
Website www.ipl.ca

Client : Process Research Associates Ltd
Project: 0805107 Ship#

95 Samples
95=Pulp 5=Repeat 1=Blk iPL 1=Std iPL [374513312480082708001] Print: Aug 27, 2008 In: Aug 12, 2008

Sample Name	Type	Au g/mt	S(tot) %	Al ppm	Sb ppm	As ppm	Ba ppm	Bi ppm	Cd ppm	Ca ppm	Cr ppm	Co ppm	Cu ppm	Fe ppm	La ppm	Pb ppm	Mg ppm	Mn ppm
2006-106- 1086	Pulp	0.51	1.43	8.03%	<5	<5	1073	<2	<0.2	37931	19	18	50	5.30%	12	137	11993	2608
2006-106- 1087	Pulp	0.73	1.02	8.68%	<5	<5	2230	<2	<0.2	26616	22	19	45	5.55%	13	104	12945	2191
2006-106- 1088	Pulp	0.15	2.16	7.80%	<5	<5	785	<2	<0.2	39398	27	18	43	5.62%	13	134	11295	2270
2006-106- 1089	Pulp	0.00	3.42	7.70%	<5	<5	258	<2	<0.2	32106	39	20	69	6.23%	10	532	13090	1767
2006-106- 1090	Pulp	0.04	1.43	8.31%	<5	<5	2593	<2	<0.2	37739	28	22	31	6.01%	12	233	17461	3351
2006-106- 1091	Pulp	0.49	2.61	48162	<5	<5	346	<2	7.9	30189	75	10	64	44055	7	1611	8323	3133
2006-106- 1092	Pulp	1.64	4.14	33999	<5	<5	281	<2	50.1	39026	86	10	214	5.39%	7	4976	6857	3443
2006-109- 1839	Pulp	0.58	2.94	5.89%	<5	<5	505	<2	<0.2	24571	55	13	55	5.19%	9	314	8667	3304
2006-109- 1840	Pulp	0.82	2.15	6.16%	<5	<5	620	<2	9.4	31981	48	14	269	47023	9	997	11323	4404
2006-109- 1841	Pulp	1.71	1.42	6.39%	<5	<5	638	<2	<0.2	56549	34	12	129	43089	12	228	10651	4993
2006-109- 1842	Pulp	0.35	2.15	8.20%	<5	<5	740	<2	<0.2	32437	27	16	78	5.70%	13	193	12930	3982
2006-109- 1843	Pulp	1.29	2.48	5.82%	<5	<5	315	<2	<0.2	26498	52	12	33	48541	8	399	9951	3620
2006-109- 1844	Pulp	1.40	2.49	7.14%	<5	<5	610	<2	<0.2	24217	45	16	67	5.49%	11	492	12369	4131
2006-109- 1845	Pulp	1.30	2.00	5.37%	<5	<5	319	<2	8.5	47556	54	11	97	42909	9	1105	10175	5089
2006-109- 1846	Pulp	2.56	2.37	5.60%	<5	<5	288	<2	<0.2	34311	68	11	111	5.96%	9	502	15068	6336
2006-109- 1847	Pulp	2.27	4.99	30072	12	<5	250	4	117.9	30190	90	10	1008	5.59%	4	6805	6274	3583
2006-109- 1848	Pulp	1.83	12.60	25384	17	<5	141	<2	182.0	39311	77	10	1024	12%	2	2.50%	4711	3091
2006-109- 1849	Pulp	0.60	4.19	45285	<5	<5	527	<2	38.3	12882	79	11	182	5.71%	3	3432	7430	1761
2006-109- 1850	Pulp	2.63	4.50	45841	<5	<5	568	<2	29.9	9165	73	12	85	5.73%	2	2332	6747	2099
2006-109- 1851	Pulp	0.67	4.19	5.38%	<5	<5	399	<2	1.5	8261	63	14	33	6.83%	3	1562	13726	3354
2006-109- 1852	Pulp	1.30	3.10	7.38%	<5	<5	381	<2	6.0	23612	54	15	74	6.92%	8	816	14821	4691
2006-115- 3932	Pulp	2.02	6.62	11199	<5	<5	214	12	205.8	47083	104	13	1126	5.83%	5	543	1248	2767
2006-115- 3933	Pulp	1.93	6.52	12236	<5	<5	285	10	199.4	56251	85	11	452	5.66%	6	2310	2341	4024
2006-115- 3934	Pulp	1.44	4.99	29436	<5	<5	220	<2	20.4	43150	83	12	87	6.82%	7	891	8507	4648
2006-115- 3935	Pulp	4.37	3.50	5.01%	<5	<5	370	<2	31.7	16698	83	14	102	5.97%	4	400	10835	2943
2006-115- 3936	Pulp	4.47	4.22	12241	<5	<5	494	<2	63.6	16596	163	9	747	43243	<2	822	2216	1690
2006-115- 3937	Pulp	1.59	5.16	14649	<5	<5	288	4	79.7	50617	107	6	658	5.16%	3	5725	2322	3554
2006-115- 3938	Pulp	1.97	2.05	25244	<5	<5	791	2	18.6	51476	103	6	245	27820	6	2211	3110	3241
2006-115- 3939	Pulp	0.82	2.23	40653	<5	<5	274	3	13.6	29429	85	9	120	36115	5	700	5217	2140
2006-115- 3940	Pulp	1.44	1.85	7.14%	<5	<5	616	2	<0.2	28734	56	15	72	46321	10	253	9759	3358
2006-115- 3941	Pulp	4.53	9.05	30381	<5	<5	173	<2	40.0	26651	79	12	204	9.73%	3	819	6592	2909
2006-115- 3942	Pulp	1.82	6.90	42382	<5	<5	225	2	<0.2	36140	72	14	220	8.53%	6	885	8763	3443
2006-115- 3943	Pulp	0.31	2.00	6.13%	<5	<5	316	5	<0.2	43342	59	10	137	46041	9	340	11646	4108
2006-115- 3944	Pulp	0.27	2.58	30241	<5	<5	231	<2	15.2	98207	70	7	231	37340	6	1821	5631	5676
2006-115- 3945	Pulp	1.71	2.51	6.80%	<5	<5	321	<2	58.3	28249	48	15	170	5.51%	12	441	13699	4273
2006-115- 3946	Pulp	0.78	1.97	6.59%	<5	<5	290	<2	<0.2	25284	55	13	63	5.13%	10	133	13091	3199
2006-121- 5389	Pulp	0.98	3.23	5.87%	<5	<5	257	<2	91.6	56206	43	15	124	5.54%	12	323	11856	3491
2006-121- 5390	Pulp	1.32	2.71	5.87%	<5	<5	370	<2	103.3	27938	49	15	223	5.36%	9	378	13212	3404
2006-121- 5391	Pulp	0.44	8.19	22646	<5	<5	128	3	260.9	19309	101	9	509	7.55%	<2	3065	4964	2026
Minimum Detection		0.01	0.01	100	5	5	2	2	0.2	100	1	1	1	100	2	2	100	1
Maximum Detection		5000.00	20.00	50000	2000	10000	10000	2000	2000.0	100000	10000	10000	20000	50000	10000	10000	100000	10000
Method		FA/AAS	Leco	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

---=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

INTERNATIONAL PLASMA LABS LTD.
ISO 9001:2000 CERTIFIED COMPANY

CERTIFICATE OF ANALYSIS
iPL 08H3745

200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5
Phone (604) 272-7818
Fax (604) 272-0851
Website www.ipl.ca

Client : Process Research Associates Ltd
Project: 0805107 Ship#

95 Samples
95=Pulp 5=Repeat 1=Blk iPL 1=Std iPL [374513312480082708001]

Print: Aug 27, 2008
In: Aug 12, 2008

Sample Name	Hg ppm	Mo ppm	Ni ppm	P ppm	K ppm	Sc ppm	Ag ppm	Na ppm	Sr ppm	Tl ppm	Ti ppm	W ppm	V ppm	Zn ppm	Zr ppm
2006-106- 1086	<3	16	<1	1116	43963	16	2.9	2125	110	<2	3124	17	157	152	34
2006-106- 1087	<3	18	<1	1152	44931	18	2.2	3675	73	<2	3425	14	172	255	62
2006-106- 1088	<3	18	<1	1022	42161	16	3.1	2512	97	<2	2617	17	157	204	37
2006-106- 1089	<3	17	<1	1034	43693	17	2.7	383	91	<2	3034	10	157	602	23
2006-106- 1090	<3	17	<1	1046	37975	17	1.1	7553	118	<2	3099	9	148	824	40
2006-106- 1091	<3	13	<1	598	38351	7	3.6	4094	163	<2	987	<5	65	2284	13
2006-106- 1092	<3	10	<1	425	27475	5	19.9	1922	198	<2	666	<5	57	7870	9
2006-109- 1839	<3	16	<1	782	46643	10	3.4	4142	126	<2	1734	13	98	658	22
2006-109- 1840	<3	12	<1	828	42197	12	6.4	4323	101	<2	2001	<5	102	3084	27
2006-109- 1841	<3	12	<1	886	43758	13	3.6	4094	159	<2	2264	11	116	1284	26
2006-109- 1842	<3	17	<1	1178	53729	16	1.3	5305	141	<2	3011	17	156	956	38
2006-109- 1843	<3	14	<1	797	45593	10	1.6	5413	135	<2	1855	<5	96	823	17
2006-109- 1844	<3	15	<1	962	46695	13	0.7	8394	128	<2	2291	6	124	1388	20
2006-109- 1845	<3	10	<1	732	37989	10	3.6	3727	114	<2	1529	<5	93	2893	18
2006-109- 1846	<3	15	<1	728	32287	10	1.8	6737	123	<2	1573	6	87	1446	17
2006-109- 1847	<3	8	<1	344	21637	5	19.8	3625	98	<2	723	<5	44	1.64%	9
2006-109- 1848	<3	12	<1	285	20304	4	30.5	1707	110	<2	555	<5	36	3.10%	8
2006-109- 1849	<3	13	<1	591	31965	9	9.7	330	50	<2	1456	<5	71	7273	22
2006-109- 1850	<3	15	<1	586	41364	7	5.9	677	70	<2	1283	<5	65	5999	17
2006-109- 1851	<3	17	<1	698	42091	8	2.4	760	68	<2	1348	5	84	2694	16
2006-109- 1852	<3	14	<1	882	48892	13	2.6	437	111	<2	2285	<5	126	3569	19
2006-115- 3932	<3	6	<1	<100	9131	2	15.5	205	110	<2	193	<5	23	2.92%	4
2006-115- 3933	<3	6	<1	118	7971	2	9.6	576	206	<2	199	<5	33	2.85%	5
2006-115- 3934	<3	12	<1	398	20953	6	3.0	260	101	<2	742	<5	56	4518	11
2006-115- 3935	<3	13	<1	708	36220	10	8.6	415	70	<2	1322	<5	95	5885	17
2006-115- 3936	<3	5	<1	224	6362	2	10.9	2874	71	<2	239	<5	15	8962	5
2006-115- 3937	<3	6	<1	210	11537	3	24.9	260	193	<2	343	<5	23	1.30%	6
2006-115- 3938	<3	6	<1	307	21072	4	7.8	1026	124	<2	713	<5	33	3469	12
2006-115- 3939	<3	9	<1	483	32781	6	2.7	1572	75	<2	1054	<5	56	3141	12
2006-115- 3940	<3	14	<1	938	42260	13	1.6	2002	85	<2	2305	10	121	503	25
2006-115- 3941	<3	12	<1	386	25138	4	6.1	423	83	<2	638	<5	47	7968	10
2006-115- 3942	<3	12	<1	539	30799	8	5.9	849	91	<2	1021	<5	74	2158	14
2006-115- 3943	<3	11	<1	788	45332	11	1.2	4662	152	<2	1432	7	104	591	14
2006-115- 3944	<3	8	<1	363	25907	5	8.3	903	131	<2	737	<5	45	3199	9
2006-115- 3945	<3	14	<1	935	49917	11	3.7	10604	181	<2	2015	<5	119	8067	20
2006-115- 3946	<3	11	<1	920	49391	11	<0.5	9131	152	<2	1634	12	107	520	17
2006-121- 5389	<3	13	<1	829	28775	12	2.3	2754	141	<2	2403	<5	118	1.37%	28
2006-121- 5390	<3	11	<1	860	27849	12	4.7	4519	124	<2	2575	<5	119	1.53%	30
2006-121- 5391	<3	10	<1	283	10731	4	9.3	1054	89	41	637	<5	40	3.64%	11
Minimum Detection	3	1	1	100	100	1	0.5	100	1	2	100	5	1	1	1
Maximum Detection	10000	1000	10000	50000	100000	10000	500.0	100000	10000	1000	100000	1000	10000	10000	10000
Method	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

iPL
INTERNATIONAL PLASMA LABS LTD.
ISO 9001:2000 CERTIFIED COMPANY

CERTIFICATE OF ANALYSIS
iPL 08H3745

200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5
Phone (604) 272-7818
Fax (604) 272-0851
Website www.ipl.ca

Client : Process Research Associates Ltd
Project: 0805107 Ship#

95 Samples
95=Pulp 5=Repeat 1=Blk iPL 1=Std iPL [374513312480082708001]

Print: Aug 27, 2008
In: Aug 12, 2008

Sample Name	Type	Au g/mt	S(tot) %	Al ppm	Sb ppm	As ppm	Ba ppm	Bi ppm	Cd ppm	Ca ppm	Cr ppm	Co ppm	Cu ppm	Fe ppm	La ppm	Pb ppm	Mg ppm	Mn ppm
2006-121- 5392	Pulp	1.50	3.88	49854	<5	<5	180	<2	4.5	18366	80	13	40	6.08%	5	287	9641	2083
2006-121- 5393	Pulp	1.06	6.33	41015	10	<5	102	<2	2.3	43998	79	13	31	8.63%	5	351	8393	2802
2006-121- 5394	Pulp	0.53	1.72	6.15%	<5	<5	707	<2	<0.2	89657	47	18	124	5.26%	14	802	11899	5222
2006-121- 5395	Pulp	1.10	7.89	33609	<5	<5	94	6	195.8	18955	88	15	240	9.70%	3	498	9373	2684
2006-121- 5396	Pulp	4.41	7.93	30319	<5	<5	153	5	35.0	23163	92	14	97	10%	5	226	8781	2800
2006-121- 5397	Pulp	4.71	3.80	10983	<5	<5	294	3	4.6	14%	71	8	346	47918	16	1360	3799	6650
2006-121- 5398	Pulp	1.26	5.67	24319	6	<5	137	<2	32.0	65518	85	13	424	7.37%	6	4583	7267	3884
2006-121- 5399	Pulp	0.95	8.35	26239	<5	<5	212	11	69.1	97977	77	16	210	10%	6	4183	7293	4531
2006-121- 5400	Pulp	1.67	5.37	31529	<5	<5	83	6	7.0	67970	64	15	129	10%	6	4639	8146	3955
2006-121- 5401	Pulp	0.67	4.02	35933	<5	<5	92	<2	101.1	57950	76	15	427	7.82%	6	1.17%	7767	2907
2006-121- 5402	Pulp	1.86	14.50	24006	<5	<5	83	12	121.9	41489	91	21	567	15%	3	1.14%	7785	2636
2006-121- 5403	Pulp	5.84	10.50	29972	<5	<5	112	18	13.9	66638	80	22	136	12%	6	6219	8016	3334
2006-121- 5404	Pulp	3.25	15.70	32006	<5	<5	218	23	<0.2	22839	97	30	121	17%	3	1907	7189	2188
2006-136- 3066	Pulp	6.02	6.74	46195	<5	<5	243	<2	91.1	4251	87	11	204	6.98%	<2	8712	4956	677
2006-136- 3067	Pulp	7.81	6.92	29637	<5	<5	176	<2	196.8	3062	122	8	96	6.16%	<2	8705	3514	665
2006-136- 3068	Pulp	3.38	12.20	27227	<5	<5	219	<2	479.8	32660	90	8	524	9.28%	3	3.11%	4081	2968
2006-136- 3069	Pulp	0.52	3.09	5.44%	<5	<5	592	<2	12.4	3572	89	11	60	44962	4	830	7002	1239
2006-136- 3070	Pulp	0.14	1.88	5.91%	<5	<5	463	<2	<0.2	5085	88	11	21	39453	6	278	8131	1379
2006-136- 3071	Pulp	7.86	9.54	46296	<5	<5	316	<2	181.4	4415	53	13	505	9.13%	3	9565	6766	1118
2006-136- 3072	Pulp	0.75	2.60	6.99%	<5	<5	369	<2	<0.2	13897	64	12	21	46512	6	277	10564	1994
2006-136- 3073	Pulp	0.20	2.01	6.69%	<5	<5	487	<2	<0.2	16424	59	12	24	43286	7	270	11306	2084
2006-136- 3074	Pulp	1.08	2.77	48408	<5	<5	518	<2	0.7	12705	86	11	38	41730	5	872	7513	1419
2006-136- 3075	Pulp	10.60	6.58	40708	<5	<5	296	<2	64.5	17427	88	11	79	7.39%	3	4994	6696	1700
2006-136- 3076	Pulp	0.39	1.40	45851	<5	<5	520	<2	<0.2	18967	93	7	13	28925	7	74	6807	1593
2006-136- 3077	Pulp	0.55	1.37	5.62%	<5	<5	475	<2	<0.2	47656	67	8	20	34593	9	198	8609	3508
2006-136- 3078	Pulp	0.12	1.55	7.06%	<5	<5	506	<2	<0.2	25481	65	12	26	41694	11	79	10681	2579
2006-136- 3079	Pulp	0.19	1.25	38210	<5	<5	208	<2	<0.2	30871	108	7	61	24314	6	85	5235	2215
2006-136- 3080	Pulp	0.80	1.99	5.79%	<5	<5	497	<2	<0.2	24497	82	11	121	36266	8	580	8129	1703
2006-136- 3081	Pulp	0.26	2.40	6.82%	<5	<5	614	<2	<0.2	7194	76	12	26	42094	7	104	7944	949
2006-135- 3082	Pulp	0.11	1.90	5.51%	<5	<5	401	3	<0.2	5067	97	10	24	30834	6	87	5712	620
2006-135- 3083	Pulp	1.33	3.98	44638	<5	49	449	<2	<0.2	6314	115	9	45	5.21%	2	560	5115	831
2006-136- 3084	Pulp	0.19	2.48	5.33%	<5	<5	338	<2	<0.2	6712	71	10	17	35623	5	78	6564	1183
2006-146- 9013	Pulp	1.55	3.24	5.64%	<5	<5	308	<2	45.8	19572	36	12	31	49664	5	4888	9795	2669
2006-146- 9014	Pulp	1.92	2.27	6.58%	<5	<5	350	<2	0.5	24611	29	14	26	46621	10	583	10590	3105
2006-146- 9015	Pulp	0.49	2.30	6.77%	<5	<5	323	<2	10.9	26958	32	14	44	45776	9	1096	11385	2621
2006-146- 9016	Pulp	0.72	1.73	5.17%	<5	<5	422	<2	22.2	72869	30	12	35	42094	10	847	18605	3131
2006-146- 9017	Pulp	0.64	2.64	6.97%	<5	<5	247	<2	9.6	19876	40	15	43	43391	12	1099	11510	1252
2006-146- 9018	Pulp	1.62	1.54	21555	<5	<5	694	4	<0.2	75746	80	6	237	23756	8	202	3802	3388
2006-146- 9019	Pulp	2.23	1.05	38837	<5	<5	380	3	<0.2	56334	57	8	190	26788	7	189	6914	3205
Minimum Detection		0.01	0.01	100	5	5	2	2	0.2	100	1	1	1	100	2	2	100	1
Maximum Detection		5000.00	20.00	50000	2000	10000	10000	2000	2000.0	100000	10000	10000	20000	50000	10000	10000	100000	10000
Method		FA/AAS	Leco	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

INTERNATIONAL PLASMA LABS LTD.
ISO 9001:2000 CERTIFIED COMPANY

200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5
Phone (604) 272-7818
Fax (604) 272-0851
Website www.ipl.ca

CERTIFICATE OF ANALYSIS
iPL 08H3745

Client : Process Research Associates Ltd
Project: 0805107 Ship#

95 Samples
95=Pulp 5=Repeat 1=Blk iPL 1=Std iPL [374513312480082708001] Print: Aug 27, 2008 In: Aug 12, 2008

Sample Name	Hg ppm	Mo ppm	Ni ppm	P ppm	K ppm	Sc ppm	Ag ppm	Na ppm	Sr ppm	Tl ppm	Ti ppm	W ppm	V ppm	Zn ppm	Zr ppm
2006-121- 5392	<3	22	<1	742	26975	10	4.6	1124	59	<2	1759	<5	95	2208	20
2006-121- 5393	<3	18	<1	612	22114	9	5.9	1040	110	<2	1511	11	78	2554	19
2006-121- 5394	<3	18	<1	811	31580	15	3.6	1912	217	<2	3109	7	136	1402	26
2006-121- 5395	<3	16	<1	487	16125	6	8.8	737	66	<2	1064	<5	66	2.78%	16
2006-121- 5396	<3	15	<1	439	16502	5	5.9	546	65	<2	963	<5	55	7136	15
2006-121- 5397	<3	10	<1	121	5794	2	13.6	212	233	<2	310	<5	47	2087	5
2006-121- 5398	<3	13	<1	306	13144	5	12.9	417	138	<2	911	<5	45	5049	11
2006-121- 5399	<3	15	<1	282	15743	5	11.5	795	191	<2	892	<5	35	9885	13
2006-121- 5400	<3	17	<1	352	15954	6	9.5	795	150	<2	1148	<5	54	3460	16
2006-121- 5401	<3	12	<1	414	19404	7	20.6	690	126	<2	1069	<5	73	1.32%	15
2006-121- 5402	<3	20	<1	246	10551	4	17.7	286	95	<2	759	<5	43	2.15%	14
2006-121- 5403	<3	18	<1	336	15254	5	13.2	786	162	<2	935	<5	51	4947	15
2006-121- 5404	<3	24	<1	295	16647	5	10.0	1031	78	<2	957	<5	61	3900	15
2006-136- 3066	<3	12	<1	517	38974	6	52.4	3794	55	<2	914	<5	59	1.58%	18
2006-136- 3067	<3	8	<1	315	25231	4	30.8	2207	38	<2	594	<5	30	3.01%	11
2006-136- 3068	<3	10	<1	257	21246	4	52.5	2414	96	<2	516	80	37	7.58%	11
2006-136- 3069	<3	13	<1	618	50542	8	15.8	3048	71	<2	1248	<5	73	3792	24
2006-136- 3070	<3	15	<1	705	52135	9	9.8	5774	96	<2	1369	9	76	349	27
2006-136- 3071	<3	16	<1	549	41641	7	51.4	2370	66	<2	997	<5	65	3.01%	19
2006-136- 3072	<3	15	<1	772	70498	10	15.5	3573	142	<2	1894	7	90	318	29
2006-136- 3073	<3	14	<1	740	66369	9	1.9	2615	132	<2	1896	8	85	545	28
2006-136- 3074	<3	15	<1	535	45693	7	5.2	1384	105	<2	1172	<5	61	1267	21
2006-136- 3075	<3	13	<1	422	35706	6	21.6	2115	91	<2	902	<5	55	1.13%	16
2006-136- 3076	<3	12	<1	488	40074	6	2.4	3411	113	<2	1021	7	55	112	24
2006-136- 3077	<3	12	<1	622	37225	8	2.1	7263	199	<2	1184	6	77	251	25
2006-136- 3078	<3	13	<1	828	49932	11	1.5	10414	185	<2	1606	<5	102	313	27
2006-136- 3079	<3	13	<1	425	31799	6	3.0	3789	116	<2	944	5	48	129	18
2006-136- 3080	<3	13	<1	643	51006	9	6.5	4276	136	<2	1587	6	80	1488	28
2006-136- 3081	<3	14	<1	734	58325	10	11.7	3860	96	<2	1631	9	81	167	32
2006-136- 3082	<3	13	<1	660	43733	9	3.1	6587	79	<2	1384	10	75	146	29
2006-136- 3083	<3	12	<1	484	40129	6	26.9	1853	68	<2	1020	6	47	876	18
2006-136- 3084	<3	13	<1	618	45410	8	2.4	4802	95	<2	1212	10	72	146	21
2006-146- 9013	<3	13	<1	753	37907	10	4.9	5703	108	<2	1436	<5	95	8677	22
2006-146- 9014	<3	16	<1	896	42437	13	2.7	5337	109	<2	2039	<5	120	1956	30
2006-146- 9015	<3	14	<1	886	43115	13	2.2	5104	130	<2	2081	<5	124	3581	31
2006-146- 9016	<3	15	<1	669	30141	9	2.1	4612	329	<2	1409	<5	85	4722	26
2006-146- 9017	<3	15	<1	953	42410	13	3.0	5285	100	<2	2320	7	125	3134	36
2006-146- 9018	<3	8	<1	276	17866	4	4.1	390	146	<2	717	<5	39	481	11
2006-146- 9019	<3	10	<1	559	30477	7	5.5	777	140	<2	1048	9	68	333	14
Minimum Detection	3	1	1	100	100	1	0.5	100	1	2	100	5	1	1	1
Maximum Detection	10000	1000	10000	50000	100000	10000	500.0	100000	10000	1000	100000	1000	10000	10000	10000
Method	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

INTERNATIONAL PLASMA LABS LTD
ISO 9001:2000 CERTIFIED COMPANY

CERTIFICATE OF ANALYSIS
iPL 08H3745

200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5
Phone (604) 272-7818
Fax (604) 272-0851
Website www.ipl.ca

Client : Process Research Associates Ltd
Project: 0805107 Ship#

95 Samples
95=Pulp 5=Repeat 1=Blk iPL 1=Std iPL [374513312480082708001]

Print: Aug 27, 2008
In: Aug 12, 2008

Sample Name	Type	Au g/mt	S(tot) %	Al ppm	Sb ppm	As ppm	Ba ppm	Bi ppm	Cd ppm	Ca ppm	Cr ppm	Co ppm	Cu ppm	Fe ppm	La ppm	Pb ppm	Mg ppm	Mn ppm
2006-146- 9020	Pulp	13.19	0.97	25433	<5	<5	351	<2	1.8	62967	73	5	476	18932	6	421	3885	3655
2006-146- 9021	Pulp	6.17	1.65	29425	<5	<5	287	<2	<0.2	55887	65	7	410	25981	6	646	4205	4058
2006-146- 9022	Pulp	2.32	1.21	22052	<5	<5	216	<2	1.0	42518	82	4	52	19013	4	147	3087	2748
2006-146- 9023	Pulp	1.71	2.20	5.79%	<5	<5	314	3	<0.2	36020	45	11	41	40942	8	325	6617	2996
2006-146- 9024	Pulp	2.20	1.66	32364	<5	<5	229	<2	<0.2	38139	81	8	94	27288	8	215	4447	2331
2006-146- 9025	Pulp	0.58	3.56	6.52%	31	<5	281	<2	<0.2	19326	37	15	50	5.71%	10	284	8856	1342
2006-146- 9026	Pulp	0.72	2.57	8.03%	<5	<5	446	<2	<0.2	13562	24	18	24	5.91%	11	134	11807	2144
2006-146- 9027	Pulp	0.35	2.02	7.53%	<5	<5	581	<2	1.1	10364	35	16	41	47527	11	652	9089	1650
2006-146- 9028	Pulp	0.95	2.45	6.66%	<5	<5	403	<2	20.5	16426	37	16	68	5.23%	8	1211	10421	2528
2006-146- 9029	Pulp	0.46	2.47	7.08%	<5	<5	329	<2	13.4	33066	34	18	89	5.92%	12	1199	12272	3962
2006-146- 9030	Pulp	2.06	5.74	5.77%	14	<5	405	<2	39.4	17479	33	17	272	8.21%	9	2500	11371	2661
SC212- 16972	Pulp	1.85	4.33	28982	<5	<5	195	<2	<0.2	35124	89	12	15	5.58%	3	263	6072	2718
SC212- 16973	Pulp	2.86	3.53	33437	<5	<5	175	<2	<0.2	31254	98	11	16	49849	4	264	6148	2777
SC212- 16974	Pulp	3.42	4.37	32238	<5	<5	150	<2	<0.2	62995	72	13	21	6.51%	6	500	9442	4809
SC212- 16975	Pulp	1.73	5.21	42802	<5	<5	392	3	<0.2	26431	55	17	19	8.09%	4	177	11777	3531
SC212- 16976	Pulp	0.82	4.29	35816	<5	<5	274	3	<0.2	44634	81	13	12	6.43%	3	68	9194	3740
SC212- 16977	Pulp	1.82	5.29	18655	<5	<5	122	<2	<0.2	63763	85	13	14	6.91%	4	94	6606	3519
RE 2006-105- 1086	Repeat	0.49	1.36	7.97%	<5	<5	1012	<2	<0.2	37914	19	18	47	5.22%	11	142	11870	2547
RE 2006-109- 1851	Repeat	0.68	4.16	5.32%	<5	<5	394	<2	1.8	8156	63	14	32	6.65%	3	1577	13340	3285
RE 2006-121- 5392	Repeat	1.61	3.99	49960	<5	<5	173	<2	4.1	18029	73	13	38	6.02%	5	284	9587	2059
RE 2006-136- 3072	Repeat	0.71	2.58	7.00%	<5	<5	330	<2	<0.2	13810	63	13	21	46290	6	261	10430	1981
RE 2006-146- 9020	Repeat	13.04	0.91	25104	<5	<5	349	<2	1.7	62214	74	5	463	18614	7	430	3778	3613
Blank iPL	Blk iPL	<0.01	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
OX167	Std iPL	1.82	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
OX167 REF	Std iPL	1.82	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Minimum Detection		0.01	0.01	100	5	5	2	2	0.2	100	1	1	1	100	2	2	100	1
Maximum Detection		5000.00	20.00	50000	2000	10000	10000	2000	2000.0	100000	10000	10000	20000	50000	10000	10000	100000	10000
Method		FA/AAS	Leco	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPH	ICPM	ICPM

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

INTERNATIONAL PLASMA LABS LTD
ISO 9001:2000 CERTIFIED COMPANY

CERTIFICATE OF ANALYSIS
iPL 08H3745

200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5
Phone (604) 272-7818
Fax (604) 272-0851
Website www.ipl.ca

Client : Process Research Associates Ltd
Project: 0805107

95 Samples
Ship# 95=Pulp 5=Repeat 1=Blk iPL 1=Std iPL [374513312480082708001] Print: Aug 27, 2008 In: Aug 12, 2008

Sample Name	Hg ppm	Mo ppm	Ni ppm	P ppm	K ppm	Sc ppm	Ag ppm	Na ppm	Sr ppm	Tl ppm	Ti ppm	W ppm	V ppm	Zn ppm	Zr ppm
2006-146- 9020	<3	7	<1	302	24993	4	14.1	474	139	<2	553	<5	28	778	7
2006-146- 9021	<3	8	<1	387	24302	5	15.9	1105	133	<2	822	<5	41	657	10
2006-146- 9022	<3	6	<1	239	22315	3	6.5	1077	134	<2	530	<5	30	651	9
2006-146- 9023	<3	14	<1	716	60043	8	2.6	1989	181	<2	1914	8	94	881	21
2006-146- 9024	<3	9	<1	462	27156	6	5.3	600	144	<2	785	6	48	288	10
2006-146- 9025	<3	16	<1	847	44010	13	6.1	647	77	<2	1512	6	104	756	27
2006-146- 9026	<3	23	<1	1087	43069	15	4.7	9355	94	<2	2728	10	137	402	46
2006-146- 9027	<3	17	<1	1044	45269	14	7.8	9976	100	<2	2296	<5	143	2153	39
2006-146- 9028	<3	17	<1	844	39645	12	14.3	10417	113	<2	2098	<5	110	4931	28
2006-146- 9029	<3	70	<1	887	34816	12	14.9	11066	165	<2	2545	<5	114	4415	29
2006-146- 9030	<3	28	<1	775	33702	9	15.3	9833	116	<2	1814	<5	92	8318	26
SC212- 16972	<3	12	<1	387	20318	5	5.2	1786	111	<2	877	13	46	303	8
SC212- 16973	<3	12	<1	397	19646	6	4.4	4497	121	<2	1060	<5	51	946	8
SC212- 16974	<3	13	<1	511	15790	7	22.8	5582	225	<2	1100	11	59	433	10
SC212- 16975	<3	17	<1	599	26457	10	6.2	4006	124	<2	1477	10	89	240	13
SC212- 16976	<3	14	<1	518	25204	8	9.9	2726	166	<2	1425	12	70	98	10
SC212- 16977	<3	12	<1	275	10956	5	15.8	532	144	<2	804	8	39	146	7
RE 2006-106- 1086	<3	16	<1	1125	43691	16	3.0	2074	109	<2	3044	16	151	159	35
RE 2006-109- 1851	<3	16	<1	689	41743	7	2.6	753	66	<2	1374	5	81	2633	19
RE 2006-121- 5392	<3	20	<1	724	26790	9	4.1	1062	55	<2	1808	<5	91	2197	21
RE 2006-136- 3072	<3	16	<1	762	70054	10	16.4	3617	146	<2	1845	6	89	301	29
RE 2006-146- 9020	<3	7	<1	298	24027	3	14.7	483	136	<2	572	<5	28	770	7
Blank iPL	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
OXI67	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
OXI67 REF	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Minimum Detection	3	1	1	100	100	1	0.5	100	1	2	100	5	1	1	1
Maximum Detection	10000	1000	10000	50000	100000	10000	500.0	100000	10000	1000	100000	1000	10000	10000	10000
Method	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM	ICPM

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

ASSAY REPORT

Client: Pinnacle Mines Ltd.-Silver Coin Project
Sample: Head Composite Samples as per ID

Date: 20-Aug-08
Project: 0805107

Items	Unit	Sample ID									Detection Limits		Analytical
		08-1	08-2	08-3	08-4	08-5	08-6	08-7	08-8	RE-08-1	Min.	Max.	Method
Au	g/mt	0.41	1.35	1.45	1.69	2.88	0.38	1.85	1.96	0.42	0.01	5000	FA/AAS
Zn	%	0.11	0.57	0.73	1.11	1.40	0.04	0.25	0.03	0.11	0.01	20	MuAICP
As	ppm	62	52	53	111	132	110	78	64	57	5	10000	Assay
Sb	%	<0.001	<0.001	0.00	<0.001	0.00	<0.001	<0.001	<0.001	<0.001	0.001	100	AsyMuA
Hg	ppb	<5	620	2089	1556	1474	<5	100	<5	<5	5	10000	CVA
Se	%	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	100	AqR/AA
S(tot)	%	2.20	4.11	4.62	8.44	5.46	2.30	2.45	5.27	2.21	0.01	20	Leco
S(-2)	%	2.20	4.11	4.62	8.44	5.44	2.29	2.45	5.27	2.21	0.01	100	AsyWet
C Tot	%	1.16	0.84	1.22	1.56	0.40	0.72	1.02	1.45	1.16	0.01	100	Leco
C(Org)	%	0.34	0.18	0.35	0.46	0.20	0.28	0.20	0.41	0.34	0.01	100	Leco
Al	ppm	72341	54467	34346	34390	49301	55264	53408	30688	72717	100	5000	ICPM
Sb	ppm	<5	<5	<5	<5	<5	<5	<5	<5	<5	5	2000	ICPM
As	ppm	<5	<5	<5	<5	<5	<5	<5	<5	<5	5	10000	ICPM
Ba	ppm	711	1045	556	301	816	734	784	550	727	2	10000	ICPM
Bi	ppm	<2	<2	2	14	<2	<2	<2	<2	<2	2	2000	ICPM
Cd	ppm	<0.2	27.3	47.1	65.7	82.7	<0.2	7.4	<0.2	<0.2	0.2	2000	ICPM
Ca	ppm	33622	28249	38555	49571	9800	19412	33794	44440	33921	100	100000	ICPM
Cr	ppm	47	66	77	81	107	116	73	106	47	1	10000	ICPM
Co	ppm	16	13	11	17	11	10	12	13	17	1	10000	ICPM
Cu	ppm	58	237	297	236	145	41	126	17	57	1	20000	ICPM
Fe	ppm	54029	59643	52989	91984	57191	36400	42239	64893	54724	100	50000	ICPM
La	ppm	16	9	8	9	6	11	13	7	15	2	10000	ICPM
Pb	ppm	576	3191	1137	3146	5323	160	737	187	568	2	10000	ICPM
Mg	ppm	13713	11337	6732	8599	7558	7592	9422	8437	13697	100	100000	ICPM
Mn	ppm	2500	3849	3209	3103	1321	1516	2636	3250	2530	1	10000	ICPM
Hg	ppm	<3	<3	<3	<3	<3	<3	<3	<3	<3	3	10000	ICPM
Mo	ppm	16	15	9	13	10	12	14	12	16	1	1000	ICPM
Ni	ppm	<1	<1	<1	<1	<1	<1	<1	<1	<1	1	10000	ICPM
P	ppm	1001	749	486	462	603	680	758	474	1017	100	50000	ICPM
K	ppm	48709	48718	33020	20956	56703	56555	44213	23651	48952	100	100000	ICPM
Sc	ppm	19	12	8	9	9	11	12	9	19	1	10000	ICPM
Ag	ppm	2.3	7.6	8.3	8.9	22.7	5.5	3.5	5.2	2.3	0.5	500	ICPM
Na	ppm	3798	3637	2463	1177	3219	6128	5265	3152	3781	100	100000	ICPM
Sr	ppm	117	113	129	127	102	139	144	158	115	1	10000	ICPM
Tl	ppm	<2	<2	<2	63	<2	<2	<2	<2	<2	2	1000	ICPM
Ti	ppm	2705	1508	913	1109	1061	1301	1569	1229	2709	100	100000	ICPM
W	ppm	<5	<5	<5	<5	<5	<5	<5	7	<5	5	1000	ICPM
V	ppm	142	92	63	70	68	80	94	61	142	1	10000	ICPM
Zn	ppm	952	5380	7033	9536	13824	396	2347	302	964	1	10000	ICPM
Zr	ppm	64	39	22	29	30	34	44	21	64	1	10000	ICPM

HEAD ASSAY REPORT - WHOLE ROCK

Client: Pinnacle Mines Ltd. - Silver Coin Project
Sample: as specified

Date: 20-Aug-08
Project: 0805107

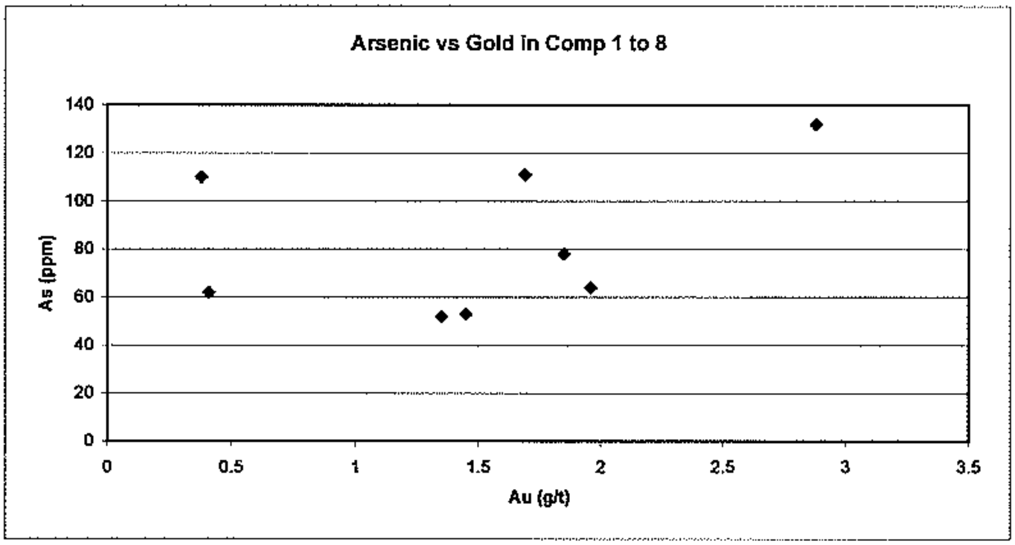
Compound	Unit	Sample ID								
		08-1	08-2	08-3	08-4	08-5	08-6	08-7	08-8	RE 08-1
Al2O3	%	13.57	10.56	6.78	6.72	9.64	10.63	10.41	5.84	13.58
BaO	%	0.32	0.45	0.32	0.16	0.43	0.4	0.37	0.25	0.32
CaO	%	4.43	3.80	5.21	6.69	1.53	2.56	4.47	5.80	4.40
Fe2O3	%	7.31	8.17	7.40	12.65	7.92	4.97	5.79	8.69	7.46
K2O	%	4.63	4.51	3.41	2.27	5.39	5.43	4.25	2.30	4.63
MgO	%	1.91	1.60	1.02	1.30	1.14	1.13	1.41	1.24	1.96
MnO	%	0.31	0.49	0.41	0.39	0.19	0.23	0.33	0.41	0.32
Na2O	%	0.49	0.49	0.34	0.15	0.45	0.83	0.72	0.44	0.51
P2O5	%	0.25	0.15	0.06	0.08	0.15	0.14	0.18	0.09	0.25
SiO2	%	60.4	64.88	70.36	61.19	67.12	69.62	66.7	67.56	60.17
TiO2	%	0.71	0.49	0.31	0.34	0.41	0.45	0.50	0.40	0.71
LOI	%	6.12	4.56	4.6	7.12	5.91	4.05	5.5	7.44	6.17
Total	%	100.45	100.16	100.24	99.06	100.3	100.44	100.64	100.45	100.47

SPECIFIC GRAVITY DETERMINATION

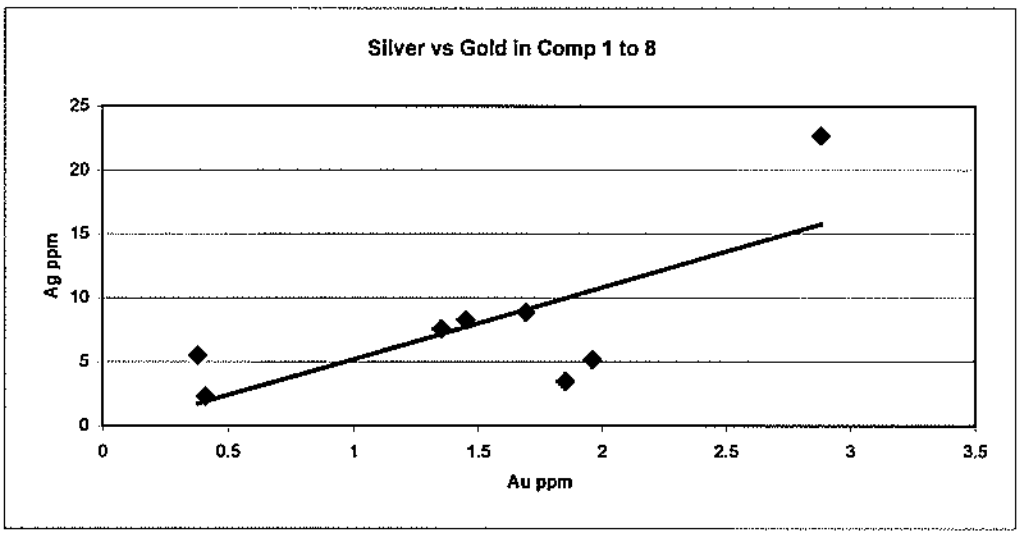
Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: SG 1 to 8
Sample: as specified composites

Date: 18-Aug-08
Project: 0805107

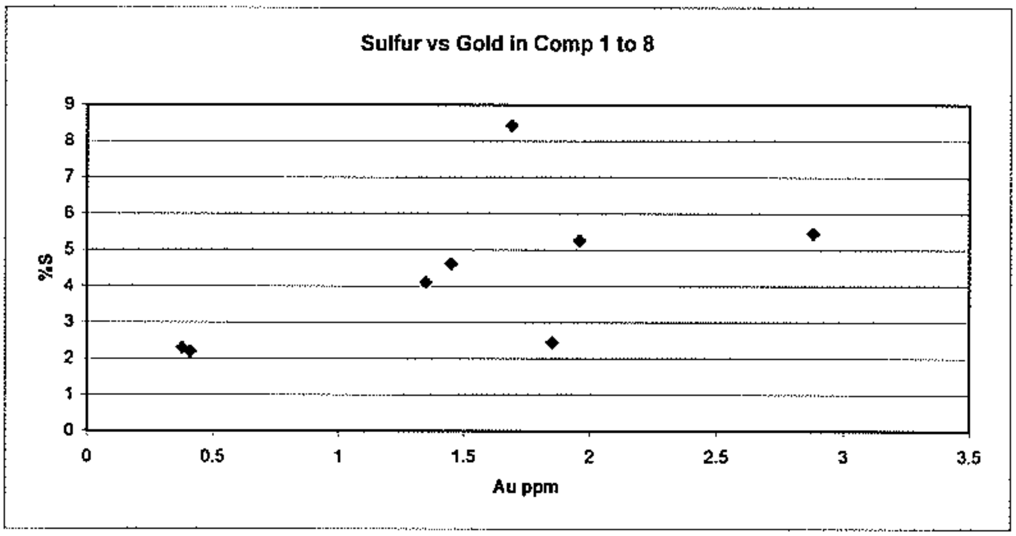
Sample ID	Solids Specific Gravity, g/cm3
Comp. 08-1	2.82
Comp. 08-2	2.80
Comp. 08-3	2.79
Comp. 08-4	2.93
Comp. 08-5	2.81
Comp. 08-6	2.71
Comp. 08-7	2.76
Comp. 08-8	2.80

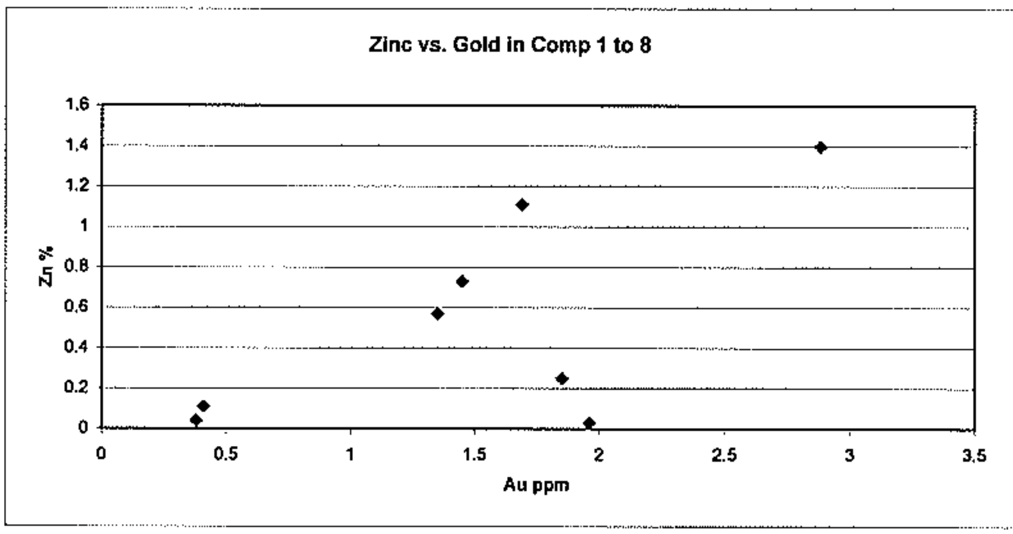


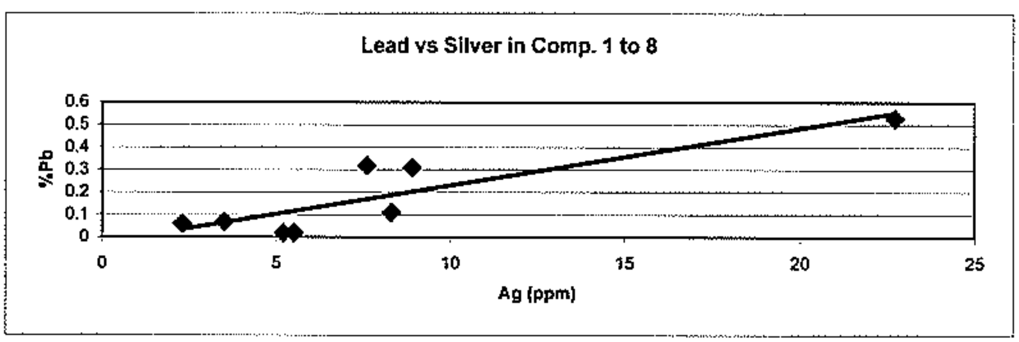
Arsenic vs Gold in Comp 1 to 8
140
120
100
80
60
40
20
0
As (ppm)
0
0.5
1
1.5
2
2.5
3
3.5
Au (g/t)



Silver vs Gold in Comp 1 to 8
25
20
15
10
5
0
Ag ppm
0
0.5
1
1.5
2
2.5
3
3.5
Au ppm


Sulfur vs Gold in Comp 1 to 8
%S
Au ppm
0
1
2
3
4
5
6
7
8
9
0
0.5
1
1.5
2
2.5
3
3.5


Zinc vs. Gold in Comp 1 to 8
Zn %
Au ppm
0
0.2
0.4
0.6
0.8
1
1.2
1.4
1.6
0
0.5
1
1.5
2
2.5
3
3.5


Lead vs Silver in Comp. 1 to 8
%Pb
Ag (ppm)
0
0.1
0.2
0.3
0.4
0.5
0.6
0
5
10
15
20
25

ASSAY REPORT

Client: Pinnacle Mines Ltd.-Silver Coin Project
Sample: MC-1 Head Composite Sample

Date: 30-Oct-08
Project: 0805107

Items	Unit	Sample ID		Detection Limits		Analytical
		MC-1	RE: MC-1	Min.	Max.	Method
Au	g/mt	1.89	1.85	0.01	5000	FA/AAS
Ag	ppm	7.90	6.00	0.5	1000	MuAICP
S(tot)	%	4.55	4.51	0.01	20	Leco
S(-2)	%	4.55	4.51	0.01	100	AsyWet
C(Org)	%	0.36	0.32	0.01	100	Leco
C Tot	%	1.35	1.33	0.01	100	Leco
Zn	%	0.57	0.56	0.01	20	MuAICP
As	ppm	20.1	22.0	0.03	10000	AsyICP
Hg	ppb	1424	1346	5	10000	CVA
Se	%	<0.01	<0.01	0.01	100	AqR/AA
Al	ppm	37659	38156	100	5000	ICPM
Sb	ppm	<5	<5	5	2000	ICPM
As	ppm	<5	<5	5	10000	ICPM
Ba	ppm	265	274	2	10000	ICPM
Bi	ppm	<2	<2	2	2000	ICPM
Cd	ppm	24.4	25.9	0.2	2000	ICPM
Ca	ppm	41126	41565	100	100000	ICPM
Cr	ppm	79	76	1	10000	ICPM
Co	ppm	11	12	1	10000	ICPM
Cu	ppm	206	213	1	20000	ICPM
Fe	ppm	55984	56106	100	50000	ICPM
La	ppm	6	7	2	10000	ICPM
Pb	ppm	728	754	2	10000	ICPM
Mg	ppm	7298	7386	100	100000	ICPM
Mn	ppm	3310	3347	1	10000	ICPM
Hg	ppm	<3	<3	3	10000	ICPM
Mo	ppm	9	8	1	1000	ICPM
Ni	ppm	<1	<1	1	10000	ICPM
P	ppm	464	476	100	50000	ICPM
K	ppm	25030	25677	100	100000	ICPM
Sc	ppm	6	7	1	10000	ICPM
Ag	ppm	7.1	7.9	0.5	500	ICPM
Na	ppm	2372	2378	100	100000	ICPM
Sr	ppm	121	122	1	10000	ICPM
Tl	ppm	<2	<2	2	1000	ICPM
Ti	ppm	1180	1247	100	100000	ICPM
W	ppm	<5	<5	5	1000	ICPM
V	ppm	58	58	1	10000	ICPM
Zn	ppm	5231	5308	1	10000	ICPM
Zr	ppm	14	21	1	10000	ICPM

HEAD ASSAY REPORT - WHOLE ROCK

Client: Silver Coin Project
Sample: MC1 Head Analyses

Date: 20-Aug-08
Project: 0805107

Compound	Unit	Sample ID		Detection Range
		MC-1	Re-MC-1	%
Al_2O_3	%	7.59	7.55	0.01 to 100
BaO	%	0.29	0.35	0.01 to 100
CaO	%	5.72	5.72	0.01 to 100
Fe_2O_3	%	8.06	8.12	0.01 to 100
K_2O	%	3.1	3.2	0.01 to 100
MgO	%	1.17	1.18	0.01 to 100
MnO	%	0.42	0.42	0.01 to 100
Na_2O	%	0.36	0.37	0.01 to 100
P_2O_5	%	<0.01	<0.01	0.01 to 100
SiO_2	%	66.7	65.47	0.01 to 100
TiO_2	%	0.38	0.38	0.01 to 100
LOI	%	5.84	6.25	0.01 to 100
Total	%	99.63	99.01	

METALLIC ASSAY REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Sample: Head Composites as per ID

Date: 20-Aug-08
Project: 0805107

Sample ID	Screen	Weight	Au		Ag	
	Tyler Mesh	(g)	(g/t)	(mg)	(g/t)	(mg)
Composite 08-1	+150	11.9	0.66	0.008	2.0	0.02
	-150	292.7	0.37	0.108	2.5	0.73
	Total	304.6	**0.38**	0.116	**2.48**	0.76
	Tyler Mesh	(g)	(g/t)	(mg)		
Composite 08-2	+150	30.5	1.88	0.057	3.5	0.11
	-150	261.6	1.67	0.437	9.4	2.46
	Total	292.1	**1.69**	0.494	**8.8**	2.57
	Tyler Mesh	(g)	(g/t)	(mg)		
Composite 08-3	+150	29.9	2.21	0.066	4.0	0.12
	-150	263.2	2.12	0.558	6.9	1.82
	Total	293.1	**2.13**	0.624	**6.6**	1.94
	Tyler Mesh	(g)	(g/t)	(mg)		
Composite 08-4	+150	25.0	2.23	0.056	5.4	0.14
	-150	268.5	1.88	0.505	9.9	2.66
	Total	293.5	**1.91**	0.561	**9.5**	2.79
	Tyler Mesh	(g)	(g/t)	(mg)		
Composite 08-5	+150	32.9	5.60	0.184	21.8	0.72
	-150	259.2	4.24	1.099	24.8	6.43
	Total	292.1	**4.39**	1.283	**24.5**	7.15
	Tyler Mesh	(g)	(g/t)	(mg)		
Composite 08-6	+150	21.4	0.94	0.020	6.9	0.15
	-150	281.0	0.43	0.121	5.5	1.55
	Total	302.4	**0.47**	0.141	**5.6**	1.69
	Tyler Mesh	(g)	(g/t)	(mg)		
Composite 08-7	+150	21.5	2.68	0.057	2.5	0.05
	-150	281.2	2.02	0.568	5.4	1.52
	Total	302.7	**2.07**	0.626	**5.2**	1.57
	Tyler Mesh	(g)	(g/t)	(mg)		
Composite 08-8	+150	32.9	2.37	0.078	3.5	0.12
	-150	253.4	2.35	0.595	5.5	1.39
	Total	286.2	**2.35**	0.673	**5.3**	1.51

BOND MILL GRINDABILITY TEST REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: BI-1
Sample: Comp 08-1+2

Date: 25-Aug-08
Project: 0805107

TEST CONDITIONS

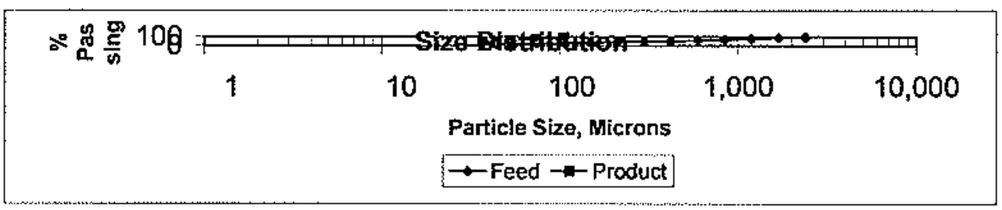
Cycle	Oversize Wt. grams	Product Wt. grams	Feed Undersize grams	Net Product grams	Product per Rev. grams/rev.	Required Rev. rev.
1	949.76	531.40	471.16	60.24	0.6024	100
2	925.78	555.38	169.04	386.34	0.9158	422
3	1036.52	444.64	176.67	267.97	0.9955	269
4	1048.01	433.15	141.44	291.71	1.0307	283
5	1044.98	436.18	137.79	298.39	1.0776	277
6	1043.61	437.55	138.75	298.80	1.1320	264

SIZE ANALYSIS

Sieve Size		% Passing	
Tyler mesh	µm	Feed	Product
8	2,380	94.3	
10	1,680	87.5	
14	1,190	76.3	
20	841	69.6	
28	595	61.3	
35	420	53.6	
48	297	47.5	
65	210	41.8	
100	149	36.6	
150	105	31.8	100.0
200	74	27.8	72.2
270	53	24.3	58.5
325	44	22.9	53.8
400	37	21.5	49.2

TEST RESULTS

Material Charge Wt.-700 mL(g) = 1,481.2
Test Screen (µm) = 105
Undersize in Feed (%)= 31.81
Circulating Load (%) = 239
Gbp (ave.) = 1.10
Product P_{80} (µm) = 83
Feed F_{80} (µm) = 1,342
W (kWh/ton) = 17.0
W (kWh/tonne) = 18.7



BOND MILL GRINDABILITY TEST REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: BI-2
Sample: Comp 08-5+6

Date: 25-Aug-08
Project: 0805107

TEST CONDITIONS

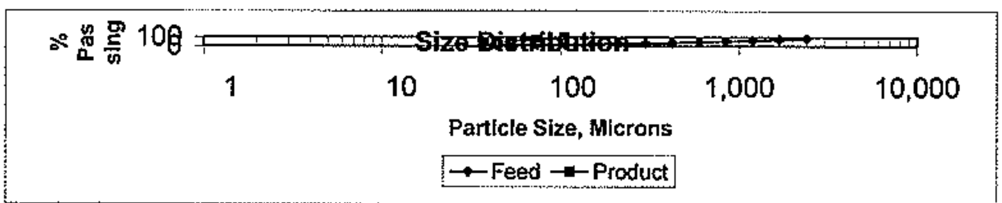
Cycle	Oversize Wt. grams	Product Wt. grams	Feed Undersize grams	Net Product grams	Product per Rev. grams/rev.	Required Rev. rev.
1	1060.85	392.77	308.02	84.75	0.8475	100
2	1000.34	453.28	83.23	370.05	0.9444	392
3	1033.36	420.26	96.05	324.21	0.9590	338
4	1009.08	444.54	89.05	355.49	1.0448	340
5	1033.95	419.67	94.20	325.47	1.0590	307
6	1033.15	420.47	88.93	331.54	1.0757	308

SIZE ANALYSIS

Sieve Size		% Passing	
Tyler mesh	μm	Feed	Product
8	2,380	86.1	
10	1,680	75.2	
14	1,190	63.1	
20	841	55.8	
28	595	47.9	
35	420	40.7	
48	297	35.0	
65	210	30.0	
100	149	25.3	
150	105	21.2	100.0
200	74	17.8	70.0
270	53	15.3	54.5
325	44	14.2	49.4
400	37	13.2	44.0

TEST RESULTS

Material Charge Wt.-700 mL(g) = 1,453.6
Test Screen (μm) = 105
Undersize in Feed (%)= 21.19
Circulating Load (%) = 246
Gbp (ave.) = 1.07
Product P_{80} (μm) = 84
Feed F_{80} (μm) = 1,970
W (kWh/ton) = 16.7
W (kWh/tonne) = 18.4



iPL
International Plasma Labs Ltd.
ISO 9001:2000 Certified Company

#200 - 11620 Horseshoe Way
Richmond, B.C.
Canada V7A 4V5

Phone: 604/879-7878 604/272-7818
Fax: 604/879-7898 604/272-0851
Website: www.ipl.ca
Email: info@ipl.ca

Certificate#: 08H3771
Client: Process Research Associates Ltd
Project: 0805107
Shipment#:
PO#: 10534
No. of Samples: 2
Analysis #1: Au(FA/AAS) S(T)
Analysis #2:
Analysis #3:
Comment #1:
Comment #2:
Date In: Aug 12, 2008
Date Out: Aug 13, 2008

Sample Name (blended 1:1 weight ratio)	SampleType	Au g/mt	S(tot) %
Comp 08-1+2	Pulp	0.72	3.33
Comp 08-5+6	Pulp	2.19	4.05
RE Comp 08-1+2	Repeat	0.72	3.38
Blank iPL	Blk iPL	<0.01	--
OXI67	Std iPL	1.81	--
OXI67 REF	Std iPL	1.82	--
Minimum detection		0.01	0.01
Maximum detection		5000	20
Method		FA/AAS	Leco

APPENDIX 3
OPEN CYCLE FLOAT

KINETIC FLOTATION SUMMARY TABLE

Client: Pinnacle Mines Ltd. - Silver Coin Project
Tests: F1 to F10
Sample: as specified two composites

Date: 2-Sep-08
Project: 0805107

Objective: Grind vs Recovery for low S and high S composites with SIPX & A208 collector at natural pH
Note: F1, F6 use Au specific collector (A6697) in 1st Stage: F5, F10 use PAX and CuSO4 not SIPX

CALCULATED HEAD AND BULK RECOVERY

Test No	Grind	Grind	Calc. Head Grades		Final Tail		1st Stage Rec.(%)			Bulk Recovery (%)			Comments
	1 kg min	P80 (u)	Au, g/t	%S	Au, g/t	%S	Au	S	Mass	Au	S	Mass	
Low S (blended 1:1 Comp 08-1 & 08-2) Au = 0.72 g/t. S= 3.36%													
F1	11.3	51	1.26	2.89	0.04	0.05	90.4	41.2	11.5	98.0	98.9	37.8	A6697 1st stg reduced S float; note high calc Au head
F2	8.5	71	0.91	2.71	0.08	0.09	90.4	93.4	12.3	93.9	97.7	30.1	SIPX and A208
F3	6.0	113	0.75	2.88	0.07	0.08	88.7	93.2	11.0	93.8	98.1	28.1	SIPX and A208, mid range grind
F4	4.5	170	0.85	3.16	0.07	0.11	88.7	92.4	10.6	93.8	97.5	24.6	SIPX and A208, coarsest grind
F5	6.0	115	0.91	2.64	0.10	0.11	88.2	92.5	12.9	91.9	92.5	26.7	similar to F3 grind but use PAX and CuSO4
High S (blended 1:1 Comp 08-5 & 08-6) Au = 2.19 g/t. S= 4.05%													
F6	15.0	53	1.97	3.90	0.02	0.04	83.6	16.6	6.9	99.3	99.3	32.2	A6697 1st stg reduced S float; more time may help Au
F7	12.0	70	2.23	4.37	0.04	0.04	95.1	96.4	13.6	98.7	99.4	29.6	SIPX and A208
F8	8.5	113	1.91	3.56	0.04	0.05	95.5	94.3	12.2	98.5	99.0	26.3	SIPX and A208, mid range grind
F9	6.0	183	1.96	3.82	0.08	0.10	90.9	89.0	10.4	96.9	98.0	23.2	SIPX and A208, coarsest grind
F10	check	~115?	2.02	3.98	0.04	0.03	90.7	95.4	13.5	98.8	99.5	28.0	similar to F8 grind but use PAX and CuSO4

Observations

low S is softer than high S composite based on lab mill grind time despite similar Bond Ball Mill WI
F1 vs others - A6697 1st stg reduced S float = higher Au in 1st stage; also 0.47% org. C (26% rec.) +2.1% Zn into 1st stage; note high calc Au head
F6 vs others - A6697 1st stg reduced S float = higher Au in 1st stage longer time may help rec; also 0.70% org. C (30% rec.) +5.6% Zn into 1st stage.
F2 to F4 - Finer grinding increases mass pull with similar low tail losses, and similar kinetics see graph on F4 PRA spreadsheet
F7 to F9 - Finer grinding increases mass pull with lower tail losses <~150 u, & faster kinetics see graph on F9 PRA spreadsheet
F5 vs F3 - Use of aggressive float reagents did not benefit recovery
F10 vs F8 - use of aggressive float reagents showed similar results

F. Wright Consulting Inc.

FLOTATION PROCEDURES

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F1
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: Scoping Kinetic Flotation (F1 to F4) vs primary grind size, F1 uses a prefloat for free gold

	Reagents					Time, minutes				
Stage	A6697	SIPX	A208	$CuSO_4$	MIBC	Grind	Cond.	Float	pH	Comments
Grind (1 kg),Mill#3						11'30"			8.9	target P80 = 53u
ROUGHER FLOTATION										
Condition	30						1			
Rougher Float 1					17			5	8.9	gold prefloat
										brassy color~2min then grey
Condition		15	15				1			
Rougher Float 2					7			5	8.9	
Condition		10	10				1			
Rougher Float 3					10			5	8.9	
Condition		10	10				1			
Rougher Float 4					10			6	8.9	white froth, no mineralization visible
TOTAL REAGENTS	30	35	35	0	43					

FLOTATION TEST METALLURGICAL BALANCE

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F1
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: Scoping Kinetic Flotation (F1 to F4) vs primary grind size, F1 uses a prefloat for free gold

Flotation Balance

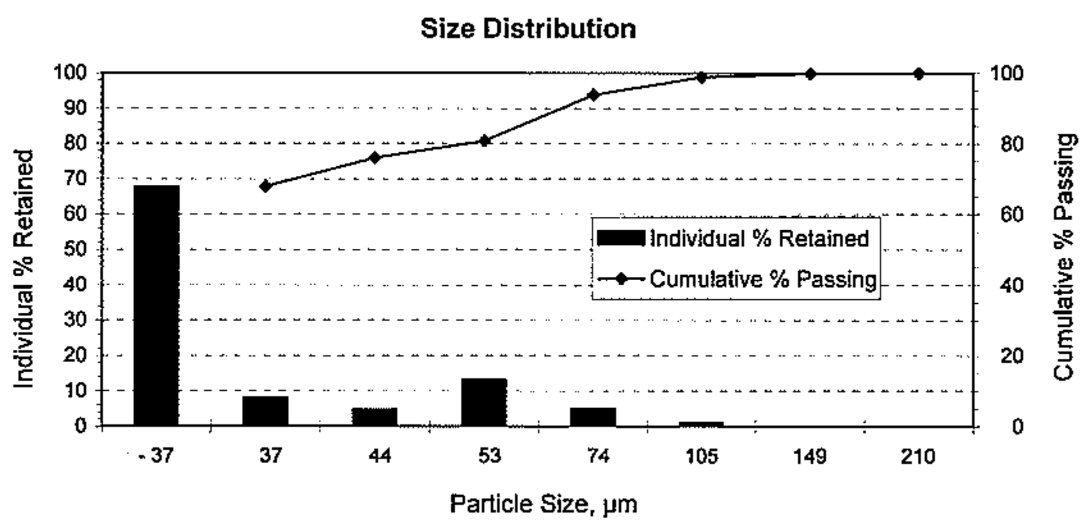
Product	Weight		Assay			Distribution		
			Au	S(tot)	C(org)	Au	S(tot)	C(org)
	(g)	(%)	(g/t)	(%)	(%)	(%)	(%)	(%)
Rougher Concentrate 1	105.4	11.5	9.87	10.30	0.47	90.4	41.2	26.3
Rougher Concentrate 2	109.7	12.0	0.71	13.30	0.43	6.8	55.4	25.1
Rougher Concentrate 1+2	**215.1**	**23.6**	**5.20**	**11.83**	**0.45**	**97.1**	**96.6**	**51.4**
Rougher Concentrate 3	64.3	7.0	0.10	0.71	0.18	0.6	1.7	6.2
Rougher Concentrate 1+2+3	**279.4**	**30.6**	**4.02**	**9.27**	0.39	**97.7**	**98.3**	**57.5**
Rougher Concentrate 4	65.9	7.2	0.06	0.25	0.18	0.3	0.6	6.3
Total Flotation Concentrate	**345.3**	**37.8**	**3.27**	**7.55**	0.35	**98.0**	**98.9**	**63.8**
Flotation Tailings	567.0	62.2	0.04	0.05	0.12	2.0	1.1	36.2
Calculated Feed	**912.3**	**100.0**	**1.26**	**2.89**	**0.21**	**100.0**	**100.0**	**100.0**
Measured Feed			0.72	3.36				

SIZE ANALYSIS REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Date: 25-Aug-08
Test: F1
Project: 0805107
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)
Grind: 1 kg sample ground for 11'30" minutes at 65% solids in stainless steel mill #3

Sieve Size		Individual	Cumulative
Tyler Mesh	**Micrometers**	**% Retained**	**% Passing**
65	210	0.0	100.0
100	149	0.1	99.9
150	105	1.0	98.9
200	74	5.0	93.9
270	53	13.2	80.8
325	44	4.7	76.0
400	37	8.1	67.9
Undersize	- 37	67.9	-
TOTAL:		100.0	

80 % Passing Size (μm) = 51



ASSAY REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Sample: Flotation Products as per ID

Date: 25-Aug-08
Project: 0805107

Items	Unit	Sample ID						Detection Limits		Analytical
		F1 Cut tails	F1 Ro Conc 4	F1 Ro Conc 3	F1 Ro Conc 2	F1 Ro Conc 1	RE:F1 Cut Tails	Min.	Max.	Method
Au	g/mt	0.04	0.06	0.10	0.71	9.87	0.04	0.01	5000	FA/AAS
S(tot)	%	0.05	0.25	0.71	13.30	10.30	0.05	0.01	20	Leco
C(Org)	%	0.11	0.18	0.18	0.43	0.47	0.13	0.01	100	Leco
Al	ppm	58739	69722	72861	54333	61151	59875	100	50000	ICPM
Sb	ppm	<5	<5	<5	<5	21	<5	5	2000	ICPM
As	ppm	<5	<5	<5	<5	<5	<5	5	10000	ICPM
Ba	ppm	3490	3546	3574	1234	1460	3522	2	10000	ICPM
Bi	ppm	<2	<2	<2	9	11	<2	2	2000	ICPM
Cd	ppm	<0.2	<0.2	10	<0.2	127	<0.2	0.2	2000	ICPM
Ca	ppm	29008	33496	32627	24438	28152	29527	100	100000	ICPM
Cr	ppm	270	624	656	436	482	267	1	10000	ICPM
Co	ppm	11	16	19	45	43	11	1	10000	ICPM
Cu	ppm	48	45	68	164	1196	52	1	20000	ICPM
Fe	ppm	30001	36551	41362	145614	98227	30541	100	50000	ICPM
La	ppm	15	17	15	10	10	15	2	10000	ICPM
Pb	ppm	83	226	365	890	15376	87	2	10000	ICPM
Mg	ppm	11315	14029	14559	10424	12283	11344	100	100000	ICPM
Mn	ppm	2963	3341	3372	2539	2938	2959	1	10000	ICPM
Hg	ppm	<3	<3	<3	<3	<3	<3	3	10000	ICPM
Mo	ppm	21	31	34	40	42	20	1	1000	ICPM
Ni	ppm	121	294	304	199	219	121	1	10000	ICPM
P	ppm	903	830	784	590	628	911	100	50000	ICPM
K	ppm	48810	54424	54996	40407	47380	49178	100	100000	ICPM
Sc	ppm	14	18	20	14	16	14	1	10000	ICPM
Ag	ppm	<0.5	<0.5	1	6.5	35.7	<0.5	0.5	500	ICPM
Na	ppm	3843	3168	3091	2304	2456	3836	100	100000	ICPM
Sr	ppm	121	127	125	90	99	120	1	10000	ICPM
Tl	ppm	<2	<2	<2	<2	<2	<2	2	1000	ICPM
Ti	ppm	1941	2259	2488	1984	2115	2007	100	100000	ICPM
W	ppm	7	6	<5	<5	<5	9	5	1000	ICPM
V	ppm	108	144	153	120	134	107	1	10000	ICPM
Zn	ppm	89	874	3235	4376	21164	94	1	10000	ICPM
Zr	ppm	38	48	58	55	58	38	1	10000	ICPM

FLOTATION PROCEDURES

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F2
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: F2 to F4 vary primary grind on low S composite blend

Stage	SIPX	A208	$CuSO_4$	MIBC	Time, minutes Grind	Cond.	Float	pH	Comments
Grind (1 kg),Mill#3					8'30"			8.7	target P80 = 74 u
ROUGHER FLOTATION									
Condition	15	15				1			
Rougher Float 1				17			5	8.6	sph with golden luster of py
									to barren
Condition	10	10				1			
Rougher Float 2				13			5	8.6	grey~2min
Condition	10	10				1			
Rougher Float 3				7			5	8.7	slightly grey
Condition	10	10				1			
Rougher Float 4				7			6	8.7	
TOTAL REAGENTS	45	45	0	43					

FLOTATION TEST METALLURGICAL BALANCE

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F2
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: F2 to F4 vary primary grind on low S composite blend

Flotation Balance

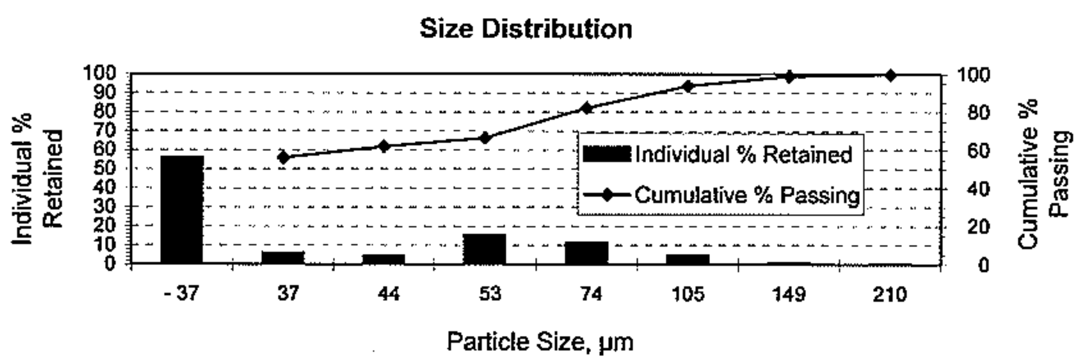
Product	Weight		Assay		Distribution	
			Au	S(tot)	Au	S(tot)
	(g)	(%)	(g/t)	(%)	(%)	(%)
Rougher Concentrate 1	117.2	12.3	6.73	20.60	90.4	93.4
Rougher Concentrate 2	60.1	6.3	0.29	1.14	2.0	2.7
Rougher Concentrate 1+2	**177.3**	**18.6**	**4.55**	**14.00**	**92.4**	**96.0**
Rougher Concentrate 3	56.2	5.9	0.13	0.49	0.8	1.1
Rougher Concentrate 1+2+3	**233.5**	**24.4**	**3.48**	**10.75**	**93.2**	**97.1**
Rougher Concentrate 4	54.1	5.7	0.11	0.28	0.7	0.6
Total Flotation Concentrate	**287.5**	**30.1**	**2.85**	**8.78**	**93.9**	**97.7**
Flotation Tailings	668.0	69.9	0.08	0.09	6.1	2.3
Calculated Feed	**955.5**	**100.0**	**0.91**	**2.71**	**100.0**	**100.0**
Measured Feed			0.72	3.36		

SIZE ANALYSIS REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project **Date:** 25-Aug-08
Test: F2 **Project:** 0805107
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)
Grind: 1 kg sample ground for 8'30" minutes at 65% solids in stainless steel mill #3

Sieve Size		Individual	Cumulative
Tyler Mesh	**Micrometers**	**% Retained**	**% Passing**
65	210	0.1	99.9
100	149	0.9	99.0
150	105	5.1	93.9
200	74	11.5	82.3
270	53	15.7	66.6
325	44	4.5	62.1
400	37	6.1	56.0
Undersize	- 37	56.0	-
TOTAL:		100.0	

80 % Passing Size (µm) = 71



FLOTATION PROCEDURES

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F3
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: F2 to F4 vary primary grind on low S composite blend

					Time, minutes				
Stage	SIPX	A208	$CuSO_4$	MIBC	Grind	Cond.	Float	pH	Comments
Grind (1 kg),Mill#3					6.0			8.7	target P80 = 105 u
ROUGHER FLOTATION									
Condition	15	15				1			
Rougher Float 1				20			5	8.7	grey slightly brassy ~3'
Condition	10	10				1			
Rougher Float 2				20			5	8.6	after 2' almost barren
Condition	10	10				1			
Rougher Float 3				13			5	8.6	slightly greyish ~1'
Condition	10	10				1			
Rougher Float 4				10			6	8.6	
TOTAL REAGENTS	45	45	0	63					

FLOTATION TEST METALLURGICAL BALANCE

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F3
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: F2 to F4 vary primary grind on low S composite blend

Flotation Balance

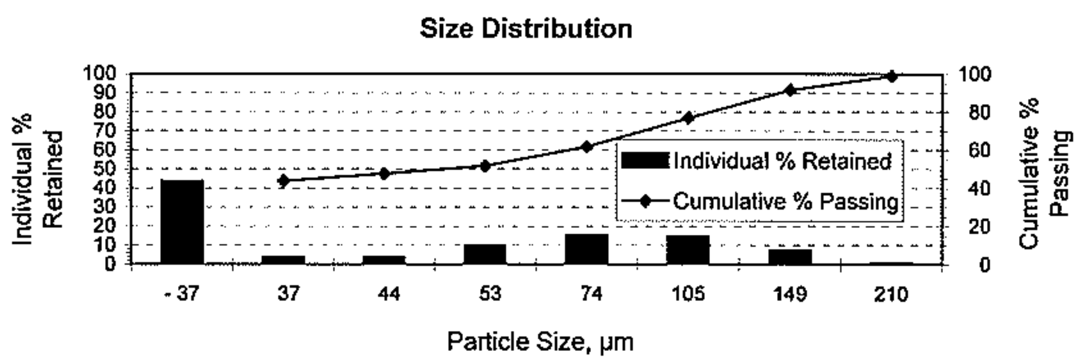
Product	Weight		Assay		Distribution	
			Au	S(tot)	Au	S(tot)
	(g)	(%)	(g/t)	(%)	(%)	(%)
Rougher Concentrate 1	104.4	11.0	6.04	24.40	88.7	93.2
Rougher Concentrate 2	69.6	7.3	0.31	1.19	3.0	3.0
Rougher Concentrate 1+2	**174.0**	**18.3**	**3.75**	**15.11**	**91.7**	**96.2**
Rougher Concentrate 3	48.9	5.1	0.17	0.70	1.2	1.3
Rougher Concentrate 1+2+3	**222.9**	**23.5**	**2.96**	**11.95**	**92.9**	**97.5**
Rougher Concentrate 4	44.0	4.6	0.14	0.39	0.9	0.6
Total Flotation Concentrate	**266.9**	**28.1**	**2.50**	**10.05**	**93.8**	**98.1**
Flotation Tailings	683.1	71.9	0.07	0.08	6.2	1.9
Calculated Feed	**950.0**	**100.0**	**0.75**	**2.88**	**100.0**	**100.0**
Measured Feed			0.72	3.36		

SIZE ANALYSIS REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Date: 25-Aug-08
Test: F3
Project: 0805107
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)
Grind: 1 kg sample ground for 6 minutes at 65% solids in stainless steel mill #3

Sieve Size		Individual	Cumulative
Tyler Mesh	Micrometers	% Retained	% Passing
65	210	0.9	99.1
100	149	7.3	91.7
150	105	14.8	77.0
200	74	15.1	61.8
270	53	10.2	51.7
325	44	4.0	47.7
400	37	3.9	43.8
Undersize	- 37	43.8	-
TOTAL:		100.0	

80 % Passing Size (μm) = 113



FLOTATION PROCEDURES

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F4
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: F2 to F4 vary primary grind on low S composite blend

Stage	SIPX	A208	$CuSO_4$	MIBC	Time, minutes Grind	Cond.	Float	pH	Comments
Grind (1 kg), Mill#3					4.5			8.5	target P80 = 149u
ROUGHER FLOTATION									
Condition	15	15				1			
Rougher Float 1				17			5	8.6	similar to F3
Condition	10	10				1			
Rougher Float 2				13			5	8.6	
Condition	10	10				1			
Rougher Float 3				10			5	8.7	
Condition	10	10				1			
Rougher Float 4				7			6	8.7	
TOTAL REAGENTS	45	45	0	as reqd					

FLOTATION TEST METALLURGICAL BALANCE

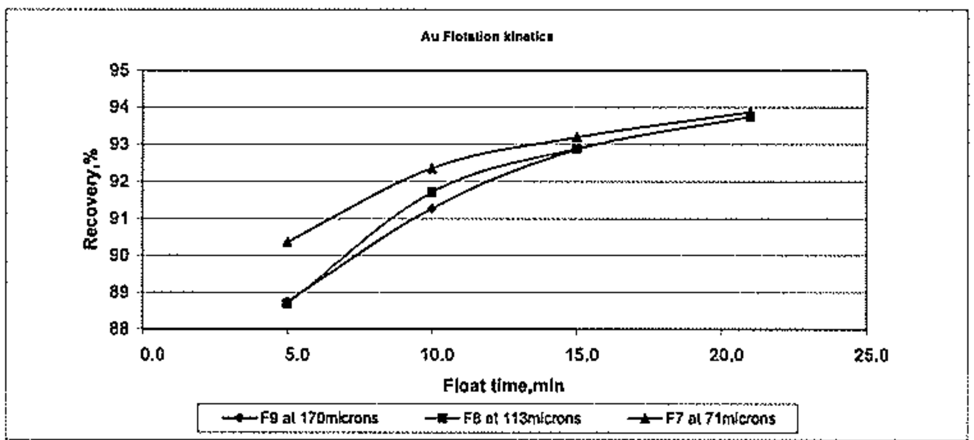
Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F4
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: F2 to F4 vary primary grind on low S composite blend

Flotation Balance

Product	Weight		Assay		Distribution	
			Au	S(tot)	Au	S(tot)
	(g)	(%)	(g/t)	(%)	(%)	(%)
Rougher Concentrate 1	101.0	10.6	7.07	27.50	88.7	92.4
Rougher Concentrate 2	49.6	5.2	0.41	1.92	2.5	3.2
Rougher Concentrate 1+2	**150.5**	**15.8**	**4.88**	**19.08**	**91.3**	**95.6**
Rougher Concentrate 3	45.6	4.8	0.28	0.87	1.6	1.3
Rougher Concentrate 1+2+3	**196.1**	**20.6**	**3.81**	**14.84**	**92.9**	**96.9**
Rougher Concentrate 4	37.8	4.0	0.19	0.48	0.9	0.6
Total Flotation Concentrate	**234.0**	**24.6**	**3.22**	**12.52**	**93.8**	**97.5**
Flotation Tailings	717.6	75.4	0.07	0.11	6.2	2.5
Calculated Feed	**951.6**	**100.0**	**0.85**	**3.16**	**100.0**	**100.0**
Measured Feed			0.72	3.36		



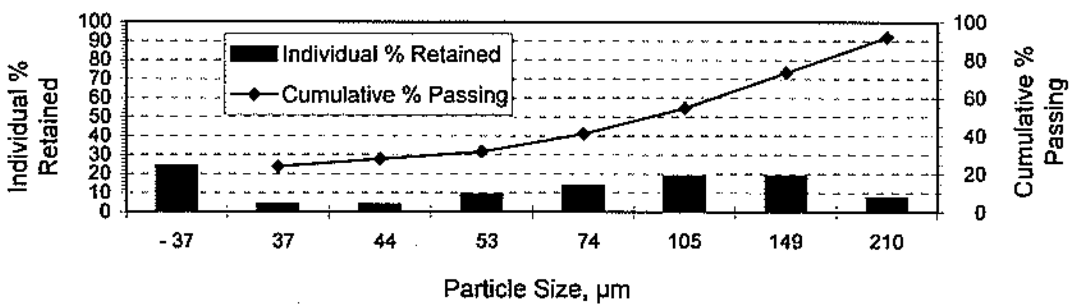
SIZE ANALYSIS REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project **Date:** 25-Aug-08
Test: F4 **Project:** 0805107
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)
Grind: 1 kg sample ground for 4'30" minutes at 65% solids in stainless steel mill #3

Sieve Size		Individual	Cumulative
Tyler Mesh	Micrometers	% Retained	% Passing
65	210	7.8	92.2
100	149	18.8	73.4
150	105	18.6	54.8
200	74	13.6	41.2
270	53	9.4	31.8
325	44	3.9	27.9
400	37	3.9	24.0
Undersize	- 37	24.0	-
TOTAL:		100.0	

80 % Passing Size (µm) = 170

Size Distribution



FLOTATION PROCEDURES

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F5
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: Same as F3 but use PAX and CuSO4

Stage	PAX	A208	$CuSO_4$	MIBC	Time, minutes Grind	Cond.	Float	pH	Comments
Grind (1 kg),Mill#3					6.0			8.6	target P80 = 105 u
ROUGHER FLOTATION									
Condition			75			3			
	25					1			
Rougher Float 1				30			5	8.5	brownish sph init with py
Condition	15					1			
Rougher Float 2				30			5	8.5	darker grey~2min
Condition	10					1			
Rougher Float 3				7			5	8.5	barren after 3min
Condition	10					1			
Rougher Float 4				7			6	8.6	
TOTAL REAGENTS	60	0	75	as reqd					

FLOTATION TEST METALLURGICAL BALANCE

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F5
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: Same as F3 but use PAX and $CuSO_4$

Flotation Balance

Product	Weight		Assay		Distribution	
			Au	S(tot)	Au	S(tot)
	(g)	(%)	(g/t)	(%)	(%)	(%)
Rougher Concentrate 1	123.4	12.9	6.24	19.00	88.2	92.5
Rougher Concentrate 2	47.0	4.9	0.37	1.58	2.0	2.9
Rougher Concentrate 1+2	**170.4**	**17.7**	**4.62**	**14.20**	**90.2**	**95.4**
Rougher Concentrate 3	41.9	4.4	0.20	0.55	1.0	0.9
Rougher Concentrate 1+2+3	**212.2**	**22.1**	**3.75**	**11.51**	**91.1**	**96.3**
Rougher Concentrate 4	44.4	4.6	0.16	0.37	0.8	0.6
Total Flotation Concentrate	**256.6**	**26.7**	**3.13**	**9.58**	**91.9**	**96.9**
Flotation Tailings	703.5	73.3	0.10	0.11	8.1	3.1
Calculated Feed	**960.2**	**100.0**	**0.91**	**2.64**	**100.0**	**100.0**
Measured Feed			0.72	3.36		

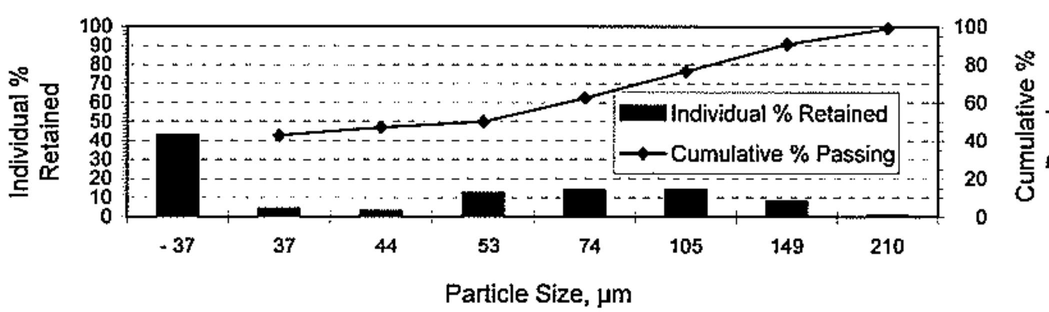
SIZE ANALYSIS REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Date: 25-Aug-08
Test: F5
Project: 0805107
Sample: Comp 08-1 & 0 8-2 (low S, blended 1:1)
Grind: 1 kg sample ground for 6 minutes at 65% solids in stainless steel mill #3

Sieve Size		Individual	Cumulative
Tyler Mesh	Micrometers	% Retained	% Passing
65	210	0.8	99.2
100	149	8.3	90.9
150	105	14.3	76.6
200	74	14.0	62.6
270	53	12.5	50.0
325	44	3.0	47.0
400	37	4.3	42.7
Undersize	- 37	42.7	-
TOTAL:		100.0	

80 % Passing Size (µm) = 115

Size Distribution



FLOTATION PROCEDURES

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F6
Sample: Comp 08-5 & 0 8-6 (high S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: Scoping Kinetic Flotation (F6 to F9) vs primary grind size, F6 uses a prefloat for free gold, similar to F1

	Reagents					Time, minutes				
Stage	A6697	SIPX	A208	$CuSO_4$	MIBC	Grind	Cond.	Float	pH	Comments
Grind (1 kg),Mill#3						15.0			8.7	target P80 = 53u
ROUGHER FLOTATION										
Condition	30						1			
Rougher Float 1					13			5	8.6	gold prefloat
										silvery froth for 3min than darker grey
Condition		15	15				1			
Rougher Float 2					7			5	8.7	golden froth appearance ~1 min
Condition		10	10				1			
Rougher Float 3					17			5	8.7	slightly grey
Condition		10	10				1			
Rougher Float 4					17			6	8.6	
TOTAL REAGENTS	30	35	35	0	53					

FLOTATION TEST METALLURGICAL BALANCE

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F6
Sample: Comp 08-5 & 0 8-6 (high S, blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: Scoping Kinetic Flotation (F6 to F9) vs primary grind size, F6 uses a prefloat for free gold, similar to F1

Flotation Balance

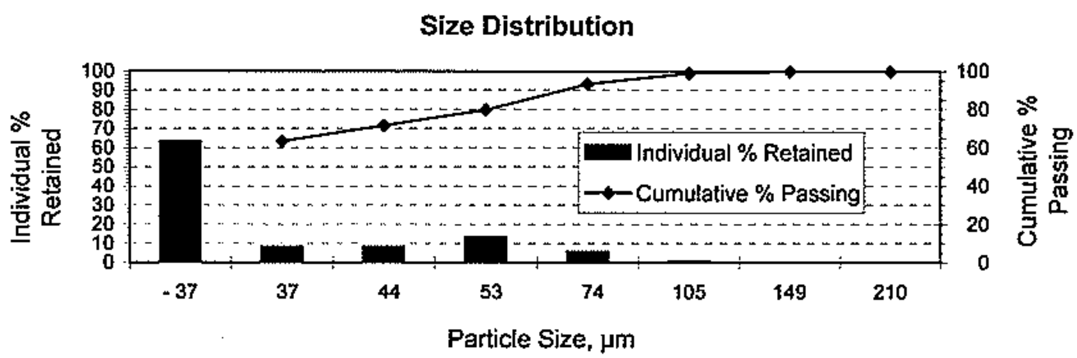
Product	Weight		Assay			Distribution		
			Au	S(tot)	C(org)	Au	S(tot)	C(org)
	(g)	(%)	(g/t)	(%)	(%)	(%)	(%)	(%)
Rougher Concentrate 1	66.0	6.9	24.07	9.44	0.70	83.6	16.6	36.0
Rougher Concentrate 2	109.6	11.4	2.61	27.60	0.18	15.1	80.6	15.4
Rougher Concentrate 1+2	**175.6**	**18.2**	**10.68**	**20.77**	**0.38**	**98.7**	**97.2**	**51.4**
Rougher Concentrate 3	73.2	7.6	0.12	0.85	0.13	0.5	1.7	7.4
Rougher Concentrate 1+2+3	**248.8**	**25.8**	**7.57**	**14.92**	**0.30**	**99.2**	**98.9**	**58.8**
Rougher Concentrate 4	61.3	6.4	0.05	0.26	<0.01	0.2	0.4	0.5
Total Flotation Concentrate	**310.1**	**32.2**	**6.09**	**12.02**	**0.25**	**99.3**	**99.3**	**59.3**
Flotation Tailings	653.0	67.8	0.02	0.04	0.08	0.7	0.7	40.7
Calculated Feed	**963.1**	**100.0**	**1.97**	**3.90**	**0.13**	**100.0**	**100.0**	**100.0**
Measured Feed			2.19	4.05				

SIZE ANALYSIS REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Date: 25-Aug-08
Test: F6
Project: 0805107
Sample: Comp 08-5 & 0 8-6 (high S, blended 1:1)
Grind: 1 kg sample ground for 15 minutes at 65% solids in stainless steel mill #3

Sieve Size		Individual	Cumulative
Tyler Mesh	Micrometers	% Retained	% Passing
65	210	0.0	100.0
100	149	0.0	99.9
150	105	0.8	99.1
200	74	5.8	93.4
270	53	13.4	79.9
325	44	8.3	71.6
400	37	8.2	63.4
Undersize	- 37	63.4	-
TOTAL:		100.0	

80 % Passing Size (μm) = 53



ASSAY REPORT

Client: Pinnacle Mines Ltd. - Silver Coin Project
Sample: Flotation Products as per ID

Date: 25-Aug-08
Project: 0805107

Items	Unit	Sample ID						Detection Limits		Analytical
		F6 Cut tails	F6 Ro Conc 4	F6 Ro Conc 3	F6 Ro Conc 2	F6 Ro Conc 1	RE:F6 Cut Tails	Min.	Max.	Method
Au	g/mt	0.02	0.05	0.12	2.61	24.07	0.02	0.01	5000	FA/AAS
S(tot)	%	0.04	0.26	0.85	27.60	9.44	0.04	0.01	20	Leco
C(Org)	%	0.08	<0.01	0.13	0.18	0.70	0.08	0.01	100	Leco
Al	ppm	50292	64369	61810	33022	54815	50800	100	50000	ICPM
Sb	ppm	<5	13	11	<5	25	<5	5	2000	ICPM
As	ppm	<5	<5	<5	<5	<5	<5	5	10000	ICPM
Ba	ppm	3874	4111	3965	496	766	3891	2	10000	ICPM
Bi	ppm	4	<2	3	<2	<2	4	2	2000	ICPM
Cd	ppm	<0.2	<0.2	42	112	362	<0.2	0.2	2000	ICPM
Ca	ppm	14489	18480	17688	7645	13993	14695	100	100000	ICPM
Cr	ppm	472	1160	1055	577	894	472	1	10000	ICPM
Co	ppm	11	20	20	47	29	11	1	10000	ICPM
Cu	ppm	19	56	83	298	851	19	1	20000	ICPM
Fe	ppm	16793	28938	29785	252047	80662	16936	100	50000	ICPM
La	ppm	10	14	12	3	6	10	2	10000	ICPM
Pb	ppm	93	357	506	3799	30565	99	2	10000	ICPM
Mg	ppm	6922	10330	9949	5463	9335	6997	100	100000	ICPM
Mn	ppm	1380	2185	2099	948	1798	1405	1	10000	ICPM
Hg	ppm	<3	<3	<3	<3	<3	<3	3	10000	ICPM
Mo	ppm	16	34	31	30	33	17	1	1000	ICPM
Ni	ppm	241	600	544	233	399	250	1	10000	ICPM
P	ppm	657	687	669	409	493	663	100	50000	ICPM
K	ppm	49552	54892	53991	27439	45055	49224	100	100000	ICPM
Sc	ppm	8	15	14	8	13	9	1	10000	ICPM
Ag	ppm	<0.5	0.9	3.3	46.6	119.2	<0.5	0.5	500	ICPM
Na	ppm	4605	4145	4161	1925	3100	4646	100	100000	ICPM
Sr	ppm	127	140	139	62	99	130	1	10000	ICPM
Tl	ppm	<2	<2	<2	45	<2	<2	2	1000	ICPM
Ti	ppm	948	1657	1725	1077	1421	969	100	100000	ICPM
W	ppm	6	7	<5	<5	<5	6	5	1000	ICPM
V	ppm	62	111	108	67	105	64	1	10000	ICPM
Zn	ppm	59	836	7109	26785	55883	57	1	10000	ICPM
Zr	ppm	32	51	51	38	41	31	1	10000	ICPM

FLOTATION PROCEDURES

Client: Pinnacle Mines Ltd. - Silver Coin Project
Test: F7
Sample: Comp 08-5 & 0 8-6 (blended 1:1)

Date: 25-Aug-08
Project: 0805107
Operator: BG

Objective: F7 to F9 vary primary grind on high S composite blend

Stage	SIPX	A208	$CuSO_4$	MIBC	Time, minutes Grind	Cond.	Float	pH	Comments
Grind (1 kg),Mill#3					12.0			8.7	target P80 = 74 u
ROUGHER FLOTATION									
Condition	15	15				1			
Rougher Float 1				17			5	8.8	silvery gal froth appearance ~3'
									to barren
Condition	10	10				1			
Rougher Float 2				13			5	8.7	grey~2min
Condition	10	10				1			
Rougher Float 3				13			5	8.7	slightly grey
Condition	10	10				1			
Rougher Float 4				17			6	8.7	
TOTAL REAGENTS	45	45	0	59					