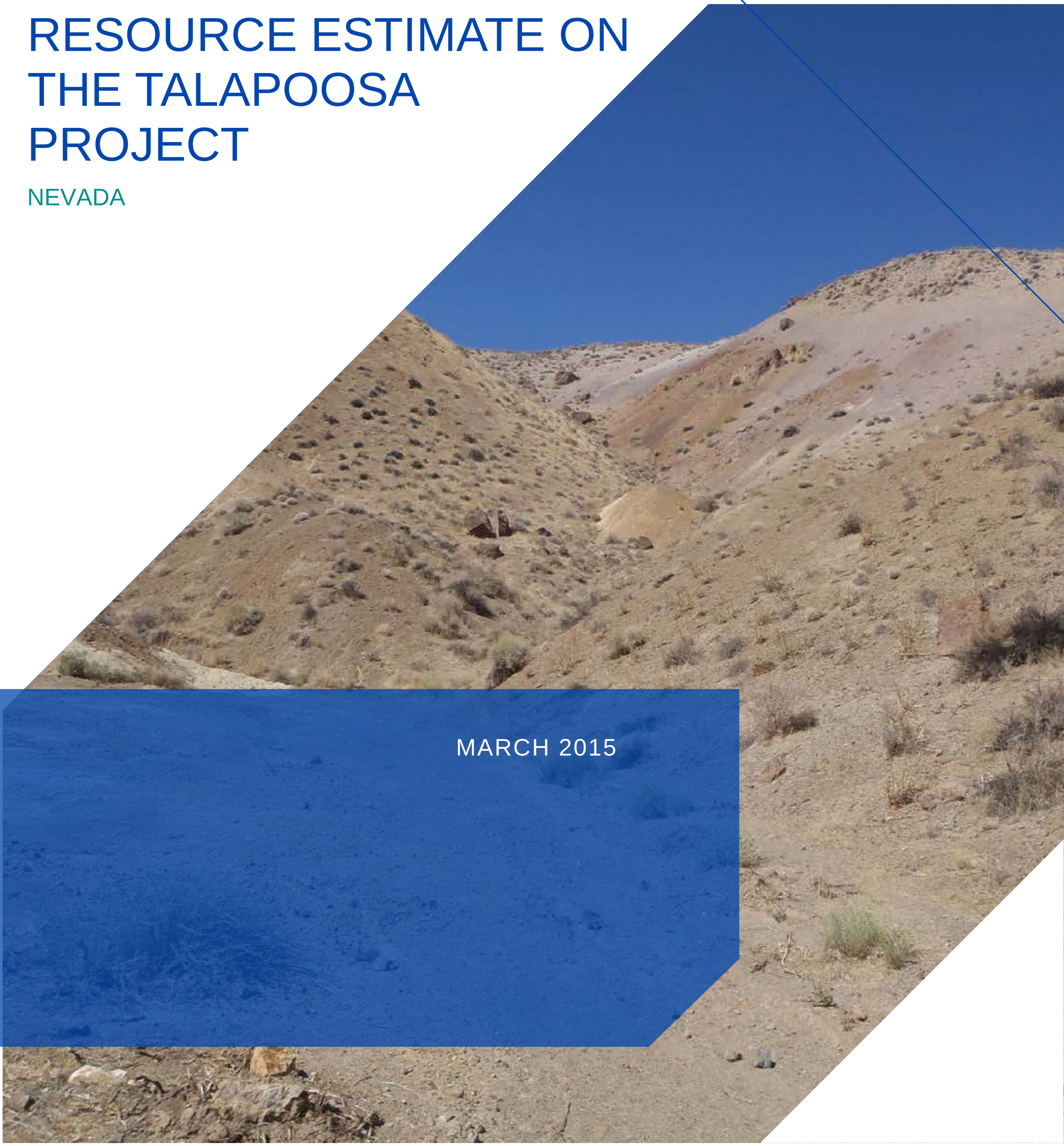
REPORT N^{O} 151-02327-00_RPT-01_R4

TECHNICAL REPORT AND RESOURCE ESTIMATE ON THE TALAPOOSA PROJECT

NEVADA



MARCH 2015

TECHNICAL REPORT AND RESOURCE ESTIMATE ON THE TALAPOOSA PROJECT

NEVADA

Timberline Resources

Project no: 151-02327-00_RPT-01_R4

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REVISION HISTORY

Rev. No	Issue Date	Prepared by	Description of revision
0	March 20, 2015	Todd McCracken	Preliminary draft for Client's review.
1	March 24, 2015	Todd McCracken	Revision 1 issued to Client.
2	March 25, 2015	Todd McCracken	Revision 2 issued to Client.
3	March 25, 2015	Todd McCracken	Revision 3 issued to Client.
4-BL	March 31, 2015	Todd McCracken	Revision 4 Black Line Version for TSX.
4	March 31, 2015	Todd McCracken	Revision 4 issued to Client.

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ABBREVIATIONS

UNITS OF MEASURE

above mean sea level .... amsl
acre .... ac
ampere .... A
annum (year) .... a
billion .... B
billion tonnes .... Bt
billion years ago .... Ga
British thermal unit ....BTU
Centimetre .... cm
cubic centimetre ....cm^3
cubic feet per minute .... cfm
cubic feet per second ....ft^3/s
cubic foot .... ft^3
cubic inch .... in
cubic metre .... m^3
cubic yard ....yd^3
Coefficients of Variation ....Cvs
day ....d
days per week .... d/wk
days per year (annum) ....d/a
dead weight tonnes ....DWT
decibel adjusted .... Ba
decibel .... dB
degree .... °
degrees Celsius ....°C
diameter ....ø
dollar (American) ....US$
dollar (Canadian) .... Cdn$
dry metric tond ....mt
foot ....ft
gallon .... gal
gallons per minute ....gpm
Gigajoule .... GJ
Gigapascal .... GPA
Gigawatt ....GW
Gram ....g
grams per litre ....g/L
grams per tonne ....g/t
greater than ....>
hectare (10,000 m2) ....ha
hertz .... Hz
horsepower ....hp
hour ....h
hours per day ....h/d
hours per week .... h/wk
hours per year ....h/a
inch ....in
kilo (thousand) ....k
kilogram ....kg
kilograms per cubic metre .... kg/m^3
kilograms per hour ....kg/h
kilograms per square metre .... kg/m^2
kilometre ....km
kilometre ....km
kilometres per hour ....km/h
kilopascal .... kPa
kiloton ....kt
kilovolt .... kV
kilovolt-ampere .... kVa
kilowatt .... kW
kilowatt hour .... kWh
kilowatt hours per tonne .... kWh/t
kilowatt hours per year .... kWh/a
less than .... <
litre L
litres per minute .... L/m
megabytes per second ....Mb/s
megapascal .... Mpa
megavolt-ampere ....Mva
megawatt .... MW
metre .... m
metres above sea level ....masl
metres Baltic sea level ....mbsl
metres per minute ....m/min
metres per second ....m/s
microns .... µm
milligram .... mg
milligrams per litre .... mg/L
millilitre .... mL
millimetre .... mm
million .... M
million bank cubic metres .... Mbm^3
million bank cubic metres per annum ... Mbm3/a
million tonnes .... Mt
minute (plane angle) ....'
minute (time) ....min
month .... mo
ounce ....oz
pascal .... Pa
centipoise .... mPa·s
parts per million .... ppm
parts per billion .... ppb

percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
short ton (2,000 lb)	st
short tons per day	st/d
short tons per year	st/y
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km2
square metre	m^2
three-dimensional	3D
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt

ACRONYMS

American Assay Laboratories	AAL
American Gold Capital US Inc.	American Gold
American Smelting and Refining Company	ASARCO
Athena Gold Inc.	Athena
Bateman Metallurgical Laboratories Inc.	Bateman
Bear Creek Mining Company	Bear Creek
Bondar Clegg & Company Ltd.	Bondar Clegg
Bureau of Land Management	BLM
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Carbon in pulp	CIP
Carbon-in-leach	CIL
Chesapeake Gold Corp.	Chesapeake
Cold Vapour Atomic Absorption	CVAA
Dawson Metallurgical Laboratories Inc.	Dawson
Environmental impact statement	EIS
Fast rolls at Dawson	DML
Final effluent	FE
Flameless Atomic Absorption	FLAA
Fred de Longchamps & Sons	Longchamps
Global positioning system	GPS
Gunpoint Exploration Ltd.	Gunpoint
Hazen Research Inc.	Hazen
Heinen-Lindstrom Consultants	HLC
High-pressure grinding rolls – double pass	HPGR-DP
High-pressure grinding rolls – single pass	HPGR-SP
High-pressure grinding rolls	HPGR
Homestake Mining Company	Homestake
Hunter Mining Laboratory	Hunter Mining
Induced polarization	IP
Inductively coupled plasma atomic emission spectroscopy	ICP-AES
Inductively coupled plasma	ICP
International Electrotechnical Commission	IEC
International Organization for Standardization	ISO
Inverse distance cubed	ID^3
Inverse distance squared	ID^2

Kennecott Copper Company .... Kennecott
Lead oxide .... PbO
Lower Bear Creek .... LBC
McClelland Laboratories Inc. .... McClelland
Mine Development Associates .... MDA
Minproc Engineers Inc. .... Minproc
Miramar Mining Corp. .... Miramar
National Instrument 43-101 .... NI 43-101
Nearest neighbour .... NN
Net smelter return .... NSR
Nevada Division of Environmental Protection .... NDEP
Newcrest Resources Inc. .... Newcrest
North American Datum .... NAD
Ordinary kriging .... OK
Oretest Pty Ltd .... Oretest
Pegasus Gold Corp. .... Pegasus
Pittsburgh Mineral and Environmental Technology Inc. .... PMET
Placer Dome U.S. Inc. .... Placer Dome
Qualified person .... QP
Quality assurance/quality control .... QA/QC
Reverse circulation .... RC
Rock Engineered Machinery Co. Inc. .... REMCO
Scanning electron microscopy – energy dispersive x-ray spectroscopy .... SEM-EDX
Semi-autogenous grinding .... SAG
Sierra Denali Minerals Inc. .... Sierra Denali Minerals
Sodium cyanide .... NaCN
Sodium isobutyl xanthate .... SIBX
Standard reference material .... SRM
Summit Valley Equipment & Engineering Inc .... Summit Valley
Talapoosa Mining Inc. .... TMI
Talapoosa Project .... the Project
Talapoosa Property .... the Property
Union Assay Office .... Union Assay
Universal Transverse Mercator .... UTM
Upper Bear Creek .... UBC
US Geological Survey .... USGS
Vertical Shaft Impact crusher .... VSI
Very low frequency .... VLF

1 SUMMARY

The Talapoosa Project (the Project or the Property) is located in the Talapoosa mining district in northwestern Nevada.

WSP has prepared this technical report on the Project at the request of Timberline Resources Inc (Timberline). This report complies with the standards set in National Instrument 43-101 (NI 43-101) and the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines. The effective date of this report and the resource estimation is March 24, 2015.

1.1 LOCATION AND PROPERTY DESCRIPTION

The Property is located in the Talapoosa mining district in northwestern Nevada. The district lies in Lyon County about 28 miles in a straight line east of Reno, Nevada, straddling the boundary between T18N and T19N, R24E, Mount Diablo Base and Meridian. Talapoosa lies on the eastern and southeastern flanks of the Virginia Range, one of the ranges of the Basin and Range Province.

The resource at Talapoosa is centered immediately south of a cluster of old mine workings in the SE/4 Section 3, T18N, R24E at coordinates 304,500 east, 4,369,300 north, Universal Transverse Mercator (UTM) Zone 11.

American Gold is a corporation incorporated under the laws of Nevada, USA and a wholly- owned subsidiary of Gunpoint Exploration (Gunpoint), and is the registered claim holder. All mining claims and mineral leases are in good standing and all taxes haves been paid in full. Timberline has an option agreement to acquire 100% of the Property from Gunpoint.

All permits to conduct exploration and reclamation bonds are currently in place to allow exploration to take place.

1.2 GEOLOGY

The Project lies in the western Basin and Range Province, a structural province of generally north trending mountain ranges and intervening valleys formed by regional extension during Tertiary time. The Sierra Nevada forms the western margin of the province. The Virginia Range, on whose east flank the Project is located, along with the Pine Nut Mountains, Wellington Hills, and Sweetwater Range to the south, forms one of four master fault-block ranges of this type that can be considered north-trending spurs of the Sierra Nevada.

The Project geology is composed of a thick sequence of Miocene-Pliocene volcanic and sedimentary rocks that overlie Mesozoic metamorphic and granite found throughout the Sierra Nevada, as described below.

Pyramid Sequence is a sequence of vesicular basalt, felsic ash-flow tuffs and hydrothermal eruption breccias associated with epithermal mineralization along the Appaloosa structure.

Kate Peak Formation hosts all of the known mineralization in the district and overlays the Pyramid Sequence. The Kate Peak Formation consists of dacitic tuff, tuff breccia, flows, lava dome carapace debris, and post-volcanic dacite porphyry sills or dykes.

Coal Creek (Canyon) Formation is a mixture of sand, silt and clay derived from pyroclastic volcanic rocks and unconformably overlays the Kate Peak formation.

Lousetown Formation is a vesicular olivine basalt or pyroxene andesite with flows ranging from a few feet thick to as much as 300 ft. in thickness and unconformably overlies the Coal Creek Formation.

Alteration and mineralization on the Project is typical of a low-sulphidation epithermal. The mineralization was divided into the following domains, separated by north-northwest fault:

- Bear Creek Hanging-Wall Vein System/Domain bounded by Ripper Fault to south and Cabin Fault to north. The Hanging-Wall vein is comprised predominantly of massive white sulphide poor silica with typical low-sulphidation epithermal textures, including recrystallization, coliform and crustiform banding, adularia bands, amethyst etc.
- Bear Creek Footwall Vein System/Domain bounded by Cabin Fault to south and Talapoosa (South) Fault to the north. The Footwall vein is more sulphide rich, associated with a number of gangue phases including, red hematitic silica, chlorite and minor white to clear silica.
- Main Zone Vein System/Domain bounded by Talapoosa (South) Fault to the south and Opal/Dyke Fault to the north.

The mineralization at both Dyke Adit and East Hill shows similarities in appearance and texture to that of the Hanging-Wall Zone at Bear Creek.

1.3 DRILLING

In 2011, Gunpoint completed seven PQ diamond drillholes totaling 4,642 m in the resource area. The purpose of the drilling was to confirm the mineralization and to demonstrate that inclined drilling programs instead of vertical drilling combined with screen metallic assays could upgrade the resource, compared to the previous methodologies employed by previous operators.

The previous operators' drilling, logging, and sampling practices all meet industry standards and are suitable for use in resource estimation.

Drilling campaigns have been completed by eight previous operators, totalling 298,305 ft. from 586 holes. The drilling was completed between 1977 and 1991 and was a mix of coring, reverse circulation (RC), and rotary drilling. Some historic drilling or sampling procedures could not be verified and as such, the data was not included in the resource estimation.

1.4 MINERAL PROCESSING AND METALLURGICAL TESTING

Between 1981 and 1999, there were 12 metallurgical testing programs conducted on the Property, by various stakeholders. An additional metallurgical testing program was completed in early 2015. Testing in the 1980s and in 2014/2015 focused mainly on heap leaching. Some work in the 1990s was more focused on agitated leaching, flotation, cyanidation of flotation concentrate, gravity concentration with cyanidation of the gravity tailings and bio-oxidation before cyanidation.

The testing programs have focused on evaluation material from the Bear Creek and Main Zones. A smaller amount of work was conducted on material from Dyke Adit and East Hill Zones. In general during the 1981 to 1999 work, the Main Zone material was described as oxidized and the Bear Creek Zone material was described as unoxidized, or sulfide. The 2014/2015 work mainly considered material from the Bear Creek Zone. That Bear Creek Zone material was divided into hanging wall and footwall zones, both oxidized and unoxidized. This compositing was based on the refined interpretation of the controls of mineralization for the property, as described in the TetraTech Talapoosa resource summary (2013).

Work in the 1980s showed that the Main Zone material was for the most part more amenable to simulated heap leach processing than the Bear Creek Zone material. The Bear Creek Zone material was usually more refractory to cyanidation. Available mineralogy generally indicated that the majority of the gold was present as electrum (gold silver), so the leach kinetics were slow, leading to long heap leach times. Both ore types were sensitive to feed size with respect to gold and silver recovery. Very fine crush sizes were required to maximize gold and silver recoveries by heap leaching. Testing included evaluation of various size reduction equipment for achieving the feed sizes tested. The use of high-pressure grinding rolls (HPGR) for size reduction generally resulted in increased gold and silver recoveries, but it was unclear whether or not those improvements resulted simply from generation of a finer particle size distribution and the resulting gold and silver liberation, for the HPGR product. In the case of the unoxidized Bear Creek Zone material, a significant portion of the contained gold and silver was believed to be locked in sulfide mineral grains.

The recovery of the gold and silver by flotation was generally high for the sulphidic Bear Creek Zone material, but was lower for most of the Main Zone material. Very little testing has been conducted to evaluate further processing of flotation concentrates for recovery of contained gold and silver.

Results from the recently completed (2014/2015) testing generally supported results from the earlier heap leach testing. The Bear Creek hanging wall and footwall unoxidized materials were shown to respond moderately well to simulated heap leaching treatment, at relatively fine (6.3 mm or finer) feed sizes. Gold recoveries tended to be lower for the Bear Creek Zone footwall material than for the Bear Creek Zone hanging wall material. In some cases, the use of HPGR for size reduction resulted in increased gold and silver recoveries, but those increases may have resulted from the finer particle size distributions generated by HPGR. It was noted that further optimization of agglomerating conditions will be required to ensure that the finely crushed ore will maintain acceptable permeability during commercial heap leaching. Only bottle roll testing was conducted on Bear Creek oxidized and Main Zone unoxidized materials. Those tests showed that the materials tested were amenable to cyanidation treatment.

Available metallurgical testing results suggest that conventional heap leaching at a relatively fine crush size is likely the best approach for processing the oxide materials from the Talapoosa property. The more refractory nature and increased feed size suggests that milling followed by flotation treatment, and possibly either direct leaching or oxidation followed by leaching of the flotation concentrate may be a preferred processing route for some of the unoxidized materials.

1.5 RESOURCE ESTIMATION

The resource estimation completed by Tetra Tech in 2012 has been validated by WSP. The resource is effective March 24, 2015. The resource estimation was generated for five higher-grade vein domains and five lower- grade host rock domains. Estimations were completed using a three-pass estimation method with the following set parameters used on each estimation pass:

- Minimum and maximum number of samples to be used;

- Maximum number of samples from any borehole;
- Search ellipse dimensions.

The search ellipse orientation was determined by dynamic anisotropy in order to better control the search direction.

Specific gravity values were determined for the vein material altered volcanics and the oxidized material. The specific gravity values were derived from 310 measurements collected by Gunpoint.

The block model used a parent block size of 30 ft. by 30 ft. by 30 ft. and sub-celled to better fill the wireframe volumes. No rotation was applied to the model. The resource estimation method used was ordinary kriging (OK) with inverse distance squared (ID^2) and nearest neighbour (NN) used for validation.

Table 1.1 is a summary of the resource estimation at Talapoosa.

Table 1.1 - Talapoosa Resource Summary

Summary	Cut-Off (oz/ton)	Tons	Au (oz/ton)	Ag (oz/ton)	Tonnes	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Oxide Measured	0.013	3,126,050	0.038	0.553	2,835,890	1.29	18.96	117,253	1,728,323
Sulphide Measured	0.013	14,044,820	0.036	0.481	12,741,180	1.22	16.50	501,215	6,760,763
Total Measured		**17,170,870**	**0.036**	**0.494**	**15,577,070**	**1.23**	**16.95**	**618,468**	**8,489,086**
Oxide Indicated	0.013	1,412,000	0.032	0.416	1,280,900	1.10	14.25	45,328	586,999
Sulphide Indicated	0.013	12,681,600	0.028	0.361	11,504,500	0.94	12.36	349,005	4,573,274
Total indicated		**14,093,600**	**0.028**	**0.366**	**12,785,400**	**0.96**	**12.55**	**394,334**	**5,160,273**
Total M&I		**31,264,470**	**0.032**	**0.437**	**28,362,470**	**1.11**	**14.97**	**1,012,802**	**13,649,358**
Oxide Inferred	0.013	1,762,000	0.027	0.065	1,598,000	0.93	2.24	47,745	115,115
Sulphide Inferred	0.013	9,436,000	0.020	0.218	8,560,000	0.68	7.48	185,787	2,057,651
Total Inferred		**11,198,000**	**0.021**	**0.194**	**10,158,000**	**0.72**	**6.65**	**233,532**	**2,172,766**

1.6 RECOMMENDATIONS

Additional exploration expenditures are warranted to improve the viability of the Project. It is recommended that Timberline undertake a two-phased program that will concentrate on the metallurgy for the open pit potential of the mineralized deposit and complete step out drilling along strike of the known resource. Phase 1 would focus on completion of a Preliminary Economic Assessment of the Project based on the current resource. The estimated cost of Phase 1 would be US$145,000.

Phase 2A would focus on expanding and upgrading the resources as well as collect material suitable for additional metallurgical testing of the sulphide and oxide horizons of the resource. The estimated cost of Phase 2 would be US$924,000.

Phase 2B would focus on the metallurgical test program which would utilize the sample collected in Phase 2A. The estimated cost of Phase 2B would be US$600,000.

2 INTRODUCTION

Timberline entered an Option Agreement (Agreement) in March, 2015 to acquire the Talapoosa Project ("the Project") from Gunpoint Exploration Inc. ("Gunpoint"). WSP Canada Inc. ("WSP") was commissioned by Timberline to update a technical report on the Project originally completed by Tetra Tech on April 12, 2013.

WSP has prepared this report in accordance with NI 43-101 Standards of Disclosure for Mineral Projects. Mr. Todd McCracken, one of the qualified persons (QP) of the original Tetra Tech report if one of the QPs on this updated report by WSP.

This report was prepared by WSP at the request of Mr. Steven Osterberg, Vice President of Timberline.

Timberline is a Coeur d'Alene, Idaho-based company, trading on the New York MKT, and Toronto Stock Exchange Venture under the symbols TLR and TBR, respectively.

The issue date of this report is March 24, 2015. The effective date of the resource estimate is March 24, 2015.

The following qualified persons (QPs) completed a site visit of the Property:

- Todd McCracken, P.Geo. visited the site from September 23 to 25, 2012 inclusive as the QP for the original Tetra Tech report.
- Jack McPartland, P. Eng. has not visited the site.

WSP considers the site visit current, per NI 43-101CP, Section 6.2, on the basis that no material work has been completed on the Property since the date of the site visit and all practices and procedures documented were reviewed. WSP has reviewed the financial statements filed by Gunpoint and posted on SEDAR which supports the notion that the work completed on the Project by Gunpoint in 2013 and 2014 does not have any material change on the resource estimation (Wong, 2014).

All units of measurement used in this technical report are in US imperial unless otherwise indicated. All dollar figures discussed in this technical report are in US dollars unless otherwise indicated.

All data sourced for this report are identified in Section 19.0 of this report.

3 RELIANCE ON OTHER EXPERTS

The QPs who prepared this report relied on information provided by experts who are not QPs. The relevant QPs believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

- Todd McCracken, P.Geo., relied upon Mr. Ian D. Robertson of the law firm of Robertson Neil LLP for matters pertaining to mineral claims and mining leases as well as the acquisition agreement as disclosed in Section 4.0.
- Todd McCracken, P.Geo., relied upon Mr. Edward Devenyns, Mineral Land Consultant for matters pertaining to mineral claims and mining leases as disclosed in Section 4.0.

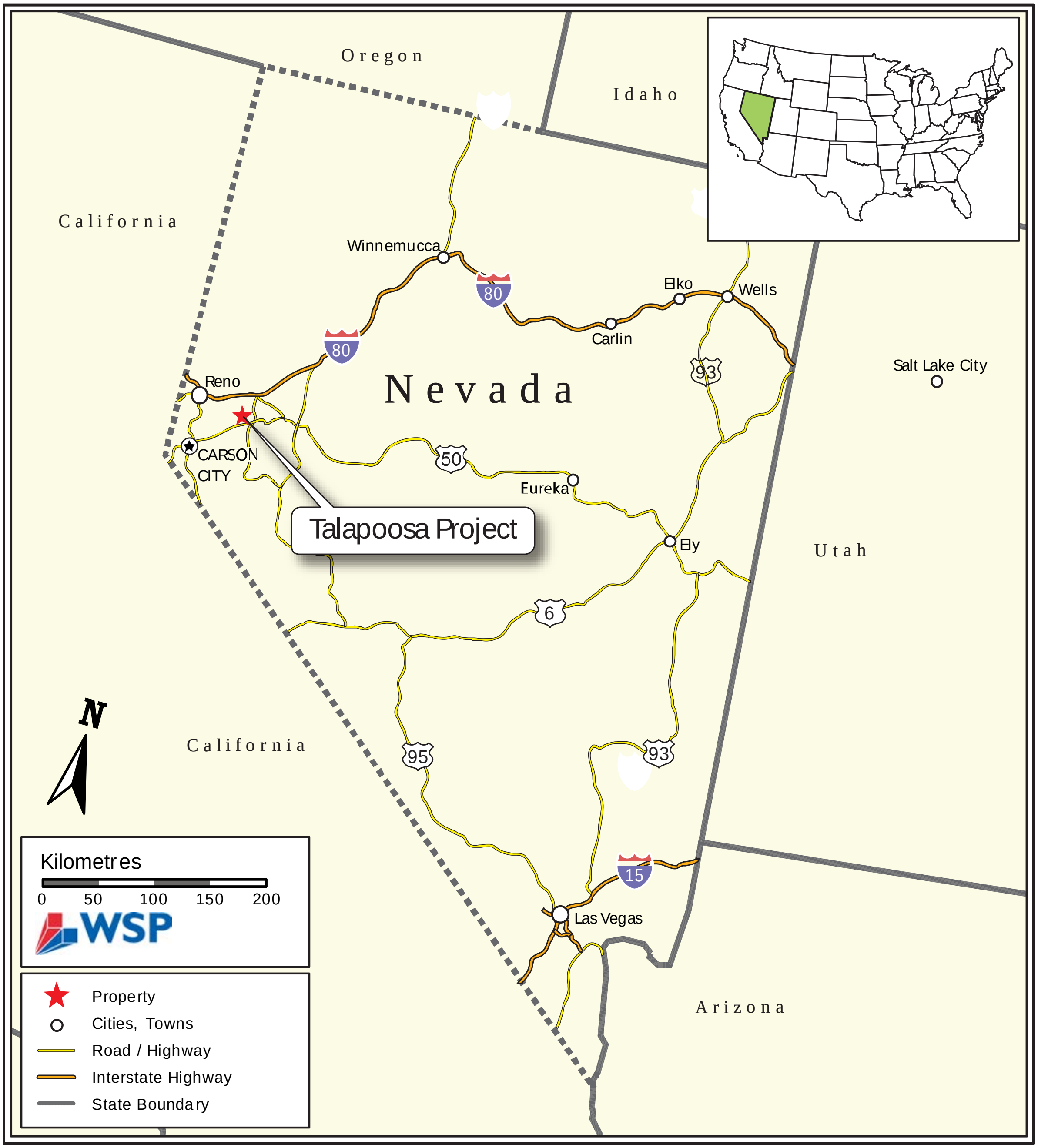
4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Project is located in the Talapoosa mining district in northwestern Nevada. The district lies in Lyon County about 28 miles (in a straight line) east of Reno, Nevada, straddling the boundary between T18-19N, R24E (Figure 4.1). The Project is on the Stockton Well (1:24,000), Carson City (1:100,000), and Reno (1:250,000) topographic maps.

The resource at Talapoosa is centered immediately south of a cluster of old mine workings in the SE/4 Section 3, T18N, R24E at coordinates 304,500 East, 4,369,300 North, UTM Zone 11 (Danley 1999a).

Figure 4.1 - Location Map



4.2 LAND AREA

American Gold is the registered, legal and beneficial owner or leassee of the Talapoosa Claims (described in Table 4.1 and displayed in Figure 4.2) free and clear of any encumbrances, agreements, adverse claims, royalties, profit interests or other payments in the nature of a royalty, recorded or unrecorded, except:

- The unpatented mining claims are located on land controlled by the US Department of the Interior Bureau of Land Management (BLM), which required annual mining claim maintenance fees to be timely paid by August 31, 2015 and a notice to hold mining claims to be timely recorded in the Official Records of the Lyons County Recorder's Office by October 31, 2015.
- American Gold is a corporation incorporated under the laws of Nevada, USA and is a wholly-owned subsidiary of Gunpoint, a corporation incorporated under the laws of BC, Canada.
- Gunpoint acquired all of the issued and outstanding shares of American Gold US on November 26, 2010 from American Gold, a wholly-owned subsidiary of Chesapeake Gold Corp. (Chesapeake) pursuant to an acquisition agreement (the Acquisition Agreement) made between Gunpoint, American Gold, and Chesapeake, and dated June 15, 2010 as amended July 15, 2010 and November 10, 2010.
- Pursuant to the terms of the Acquisition Agreement, Chesapeake's subsidiary American Gold was issued 31,977,899 common shares in the capital stock of Gunpoint, representing approximately 81.8% of the then issued and outstanding shares of Gunpoint in satisfaction of the purchase price of the shares of American Gold.

To clarify the transaction:

- Chesapeake owns 81.8% of Gunpoint;
- Gunpoint owns 100% of American Gold US;
- American Gold US owns Talapoosa Claims subject to encumbrances.

American Gold owns 509 unpatented mining claims at Talapoosa located in Sections 2, 3, 4, 5, 8, 9, 10, 11, and 14 of T18N, R24E and Section 6 of T18N, R25E and Sections 20,22,26, 28, 32, 34, and 36, T19N, R24E, Mount Diablo Base and Meridian of which two are located on the resource area. In addition, through a lease with Sierra Denali Minerals Inc. (Sierra Denali Minerals) described below, American Gold leases 26 unpatented lode claims in Sections 2, 3, and 11, T18N, R24E and Section 34, T19N, R24E, of which nine are located on the resource area.

American Gold also owns fee land consisting of the N/2 Section 3 and the N/2 S/2 Section 3, T18N, R24E, excluding certain public lands within this section, which is located on the resource area. The annual property taxes haves been timely paid to Lyon County Treasurers Office and are considered current.

American Gold leases Sections 27 (excepting a 50 ft-wide road easement), 29, 33, and 35, T19N, R24E from the Sario Livestock Company. American Gold also leases Section 21 and 23, T19NR24E from Nevada Bighorn Unlimited. Their leases are not located on the resource area.

The claims, leased fee land, and fee land owned by American Gold are contiguous.

American Gold paid the federal annual mining claim maintenance fees for the annual assessment years September 1, 2011, to September 1, 2012, September 1, 2012, to September 1, 2013, September 1, 2013, to September 1, 2014, and September 1, 2014, to September 1, 2015, and the unpatented mining claims remain and will be in good standing until September 1, 2015. American Gold has recorded in the Office of the Lyon County Recorder, the notices of intent to hold the claims in accordance with Nevada law through October 31, 2015.

Table 4.1 lists the 91 mining claims owned or controlled by American Gold within the resource area.

Figure 4.2 shows the general location of the Property controlled by American Gold.

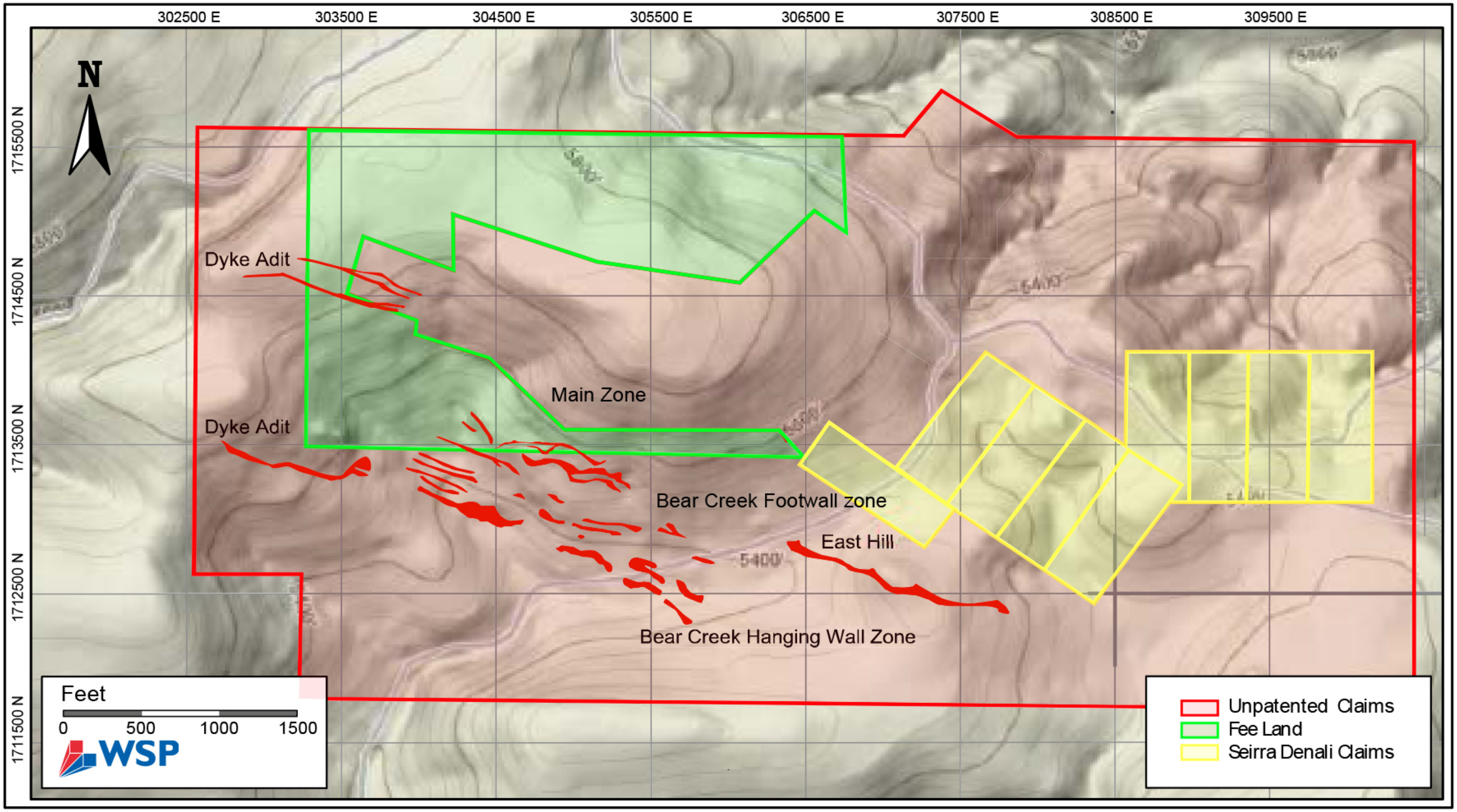
Table 4.1 - Claim Owned or Leased by American Gold within the Resource Area

No. of Claims	Claim Name and/or No.	County Recording Document No.	BLM NMC No.
1	Alpha	369121	NMC912930
2	Alpha Fr	369122	NMC912931
3	Cuba	369123	NMC912932
4	Equity 1	369124	NMC912933
5	Equity 2	369125	NMC912934
6	First Strike	369126	NMC912935
7	Georgia Amended	369127	NMC912936
8	Justice	369128	NMC912937
9	Justice Fr	369129	NMC912938
10	Lincoln 3	369130	NMC912939
11	Omega	369131	NMC912940
12	Second Strike	369132	NMC912941
13	Virginia	369133	NMC912942
14	Virginia Extension	369134	NMC912943
15	Wedge 1	369135	NMC912944
16	Wedge 2	369136	NMC912945
17	Wedge 3	369137	NMC912946
18	AGC 15	369152	NMC912961
19	AGC 16	369153	NMC912962
20	AGC 17	369154	NMC912963
21	AGC 18	369155	NMC912964
22	AGC 37	369174	NMC912983
23	AGC 38	369175	NMC912984
24	AGC 39	369176	NMC912985
25	AGC 40	369177	NMC912986

No. of Claims	Claim Name and/or No.	County Recording Document No.	BLM NMC No.
26	AGC 41	369178	NMC912987
27	AGC 42	369179	NMC912988
28	AGC 43	369180	NMC912989
29	AGC 44	369181	NMC912990
30	AGC 45	369182	NMC912991
31	AGC 46	369183	NMC912992
32	AGC 47	369184	NMC912993
33	AGC 48	369185	NMC912994
34	AGC 49	369186	NMC912995
35	AGC 50	369187	NMC912996
36	AGC 51	369188	NMC912997
37	AGC 52	369189	NMC912998
38	AGC 53	369190	NMC912999
39	AGC 54	369191	NMC913000
40	AGC 55	369192	NMC913001
41	AGC 56	369193	NMC913002
42	AGC 57	369194	NMC913003
43	AGC 58	369195	NMC913004
44	AGC 59	369196	NMC913005
45	AGC 60	369197	NMC913006
46	AGC 61	369198	NMC913007
47	AGC 62	369199	NMC913008
48	AGC 63	369200	NMC913009
49	AGC 64	369201	NMC913010
50	AGC 65	369202	NMC913011
51	AGC 66	369203	NMC913012
52	AGC 67	369204	NMC913013
53	AGC 68	369205	NMC913014
54	AGC 69	369206	NMC913015
55	AGC 70	369207	NMC913016
56	AGC 71	369208	NMC913017
57	AGC 72	369209	NMC913018
58	AGC 73	369210	NMC913019
59	AGC 74	369211	NMC913020

No. of Claims	Claim Name and/or No.	County Recording Document No.	BLM NMC No.
60	AGC 75	369212	NMC913021
61	AGC 76	369213	NMC913022
62	AGC 77	369214	NMC913023
63	AGC 78	369215	NMC913024
64	AGC 79	369216	NMC913025
65	AGC 80	369217	NMC913026
66	AGC 81	369218	NMC913027
67	AGC 82	369219	NMC913028
68	AGC 83	369220	NMC913029
69	AGC 84	369221	NMC913030
70	AGC 85	369222	NMC913031
71	AGC 86	369223	NMC913032
72	AGC 87	369224	NMC913033
73	AGC 88	369225	NMC913034
74	AGC 93	369230	NMC913039
75	AGC 94	369231	NMC913040
76	AGC 95	369232	NMC913041
77	AGC 96	369233	NMC913042
78	Washington	-	NMC117406
79	Lincoln #1	-	NMC117407
80	Lincoln #2	-	NMC117408
81	Jefferson	-	NMC117409
82	Roosevelt	-	NMC117410
83	Essex 1	369241	NMC912904
84	Essex 2	369242	NMC912905
85	Essex 3	369243	NMC912906
86	Essex 4	369244	NMC912907
87	Essex 5	369245	NMC912908
88	Lexington 1	369246	NMC912909
89	Lexington 2	369247	NMC912910
90	Lexington 3	369248	NMC912911
91	Lexington 4	369249	NMC912912

Figure 4.2 - Claims Map



On March 17, 2015, Timberline completed a Definitive Agreement (“Agreement”) to acquire an option to purchase Gunpoint’s 100% owned Talapoosa property (the “Property”) for a period of thirty months from the effective date of the Agreement. During the option period, the Agreement grants Timberline the exclusive and irrevocable option to purchase all of Gunpoint’s interest in the Property. In consideration thereof, Timberline agreed to pay Gunpoint $300,000 in cash and to issue 2,000,000 shares of Timberline’s common stock, to be vested in 500,000 share increments at 6 months, 12 months, 18 months, and 24 months from the effective date of the closing of the option acquisition transaction. In addition, during the thirty-month option period, Timberline assumes responsibility for the payment of all property holding costs.

Within 90 days of exercise of the option granted in the Option Agreement, Timberline agrees to pay Gunpoint $10,000,000 in cash as consideration for purchase of the Property. In addition, Gunpoint’s parent company, will retain a 1% NSR on the mineral production from the Property, subject to a purchase option by Timberline for $3,000,000.

In addition to terms of the Agreement, Timberline has agreed to provide contingent consideration to Gunpoint’s parent company based on the future price of gold. For a period of five (5) years following the exercise of the option, should the daily price of gold (as determined by the London PM Fix) be fixed at U.S. $1,600 per ounce or greater for a period of ninety (90) consecutive trading days (“Trigger Event”), Timberline will pay Gunpoint’s parent company an additional payment of $10,000,000, comprised of cash and potentially, at Timberline’s discretion, shares of Timberline’s common stock within 90 days of the date that the Trigger Event is deemed to have occurred.

4.3 AGREEMENTS AND ENCUMBRANCES WITH THE RESOURCE AREA

4.3.1 SIERRA DENALI MINERALS INC. (VON HAFFTEN) AGREEMENT

Talapoosa Mining, Inc. leased 26 unpatented mining claims from the estates of Alexander von Hafften and Sebelle Harden von Hafften in a lease originally dated July 14, 1990, and amended on August 25, 1998. These claims are now owned by Sierra Denali Minerals and leased by American Gold. Based on the 1998 amendment, the annual minimum payment was $75,000; however, until payment of a production royalty begins, the minimum annual payment due was $25,000 with the difference to be considered a deferred payment until commencement of production royalty payments. As described by Devenyns (2007), “beginning in the first lease year following the commencement of production royalty payments from the Project, the deferred payments would be paid at the rate of $75,000.00 per year from proceeds of products mined from the entirety of the Project until the total of the deferred amounts was paid. Payments of the deferred amounts were in addition to the minimum payments.” As of July 14, 2014, including the deferral of $40,000 of that year’s minimum annual payment, the current total deferral amount is $760,000. Annual mining lease payments have been timely made and the mining lease is considered to be in good standing.

The owners will receive a 5% net smelter return (NSR) production royalty with credit for one-half of the annual payment. The original term of the lease was for 10 years with the opportunity to extend it for two additional five-year periods.

A second amendment of mining lease was entered into effect July 13, 2010 which contained the following terms:

- The parties to the lease are now Sierra Denali Minerals and American Gold.

- The lease term is extended by 10 years from July 14, 2010 and may be extended for two additional five year periods, provided the Project has commenced production and continues to pay production royalty and deferred payments.
- The owner was paid $10,000.00 for signing the extension of the lease and $25,000.00 for the payment due July 14, 2010 with $50,000.00 being credited to the deferred payment balance. Note: these payments have been made.
- Beginning with the payment due July 14, 2011 and thereafter, the minimum payment of $35,000 per year with $40,000 per year being considered a deferred payment.
- Acknowledgement that through July 14, 2010, the deferred payment balance is $635,000.00, Which has since been re-calculated to be $760,000 through July 14, 2014.

Except as modified by the second amendment, the terms of the lease remain effective.

4.3.2 UNPATENTED LODE MINING CLAIMS OWNED AND LEASED BY AMERICAN GOLD

American Gold paid the federal annual mining claim maintenance fees for the annual assessment years from September 1, 2011 to September 1, 2012, September 1, 2012 to September 1, 2013, September 1, 2013, to September 1, 2014, and September 1, 2014, to September 1, 2015, and the unpatented mining claims remain and will be in good standing until September 1, 2015. American Gold has recorded in the Office of the Lyon County Recorder, the notices of intent to hold the claims in accordance with Nevada law through October 31, 2015.

4.4 ENVIRONMENTAL REPORTS AND LIABILITIES

In February 2011, Gunpoint Exploration US Ltd., a Nevada corporation and wholly-owned subsidiary of Gunpoint submitted a Notice of Intent to Conduct Exploration Activities (the Notice) to the BLM, which included certain drill sites within the resource area. The Notice was revised in April and August 2011. A reclamation bond in the amount of $15,000 was posted with the BLM with the actual committed amount of $12,479 leaving an additional $2,521 available for future bonding.

An Interim Permit for Reclamation Application was also submitted in September 2011 to the Nevada Division of Environmental Protection (NDEP) in the name of Gunpoint Exploration US Ltd. The NDEP requested that the existing reclamation permit, BMRR Permit 0070 in the name of American Gold, be revised to include the current and proposed exploration activities including those within the resource area.

In a letter dated December 16, 2011, American Gold requested that the NDEP and the BLM release the vegetation requirement and re-categorize the acreage in the BMRR Permit 0700. The BLM and the NDEP conducted a site inspection December 21, 2011 and agreed to release the re-vegetation requirement by letter dated December 29, 2011.

American Gold revised the BMRR permit in January and July 2012 and submitted it to the BLM and the NDEP. The revised permit provides for a total of 104.4 acres of disturbance; of which 88.8 acres may be on BLM land and 15.6 acres may be on private land. The current and proposed disturbance by exploration activities conducted in 2011 to 2013 totals 18.7 acres. On September 25, 2012, BLM accepted the revisions to the permit and accepted the total reclamation bond amount for 18.7 acres of disturbance at $152,568. American Gold currently has a reclamation bond in place for $152,568 posted with the BLM and no additional environmental liabilities are anticipated from past activities at the Project beyond those addressed under the reclamation cost estimate and bond. American Gold has been notified that an updated revised cost estimate will need to be submitted to the BMRR and the BLM by July 1, 2015 for approval by October 1, 2015 for another three year period.

4.5 PERMITTING

American Gold continues to maintain its water right permit by filing an annual application for extension of time to prove beneficial use. It is currently extended until January 2016 at which time another application for extension of time will be filed.

American Gold holds valid water and land use permits from the State to conduct surface exploration and drilling campaigns on the Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS

Access to the Talapoosa district from Reno is via Interstate 80, east about 30 miles to Fernley, then south on US Alternate 95 for 13 miles to Silver Springs, then west on US 50 for about two miles to Ruby Avenue, then north on an improved, gravel road for 3 to 4 miles to the approximate center of the district and the area of the resource (Figure 5.1).

An alternate but poor, unimproved road leaves US Alternate 95 at the south end of a large sweeping curve 3 miles north of Silver Springs. From the highway, it is about 3 miles west to the area of the resource. This route is not recommended when road conditions are wet or muddy. Access to the Project is available year round if required.

Reno has an international airport with numerous regional flight schedule daily. Carson City has a single 6,100 ft. landing strip while Silver Springs has a regional airport with a single 7,200 ft. military grade landing strip.

Figure 5.1 - Access Map

5.2 CLIMATE

The Project is located in a region of Nevada characterized as a high-desert environment, situated in the rain shadow of the Sierra Nevada to the west. The climate at Talapoosa is moderate and conducive to 12-month exploration or mining operations. Summers are hot and dry with temperatures commonly reaching or exceeding 90°F with the average around 78°F. Winter weather is moderate with highs of 45ºF and lows around 20°F with an average of 32°F.

Annual precipitation is estimated to be about 8 in., of which snowfall accounts for about one-third and rarely remains on the ground longer than a few days. Annual evaporation rates are estimated to be about 71 in per year (www.city-data.com).

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The Project is located about 45 miles in road distance from Reno, whose metropolitan area has a population of 225,221 (www.usa.com), and about 30 miles in road distance from Nevada's capital, Carson City, with a population of 55,274 (www.usa.com). The closest civic center to the Project is Silver Springs, located 3 to 4 miles from the Project with a population of 5,296 (www.usa.com).

All centers provide excellent sources of skilled and unskilled labor, professionals, and most services needed for a mining operation.

Commercial power lines pass through the Project. Upgrades to the electric infrastructure are likely required to advance the Project beyond the advanced exploration stage.

A water well (PW-1) with an approximate capacity of 60 to 90 gpm has been drilled on the Property, and other well sites were targeted. Arrangements for supplemental sources would have to be made with other private owners nearby. Previous engineering studies have identified suitable areas for plant and ancillary facilities and also tailings and waste disposal.

5.4 PHYSIOGRAPHY

Talapoosa lies on the eastern and southeastern flanks of the Virginia Range, one of the ranges of the Basin and Range Province. Churchill Valley lies to the east. Elevations range from 4,400 ft. at the valley floor to 6,500 ft. on the higher surrounding hills, with an elevation of about 5,300 to 5,500 ft. at the Project site. Ground elevation on the Property falls to the south.

There is sparse vegetation, which consists of desert grasses and brush. There are no perennial streams and no surface water accumulations on the Property. Ephemeral stream channels drain the area to the south and east. Drilling by various exploration companies has established that the water table occurs between 5,170 and 5,230 ft. in elevation in the vicinity of the mineralization.

6 HISTORY

Exploration of the Project dates back to 1863 with the discovery of silver mineralization on the Project by prospectors working outwards from the Comstock Lode area (Danley 1999). Table 6.1 summarizes the significant activities on the Project from the date of discovery.

Table 6.1 - Talapoosa History

Year	Company	Activity
1863	Prospectors	Silver mineralization discovered.
1905-1925	Talapoosa Mining Company	Operated several underground mines. Lost the Property in litigation.
1950	Fred de Longchamps & Sons (Longchamps)	Leased then purchased property.
1964	Great Basin Exploration	Leased property from Longchamps. Conducted trenching and geochemical sampling.
1966	Duval Corporation	Subleased property from Great Basin. Conducted underground mapping and sampling. Did not exercise option and property returned to Great Basin.
1966	Great Basin Exploration	Returned property to Longchamps.
1967-1975	Various Individuals	Mapping and sampling completed on the Property.
1977-1978	Homestake Mining Company (Homestake)	Completed regional soil sampling. Completed rock chip sampling; 86 samples on surface, 310 samples underground. Drilled eight holes totaling 2,380 ft.
1979	Superior Oil Company (Superior)	Acquired the Property. Drilled 21 holes totaling 8,620 ft.
1980-1983	Bear Creek Mining Company (Kennecott Copper Company (Kennecott))	Drilled 17 holes totaling 6,896 ft. Bottle roll leach tests performed by Dawson Metallurgical Laboratories Inc. (Dawson). Small column leach tests at Miller-Kappes Company.
1985-1989	Athena Gold Inc. (Athena)	Acquired the Property from the Longchamps. Drilled 205 RC holes totaling 52,700 ft. Drilled five core holes totaling 1,130 ft. Conducted two resource estimates on the Project. Bottle roll and column leach tests performed at Bateman Metallurgical Laboratories (Bateman). Bottle roll, column leach, and flotation tests with cyanidation of flotation concentrate performed by Minproc Engineers Inc. (Minproc). Bottle roll leach and flotation tests with cyanidation of flotation concentrate performed by McClelland Laboratories Inc. (McClelland).

Year	Company	Activity
1989-1990	Placer Dome U.S. Inc. (Placer Dome) Athena	Completed underground mapping. Drilled five core holes totaling 3,683 ft. Drilled 10 RC holes totaling 7,115 ft. Bottle roll leach and flotation tests with cyanidation of flotation concentrate performed by Golden Sunlight Mine Inc. Completed resource estimation.
1991	Placer Dome Athena	Surface mapping at 1 in:100 ft. Completed gradient induced polarization (IP) and magnetic-very low frequency (VLF) survey. Drilled 43 RC holes. Column leach test work performed by Barringer Laboratories.
1992-1993	Pegasus Gold Corp. (Pegasus) Athena	Completed 92 boreholes totaling 46,416 ft. Reviewed metallurgical work and resource estimation. Preliminary pit-slope study completed. Bulk sample collected on the Main Zone material. Column leach test performed by McClelland. Flotation tests performed by Montana Tunnels Mining Inc. Laboratory. Mineralogy performed by Pittsburgh Mineral and Environmental Technology Inc. Archeological survey completed. Botanical survey completed. Water resource study completed.
1995-1997	Talapoosa Mining Inc. Miramar Mining Corp. (Miramar))	Purchased the Property from Athena. Completed 11 core holes and 163 RC holes totaling 84,940 ft. Bio-oxidation followed by column leach, column leach, and bottle roll leach tests performed by McClelland. Column and bottle roll leach, different size reduction equipment and leach aids leach performance performed by Dawson. Gravity, column and bottle roll leach tests, different size reduction equipment and leach aids leach performance performed by Dawson. Concluded a feasibility study based on a heap leach operation. Completed botanical, hydrological and paleontological surveys. Completed three resource estimations.
1998-2002	Talapoosa Mining Inc. Newcrest Resources Inc. (Newcrest)	Newcrest joined as a joint venture partner. Completed data review, and remapped the mineralization area at a scale of 1 in:200 ft. Heavy media separation, gravity separation, flotation, gravity/flotation, bottle roll and vat leaching, gravity/vat test work performed by Oretest Metallurgical Testwork and Research. Conducted a structural analysis. Completed five core hole totaling 3,892 ft. Newcrest drops joint venture in 1999, returns the Project to Talapoosa Mining Inc. Completed two resource estimates.

Year	Company	Activity
2002	Cascade Metal US Inc.	Purchased the Project from Miramar.
2006	American Gold	Cascade Metal US Inc. changes name to American Gold.
2007	Chesapeake	Acquired American Gold and holds as a wholly-owned subsidiary.
2010	Christopher James Gold Corp.	Acquired the Project and American Gold from Chesapeake in exchange for shares in Christopher James Gold Corp. Changed name from Christopher James Gold Corp. to Gunpoint.
2010	Gunpoint	Conducted regional ground magnetics and induced polarization surveys.
2011	Gunpoint	Completed seven core holes totalling 5,302 ft.
2013	Gunpoint	Completed a resource estimation on the Property. Completed metallurgical tests on seven composite samples, including bottle roll tests on seven samples and column leach tests on 4 samples

Table 6.2 summarizes the drilling history on the Project. Further information regarding the drilling and sampling programs are described in Sections 10.0 and 11.0.

Table 6.2 - Talapoosa Drilling History from 1977 to 1999

Description	Number of Holes	Feet	Percent
Company			
Miramar	175	142,471	48
Superior	21	8,620	3
Newcrest	5	3,892	1
Pegasus	92	48,883	16
Homestake	8	2,380	1
Kennecott	17	6,896	2
Athena	210	53,621	18
Placer Dome	58	31,543	11
Grand Total	**586**	**298,305**	**100**
Drill Type			
Core	47	38,899	13
RC	494	216,761	73
Rotary	20	7,670	3
RC/Core	20	31,293	10
Rotary/Core	5	3,683	1
Grand Total	**586**	**298,305**	**100**
Year			
1977	8	2,380	1
1981	17	6,896	2
1995	135	131,041	44

Description	Number of Holes	Feet	Percent
1998	5	3,892	1
1992	16	7,966	3
1992 and 1993	23	9,545	3
1993	53	31,372	11
1996	40	11,430	4
1979	21	8,620	3
1988	126	28,160	9
1985	34	4,800	2
1988 and 1989	55	24,344	8
1989 and 1990	10	7,115	2
1990 and 1991	43	20,745	7
Grand Total	**586**	**298,305**	**100**

Table 6.3 summarizes the historical estimates completed by previous owners. Tetra Tech has not sufficiently evaluated the historic estimates described in Table 6.3 for classification as current mineral resources or mineral reserves, and the issuer is not treating the historic estimates as current mineral resources or mineral reserves as defined under NI 43-101. The historic estimates should not be relied upon.

Table 6.3 - Historical Estimate Summary from 1989 to 1999

Company	Year	Tons	Au (oz/ton)	Ag (oz/ton)	Au (oz)	Notes
Athena (MDA)	1989	19,592	0.045	0.61	881,640	Geologic Reserve
		12,723	0.045	0.656	572,535	Minable Reserve
Athena	1989	17,904	0.054	0.654	967,000	Global Geological Resource
Placer Dome	1990	20,886	0.032	0.28	668,352	Geologic Reserve
Pegasus	1989	31,680	0.022	-	696,960	Geologic Resource
Pegasus	1989	16,560	0.033	-	546,480	Geologic Reserve
Pegasus	1991	18,893	0.030	-	566,790	Minable Reserve
Pegasus	1991	24,711	0.044	-	1,087,284	Minable Reserve
Pegasus	1993	26,796	0.034	0.45	911,000	Probable Resource
Pegasus	1993	29,291	0.035	0.44	1,025,000	Probable Resource
Miramar	1996	60,000	0.025	0.37	1,500,000	In-place Reserves
Miramar	1996	28,000	0.026	0.37	726,000	Reserve
Miramar	1996	43,299	0.025	0.34	1,091,800	Geologic Resource Main Deposit
		29,625	0.027	0.4	800,000	Minable Reserve Main Deposit
		3,738	0.020	0.23	73,500	Geologic Resource East Hill Deposit
		873	0.018	0.23	15,800	Minable Reserve East Hill Deposit
Newcrest	1999	25,000	0.041	0.55	1,025,000	-
Newcrest	1999	23,300	0.039	0.34	900,000	-

The resource estimates described above have been superseded by the current resource estimate described in Section 14.0.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

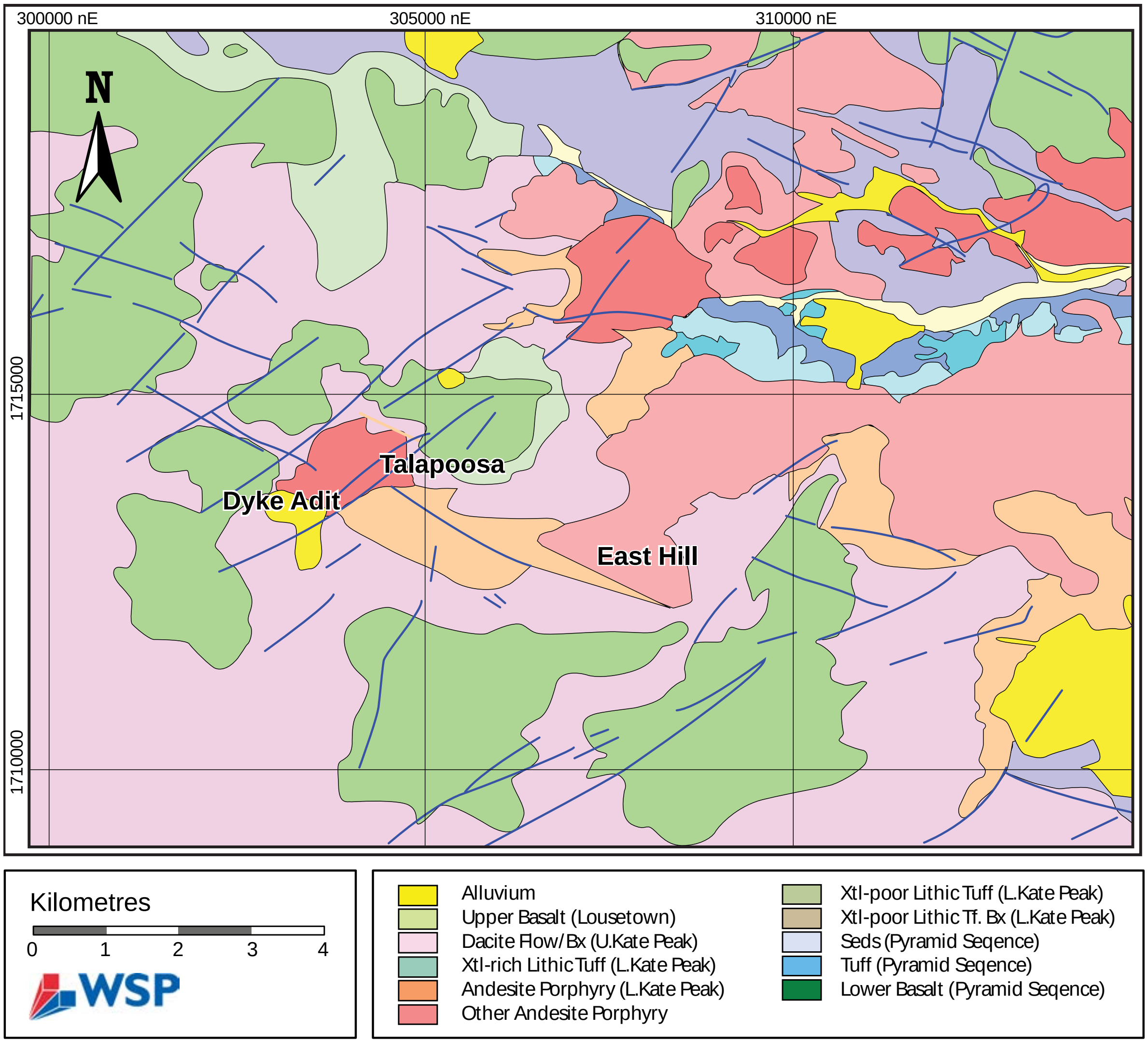
The Project lies in the western Basin and Range Province, a structural province of generally north-trending mountain ranges and intervening valleys formed by regional extension during Tertiary time. The Sierra Nevada on the California-Nevada border forms the western margin of the province. The eastern slope of the Sierra Nevada is cut by major north-trending normal faults that form north-trending mountain ranges (Moore 1969). The Virginia Range, on whose east flank the Project is located, along with the Pine Nut Mountains, Wellington Hills, and Sweetwater Range to the south, forms one of four master fault-block ranges of this type that can be considered north-trending spurs of the Sierra Nevada.

The rocks of the Sierra Nevada in this region are predominantly granitic intrusions of the Mesozoic Sierra Nevada batholith. Older Mesozoic metavolcanic and metasedimentary rocks, thought to be predominantly Late Triassic and Early Jurassic based on fossil evidence (Moore 1969), are preserved as roof pendants and septa within the batholithic intrusions.

Miocene and younger volcanic rocks overlie the Mesozoic intrusions in this part of western Nevada. Late Miocene rhyolitic tuffs with some interbedded rhyolitic lava and vesicular basalt form the base of the volcanic sequence, overlain by Miocene-Pliocene, predominantly dacitic and andesitic volcanic and related intrusive rocks with interbedded sedimentary rocks. Interbedded with and overlying the intermediate volcanic rocks throughout this region are Pliocene sedimentary rocks that were deposited by lakes and streams in isolated basins adjacent to topographic highs. Late Pliocene to Pleistocene basaltic rocks, primarily lava flows, are widespread throughout the region, and represent the youngest episode of volcanism and are post-mineralization.

Cenozoic faulting, tilting and warping associated with regional extension that resulted in the Basin and Range Province are the most recent and conspicuous structural features of the region. While the extension is manifested by a predominantly north-trending structural grain with normal faulting, in this part of western Nevada there is also the northwest-trending Walker Lane trend with oblique and strike-slip faulting and Cenozoic mineralization. The Virginia Range lies in the northern portion of the Walker Lane (Figure 7.1).

Figure 7.1 - Regional Geology



7.2 PROJECT GEOLOGY

The Project, situated within the Virginia Range, is composed of a thick sequence of Miocene-Pliocene volcanic and sedimentary rocks that overlie Mesozoic metamorphic and granite found throughout the Sierra Nevada (Figure 7.2).

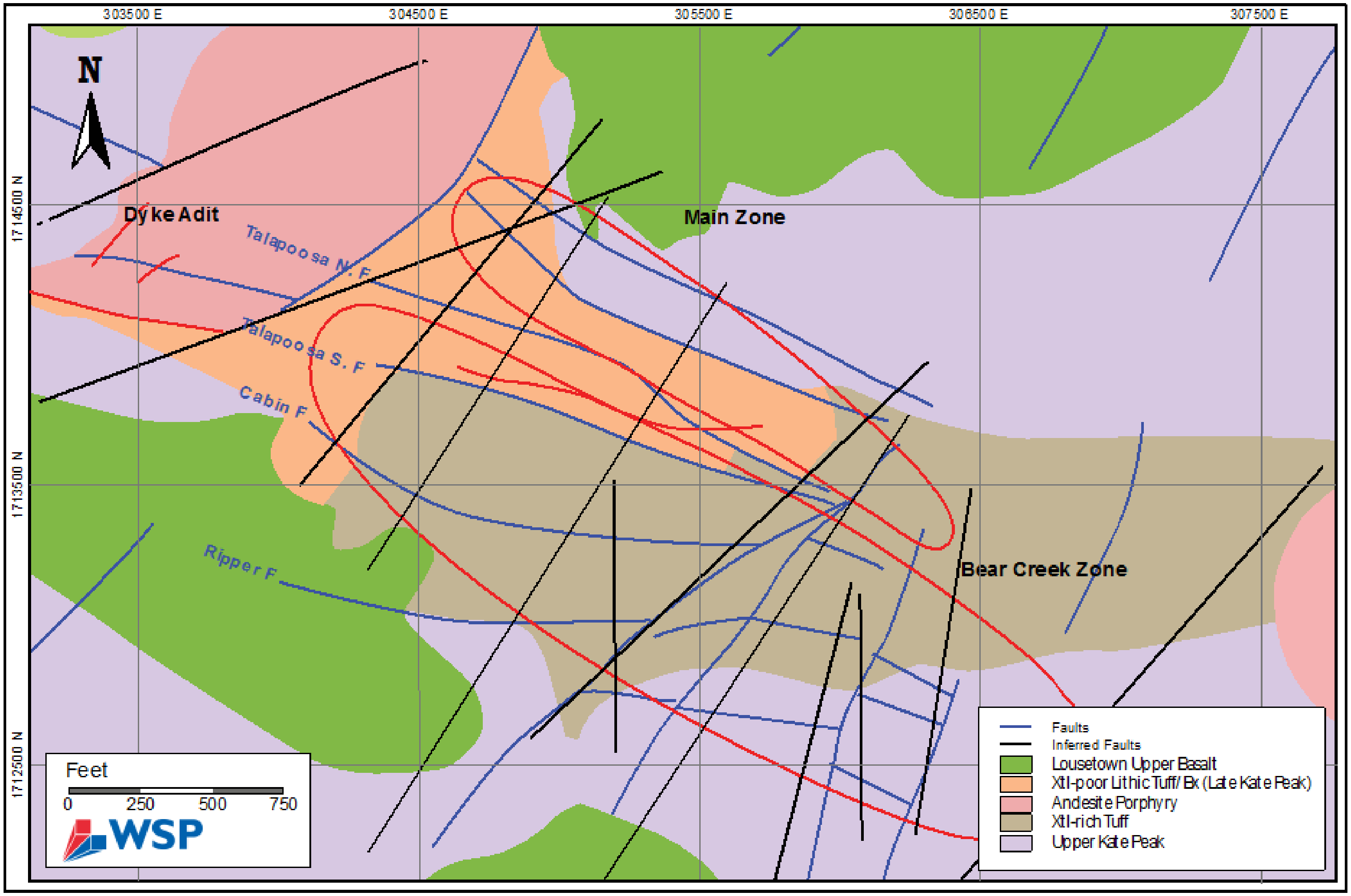
The Pyramid Sequence is the base of the geological package on the Project. It is a sequence of vesicular basalt, felsic ash-flow tuffs and hydrothermal eruption breccias associated with epithermal mineralization along the Appaloosa structure.

The Kate Peak Formation hosts all of the known mineralization in the district and overlays the Pyramid Sequence. The Kate Peak Formation consists of dacitic tuff, tuff breccia, flows, lava dome carapace debris, and post-volcanic dacite porphyry sills or dykes. The base of the formation is marked by a group of clastic sedimentary rocks that include basal volcanic conglomerate, overlain by thinly bedded shale and sandstone. The unit is estimated to be approximately 1,000 ft. thick at the Project. The formation is divided into an andesite lower member and a dacite upper member. The presence of a porous tuffaceous unit, which was silicified and then repeatedly cracked and mineralized, is referred to as the Crystal-Poor Welded Tuff. The Kate Peak Formation is described as being separated from the underlying Pyramid Sequence by the Talapoosa Fault.

The Pliocene aged Coal Creek (Canyon) Formation unconformably overlays the Kate Peak formation. It is described as a mixture of sand, silt, and clay derived from pyroclastic volcanic rocks. It is no more than a few tens of feet thick at the Project.

The Lousetown Formation, a basaltic unit ranging from a few feet thick to as much as 300 ft. in thickness, unconformably overlies the Coal Creek Formation. The unit is a vesicular olivine basalt or pyroxene andesite with flows capping the hills surrounding the Project.

Figure 7.2 - Project Geology



7.3 STRUCTURE

Throughout the Project area, the entire Kate Peak Formation and Pyramid Sequence dip gently to the south. The sequence also steps down to the south across the series of west northwest-trending faults which although predominantly post-mineral in age, do show some evidence of earlier pre- and syn-mineral movement. All fault names are taken from the historically used project nomenclature with the exception of the Mill, Middle and East Faults which were coined by Gunpoint.

Locally, sediments are more steeply dipping where they are steepened against these faults. The north northeast-trending set of faults are late syn- to post-mineral in age and are locally associated with late-stage open-spaced comb-quartz veins.

The three mineralogically and physically distinct mineral domains - the Bear Creek Hanging-Wall, Bear Creek Footwall and the Main Zone - are bounded by the north-northwest trending Ripper, Cabin, Talapoosa, and Dyke/Opal faults. Peripheral mineralization was divided into East Hill Vein/Domain to the east and Dyke Adit (North and South) Veins/Domains to the west.

7.4 ALTERATION

Alteration characteristic of epithermal precious metal deposits includes propylitic, phyllic, silicic, argillic, and opaline types, all of which are present at the Project. Propylitic alteration is usually pervasive and is characterized by chlorite, calcite and clays with local chlorite-quartz-calcite-pyrite veins crosscutting earlier pervasive propylitic alteration. Phyllic alteration, also generally pervasive, consists of sericite, quartz and pyrite with sericite dominant. Silicic alteration with multiple stages of quartz + adularia can occur in or associated with veins, stockwork, breccias or silica flooding. Argillic alteration consists primarily of montmorillonite clays, kaolinite and alunite. It can occur as a supergene product of pyrite oxidation as well as due to hypogene processes. At the Project, argillic alteration crosscuts all other types of alteration and mineralization except opaline. Opaline alteration consists predominantly of opal and chalcedony with iron oxides and occasional cinnabar and is a high-level alteration feature.

In the Talapoosa district, the silicic alteration is spatially and temporarily related to precious-metal mineralization. Silicic alteration characteristically occurs as a well-developed vein stockwork crosscutting andesite (dacite) flows but also occurs as pervasive silica flooding. In addition, there are irregular zones of hydrothermal breccias and large vein breccias up to 30 ft. wide. Structural controls are very important at Talapoosa.

7.5 MINERALIZATION

The mineralization was divided into the following domains, separated by north-northwest fault, for the purpose of resource modelling.

- Bear Creek Hanging-Wall Vein System/Domain, bounded by Ripper Fault to south and Cabin Fault to north. The Hanging-Wall vein is comprised predominantly of massive white sulphide poor silica with typical low-sulfidation epithermal textures, including recrystallization, coliform and crustiform banding, adularia bands, amethyst, etc.
- Bear Creek Footwall Vein System/Domain, bounded by Cabin Fault to south and Talapoosa (South) Fault to the north. The Footwall vein is more sulphide rich, associated with a number of gangue phases including, red hematitic silica, chlorite and minor white to clear silica.

- Main Zone Vein System/Domain bounded by Talapoosa (South) Fault to the south and Opal/Dyke Fault to the north.

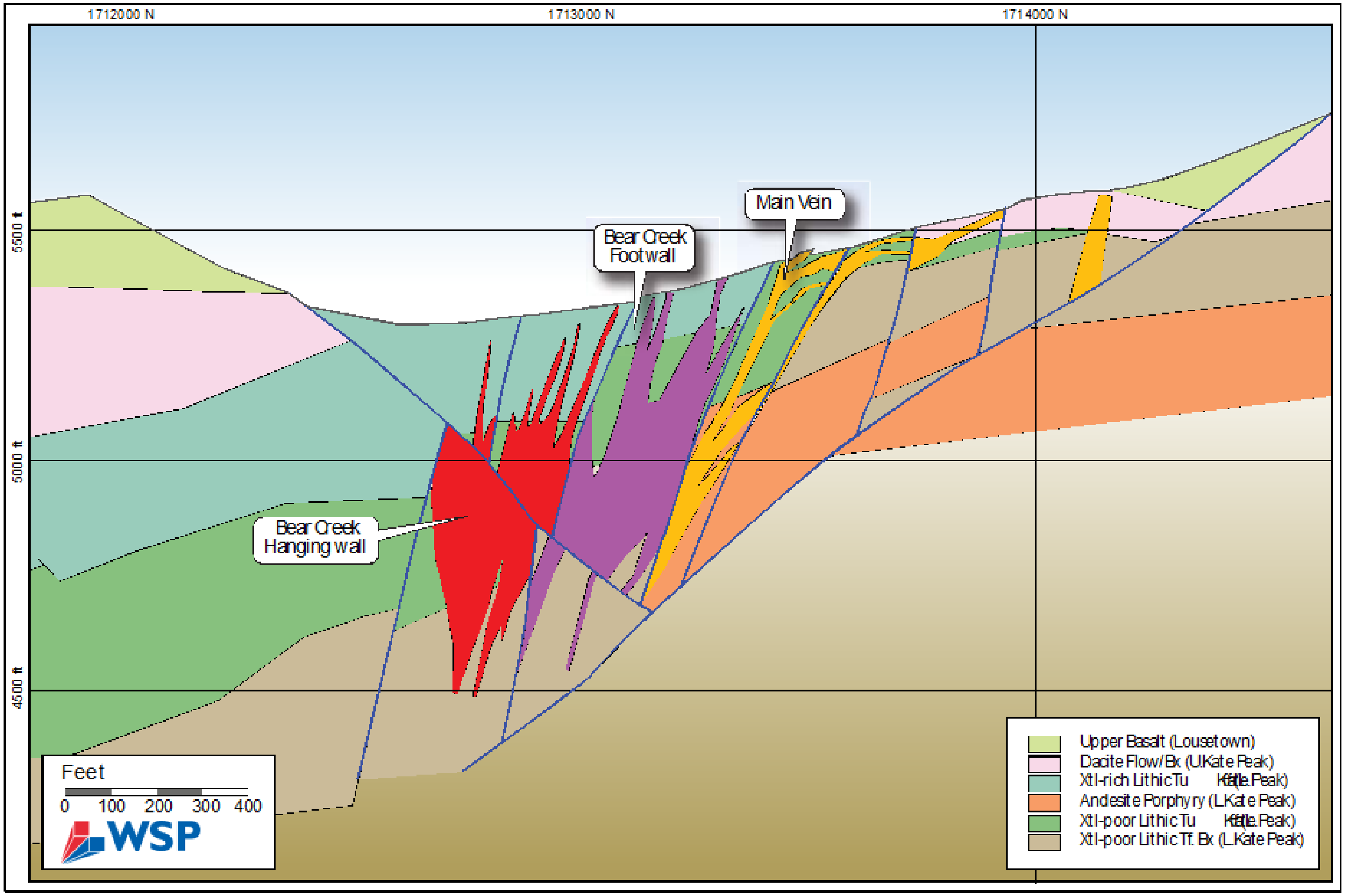
The mineralization at both Dyke Adit and East Hill shows similarities in appearance and texture to that of the Hanging-Wall Zone at Bear Creek.

The modelling of veins and their bounding faults indicates that the general trend of all mineralization is around 115°, with two prominent dip angles:

- Steeply-dipping veins at approximately 70° south, for the Hanging-Wall and Footwall Zones at Bear Creek and for the eastern-most portion of the Main Zone.
- Shallowly-dipping veins, at approximately 20 to 40° south for the Dyke Adit, northwest part of the Main Zone (north) and the East Hill Vein. At least in the Main Zone, the flattening of vein dip could be the result of dilatational zones developed between the Talapoosa and Dyke Faults. In the case of the Dyke Adit and East Hill veins the attitude of the veining appears to parallel that of the contact between the hornblende andesite porphyry and the adjacent unit.

Figure 7.3 is a generalized geological section on the Project to demonstrate the orientation of the mineralization and the complexity of the fault structures.

Figure 7.3 - Geological Cross-Section



8 DEPOSIT TYPES

8.1 LOW SULPHIDATION EPITHERMAL

Low-sulphidation epithermal deposits are precious metal-bearing quartz veins, stockworks, and breccias which formed from boiling of volcanic-related hydrothermal systems (Figure 8.1) as summarized in the US Geological Survey (USGS) deposit model 25c (http://pubs.usgs.gov/bul/b1693/html/bullfrms.htm).

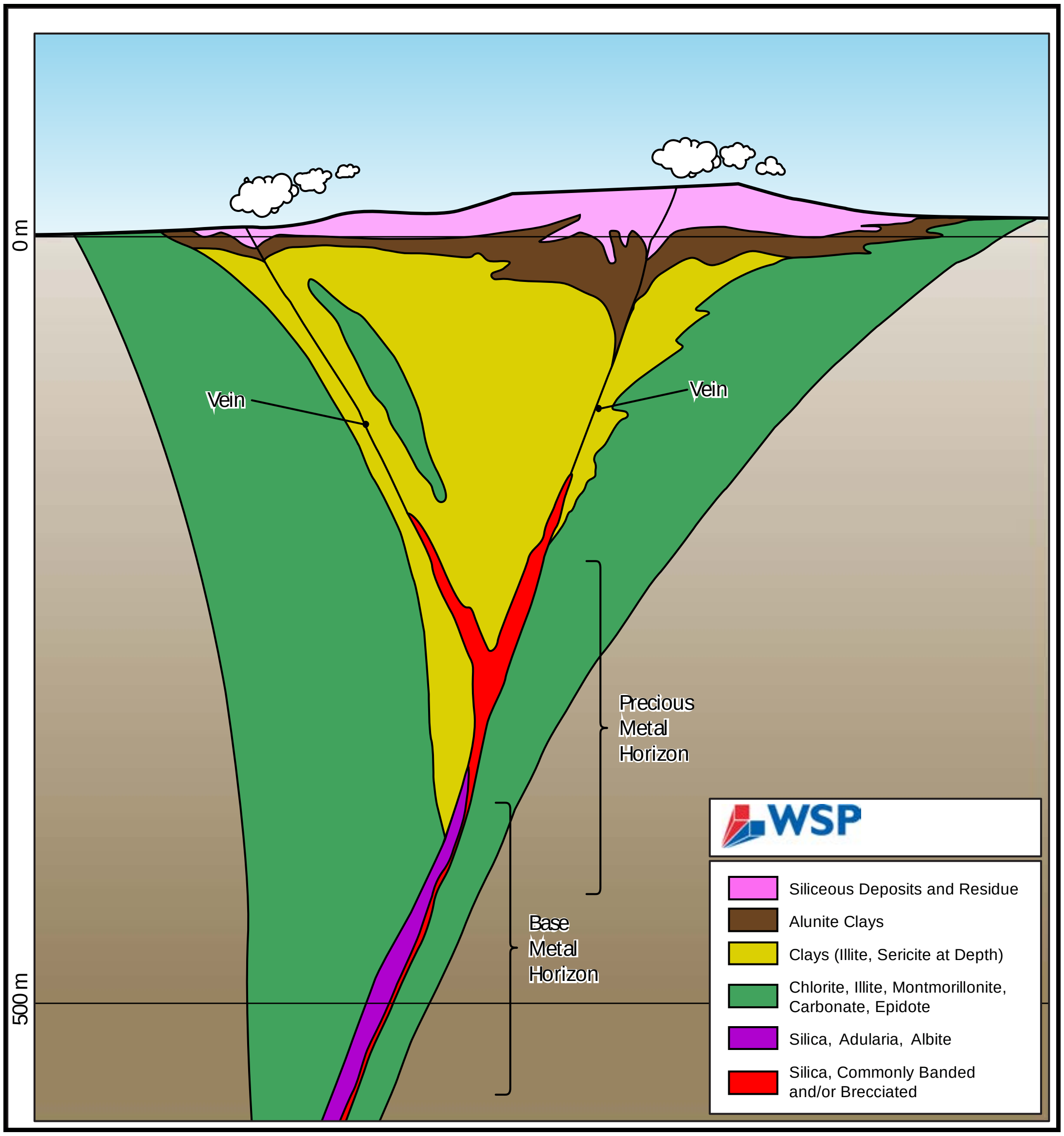
Emplacement of mineralization is generally restricted to within 1 km of the paleosurface (Panteleyev 1996). Veins typically have strike lengths in the range of hundreds to thousands of metres; productive vertical extent is seldom more than a few hundred metres. Vein widths vary from a few centimetres to metres or tens of metres.

Gangue mineralogy is dominated by quartz and/or chalcedony, accompanied by lesser and variable amounts of adularia, calcite, pyrite, illite, chlorite, and rhodochrosite.

Vein mineralogy is characterized by gold, silver, electrum and argentite with variable amounts of pyrite, sphalerite, chalcopyrite, galena, tellurides, rare tetrahedrite and sulphosalt minerals. Crustiform banded quartz veining is common, typically with interbanded layers of sulphide minerals, adularia and/or illite.

Regional structural control is important in localization of low sulphidation epithermal deposits. Higher grades are commonly found in dilational zones, in faults, at flexures, splays and in cymoid loops.

Figure 8.1 - Epithermal Model



9 EXPLORATION

Timberline has not conducted any surface exploration on the Property.

9.1 GUNPOINT EXPLORATION

Gunpoint conducted an intensive regional exploration program of the Talapoosa Appaloosa Tenement Area in 2010, of ground magnetics and IP, centered on the Talapoosa resource area.

The acquisition and interpretation of ground magnetic and IP data was supervised by Ellis Geophysical Consulting Inc., of Reno, Nevada in 2010. The ground magnetic survey was undertaken on 50 m line spacing and covered the entire Talapoosa resource area and beyond. A total of seven IP lines covered the Talapoosa resource area and beyond, orientated north-south and spaced roughly 150 m apart.

10 DRILLING

10.1 PRIOR OWNERS

Prior to Timberline's involvement on the Project, nine companies are known to have drilled at the Property (Table 10.1) (Ristorcelli, et al. 2010, McCracken, 2013). Section 6.0 summarizes when drilling was completed by the various companies.

Table 10.1 lists the companies, drilling type, and year of drilling. Over 73% of the drilling database is RC drilling. Over 13% of the drilling database is core drilling.

A majority of the drilling at the Property was oriented vertically due to the volume of RC drilling conducted. This means that a large portion of the drill results are subparallel to the high-grade vein orientation and displace grade intervals that do not represent the true thickness of the mineralization. A small portion of inclined holes were drilling primarily perpendicular to the mineralization and thus the drilled thicknesses of mineralization would closely approximate true thicknesses.

Table 10.1 - Talapoosa Historical Drilling Summary (1977 to 1999)

	Number of Holes	Feet	Percent
Company			
Miramar	175	142,471	45
Superior	21	8,620	2
Newcrest	5	3,892	1
Pegasus	92	48,883	15
Homestake	8	2,380	1
Kennecott	17	6,896	2
Athena	210	53,621	17
Placer Dome	58	31,543	10
Gunpoint	7	17,396	5
Grand Total	**593**	**315,701**	**100**
Drill Type			
Core	54	56,295	18
RC	494	216,761	69
Rotary	20	7,670	2
RC/Core	20	31,293	10
Rotary/Core	5	3,683	1
Grand Total	**593**	**315,701**	**100**
Year			
1977	8	2,380	1
1979	21	8,620	3
1981	17	6,896	2

	Number of Holes	Feet	Percent
1985	34	4,800	1
1988	126	28,160	9
1988 and 1989	55	24,344	7
1989 and 1990	10	7,115	3
1990 and 1991	43	20,745	6
1992	16	7,966	3
1992 and 1993	23	9,545	3
1993	53	31,372	10
1995	135	131,041	42
1996	40	11,430	4
1998	5	3,892	1
2011	7	17,396	5
Grand Total	**593**	**315,701**	**100**

10.1.1 HOMESTAKE

The following information is from a Homestake report by Thomssen (1978).

Homestake drilled eight vertical core holes for a total of 2,380 ft from November 17, 1977 through January 30, 1978. The borehole series used was T-001 to T-008. There were samples for 2,312 ft. Drilling was completed using Boyles Brothers Drilling as the drill contractor. The drilling was located in the approximate center of the Talapoosa district in the vicinity of the Dyke Adit, Christiansen Shaft, and Glory Hole.

Of the total footage drilled, 68 ft. were done with a rock bit with no samples recovered. Another 63 ft. were drilled with a core drill producing NX core. A total of 2,249 ft. were cored with NC core. Depth of the holes ranged from 118 to 525 ft. Core recovery averaged about 90%.

10.1.2 SUPERIOR

The following information is from compilations by Athena (Van Nieuwenhuyse 1989) and Newcrest (Danley 1999a).

Superior drilled 20 vertical, large-diameter, percussion rotary holes (DH1-DH20) totalling 7,670 ft. and one vertical core hole (SS-21) to a depth of 950 ft. from 1978 to 1979. The core was NC size.

The rotary holes were collared around East Hill. The one core hole was drilled in the Bear Creek Zone and at 950 ft. is still the deepest hole drilled on the Property.

10.1.3 KENNECOTT

The following information is from compilations by Athena (Van Nieuwenhuyse 1989) and Newcrest (Danley 1999a).

Kennecott drilled 17 vertical NC core holes totaling 6,896 ft. on the Property. Borehole series was TA-001 to TA-017. The holes were distributed from Dyke Adit to East Hill.

10.1.4 ATHENA

The following information is taken from Van Nieuwenhuyse (1989) and Athena (1991).

A total of thirty four RC holes totaling 4,800 ft. were completed in 1985 (TRC-001 – TRC-034). Allen Drilling as the contractor. In 1988, 121 RC holes were completed (TAL-001 – TAL-121). The drill contracted used was Delong Drilling. Drilling totaled 24,452ft, according to Van Nieuwenhuyse (1989.

In 1989, Athena drilled 50 RC holes (TAL-122 – TAL-171) that totaled 23,448 ft. (Van Nieuwenhuyse 1989), using Drilling Services as the contractor. An additional five NC core holes (TC-001 – TC-005) were completed in 1989 for a total of 1,130.5 ft. No records of the drill contract name were available.

10.1.5 PLACER DOME

The following information is taken from Placer (1990), Athena (1991), and Danley (1999a).

During Placer's initial evaluation of Talapoosa from December 1989 through February 1990, five HX core holes (TC-006 – TC-010) and 10 RC holes (TAL-172 – TAL-181) were drilled. In 1990-1991, an additional 43 RC (TAL-182 – TAL-204; TAL-204A; TAL-205 – TAL-223) were completed.

The five core holes totaling 3,683ft,. were started with rotary drilling, followed by coring to the final depth. Boyles Brothers Drilling Company drilled all five holes using a Longyear 44 and a BD30. The core was logged for geology, recovery, and RQD and was then photographed.

The initial 10 RC holes (TAL-172 – TAL-181) included six vertical and four angle RC rotary holes totaling 7,115ft. Drilling Services drilled the vertical holes using a TH-60 rig, and Hackworth drilled the inclined holes using a CP-700 rig. A down- hole hammer was used for drilling above and immediately below the water table, then a tricone bit was used when large volumes of water were encountered. Both bits were 5¼ inches in diameter. Drill chips were collected for geology in plastic vials, and chip boards were constructed. Cuttings were logged on site by a Placer geologist and later reclogged with a binocular microscope.

For the remaining 43 RC holes (TAL-182 – TAL-204; TAL-204A; TAL-205 – TAL-223), Placer used Hackworth.

10.1.6 PEGASUS

The following information is taken from Longo (1992), Pegasus (1992, 1993, 1994), and Danley (1999a).

In 1992, sixteen holes were drilled (PM series), of which eight were core holes drilled for metallurgical testing. One additional core hole and five rotary holes were drilled for exploration purposes. The drilling totaled 2,270ft RC and 3,429 ft of HQ core (2.5 in). Core recoveries averaged close to 95%. Drilling was completed by Hackworth Drilling for RC holes and Allcore Drilling and Coates Drilling for the core holes.

In 1992-93, Pegasus drilled 9,545ft of RC drilling in 23 holes and 2,267ft of HQ core in five holes. The five core holes were pre-collared with RC drilling. Core recovery in these five holes averaged close to 97%. Boyles Brothers drilled the five core holes, and Hackworth Drilling drilled the RC holes.

Later in 1993, Pegasus completed drilling of 52 additional holes for a total of 27,072ft of RC drilling and 1,848ft of HQ core drilling. Holes PE33-PE36 and PE38-PE81, including PE80A, were RC holes. Holes PE30-PE32 were drilled with RC to the water table and then completed with core. Hole PE37 was a core hole. For this program, Hackworth Drilling was used for the RC drilling, and Boyles Brothers did the core drilling. Core recoveries averaged about 94.6%.

RC drilling methods changed during this last program from a conventional hammer to a center-face return hammer in order to improve sample recovery.

10.1.7 MIRAMAR

The following information is taken from reports by Fluor Daniel Wright (1996a; 1996b) with additional information provided by American Gold**.**

Miramar drilled 174 holes for a total of 84,940.8ft. They drilled TAL-224 through TAL-331 and TC-11 through TC-22 for geology, geotechnical data, and metallurgy. Holes CON-1 through CON-48 were drilled for condemnation, but CON-35 was renamed MON-1. MON-1 through MON-7 were monitoring wells. Hole TAL-273 was subsequently widened and deepened by 10 feet to use as a water well; it was renamed PW-1. PW-1 is not counted as a separate hole, and the additional 10 feet are not included in the database count of holes and footage.

The results from the condemnation drilling were mixed but generally did not encounter sufficient mineralization to cause re-planning of the project except for some significant mineralization encountered in the planned waste dump areas which will require further investigation.

10.1.8 NEWCREST

The following information is taken from Danley (1999a).

Newcrest drilled five PQ (3.35 in) core holes for a total of 3,892.2ft (NCTAL-1 – NCTAL-5). Boart Longyear was the drilling contractor. Hole NCTAL-5 was reduced to BQ (1.43 in) from 652 to 901ft because of caving problems.

Newcrest holes were gyroscopically surveyed by Wellbore Navigation. When practical, clay impressions were taken to orient the core for structural information. The core was photographed and logged for lithology, alteration, mineralization, and structure. Structural elements were recorded and preserved in a database.

10.2 GUNPOINT EXPLORATION LTD.

10.2.1 TALAPOOSA

Seven diamond drillholes were completed on two fences to drill through the Bear Creek Zone in late 2011. The purpose of the program was to determine the significance of the nugget effect on historic drill data, and to confirm the re-interpretation of mineralization as being steeply dipping vein zones. As part of the program, drill core was orientated and numerous measurements made on the orientation of structures and vein mineralization.

Table 10.2 summarizes the drill collar information, while Table 10.3 summarizes the significant results from this drilling program. Figure 10.1 highlights the location of the Gunpoint drilling program relative to the historical drilling and the mineral resource.

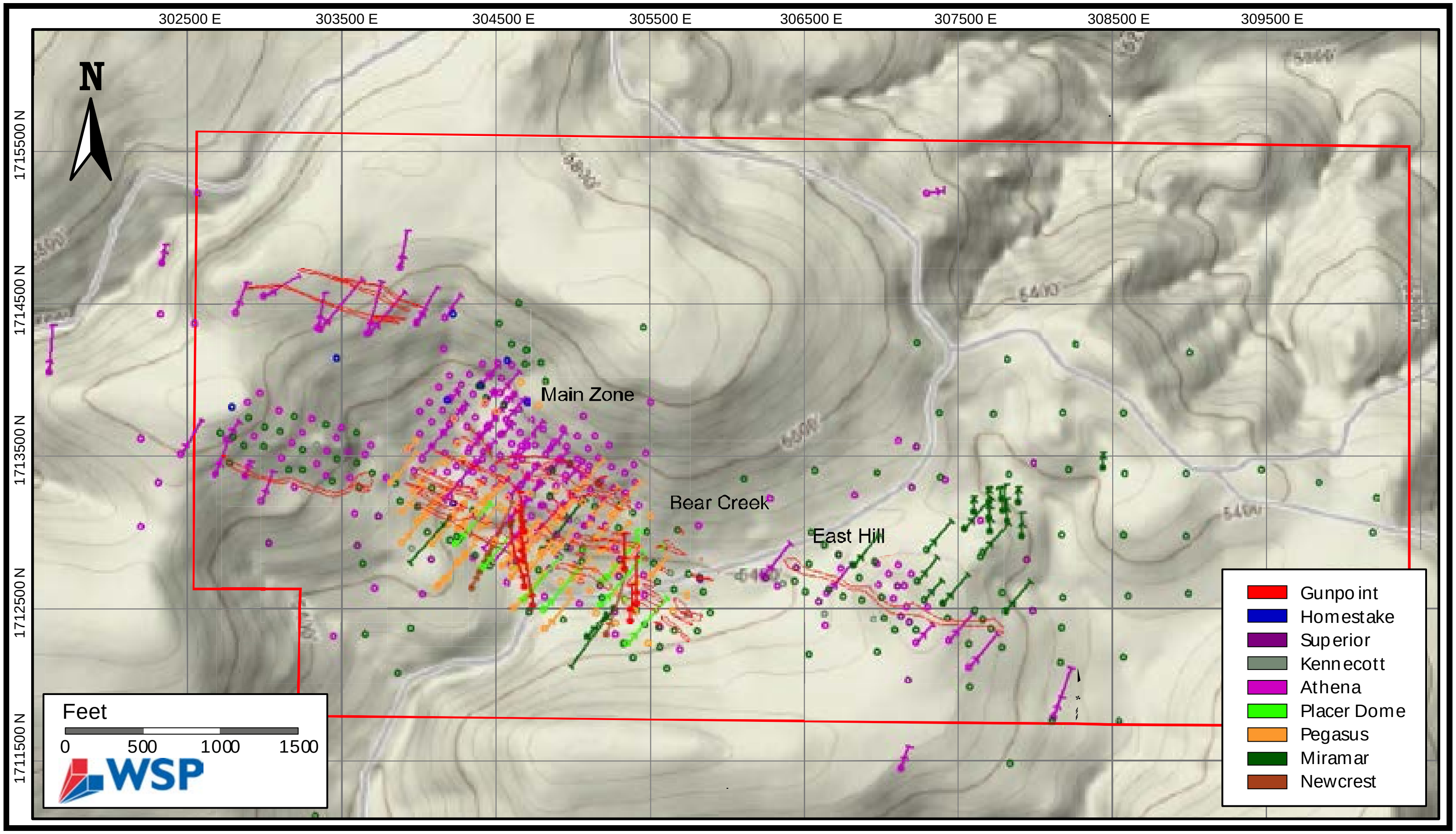
Table 10.2 - Gunpoint Drilling Collar Summary

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Length (m)
GTI-001	304756.20	1712505.31	5336.55	355.00	-60.00	605.50
GTI-002	304699.99	1712650.01	5301.60	355.00	-61.00	749.00
GTI-003	304693.09	1712819.54	5307.11	349.00	-55.00	776.00
GTI-004	304694.87	1713008.11	5336.00	351.00	-60.00	900.00
GTI-005	305420.38	1712538.45	5371.64	1.00	-60.00	698.00
GTI-006	305342.25	1712767.24	5342.92	355.00	-60.00	730.00
GTI-007	305376.46	1712251.15	5460.41	355.00	-60.00	844.00

Table 10.3 - Gunpoint Drill Results Summary

Hole ID	From (ft)	To (ft)	Interval (ft)	Au (oz/ton)	Ag (oz/ton)	Au (g/t)	Ag (g/t)
GTI-001	382	733	351	0.036	0.251	1.24	8.59
GTI-002	332	530	198	0.034	0.557	1.15	19.11
GTI-003	380	592	212	0.040	0.445	1.38	15.27
GTI-004	251	363	112	0.032	-	1.10	-
GTI-004	399	497	98	0.032	0.486	1.11	16.67
GTI-005	257	524	267	0.035	0.661	1.21	22.66
GTI-006	299	454	155	0.027	0.374	0.94	12.83
GTI-007	356	745	389	0.041	0.535	1.42	18.34

Figure 10.1 - Drill Collar Location



The 2011 drilling program was completed by Timberline Drilling Ltd. of Elko, Nevada. Coring was done with a UDR-1 track mounted diamond drill (Figure 10.2) which cored PQ (3.27 in diameter) sized holes.

Figure 10.2 - Diamond Drill Rig on the Talapoosa Project

Drilling was completed with two shifts working 12 hours.

10.2.2 SURVEYING

COLLAR SURVEY

Gunpoint surveyed the diamond drill collars using a Trimble handheld global positioning system (GPS). The final coordinates for the collars were based on the average of five separate reading at each collar location. Although each individual reading could have an error of 2 or 3 m, the average of the reading will help reduce this error margin slightly.

DOWNHOLE SURVEY

Downhole surveys were completed at 50 ft., 100 ft., and then at 100 ft. intervals to the bottom of the hole. The surveys were conducted by the drilling contractor using a Reflex ACT II. The ACT II system is used to provide downhole orientation as well as core orientation.

10.2.3 CORE DELIVERY

Core is placed in wax cardboard boxes and stacked on wooden pallets close to the drill rig by the drilling contractor. The core is collected daily by a Gunpoint employee and taken by pick-up truck to the secure core logging facility at the Sayeret Training Facility located approximately two miles from the drilling site. Access to the core logging facility is limited to Gunpoint employees or designates.

10.2.4 CORE LOGGING

The following steps are completed during the core logging process:

- Core is unloaded from trucks and placed on core logging tables (Figure 10.3).
- Run markers and other marker blocks are checked for accuracy.
- Core box labels are verified with hole ID, box number and core interval.
- Geotechnical logging is completed by logger, including the collection recovery data and rock quality designation (RQD).
- Groups of four boxes are photographed (Figure 10.4).
- Geologist log core on a paper logging sheets documenting, lithology, structure, alteration and sample intervals (Figure 10.5).
- Core orientations are measured using a wooden core orientation stand (Figure 10.6).

Figure 10.3 - Core Logging Facility

Figure 10.4 - Core Photo Station

Figure 10.5 - Logging Form

Figure 10.6 - Core Orientation Stand

10.3 QP'S OPINION

It is WSP's opinion that the drilling and logging procedures put in place by Gunpoint meet acceptable industry standards and that the information can be used for geological and resource modelling.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 PRIOR OWNERS

The available information on sampling methods, sample preparation, and analytical procedures used by past operators is derived from previous operators' work.

11.1.1 CORE SAMPLING

HOMESTAKE

Core was split with samples ranging from 1 to 7 ft. in length with the average 4.3 ft.

SUPERIOR

For the rotary drill program, sampling consisted of a continuous collection of cuttings through a cyclone or straight from a tube issuing from the casing. Sampling was not begun until about 10 or 20 ft. below the surface and was conducted almost entirely on 10 ft. intervals. Samples weighed from about 4 to 18 lb., depending on the degree of moisture in the sample.

Superior split their NC core, but no other details are known.

KENNECOTT

Kennecott split their NC core, but no other details are known.

ATHENA

For the 34 RC holes Athena drilled in 1985, samples averaged 8 lb. Wet and dry samples were split from ¼ to ⅛ split using a Jones Riffle splitter. The sample was then split again and bagged into two samples, one of which was sent for assaying and the other kept in storage for later metallurgical testing.

For the 121 RC holes drilled in 1988, samples averaged 8 lb. Dry samples were split from ¼ to ⅛ split using a Jones Riffle splitter. Wet samples were split in the same proportions using a rotary wet splitter. Both the dry and wet samples were then split again and bagged separately, one for assay and one for later testing.

In 1989, Athena drilled 50 more RC holes, using three different sampling methods. Dry samples, generally to a depth of 200 ft., were collected in the cyclone and dropped through the Jones Riffle splitter every 5 ft. (¼ to ⅛ split). When drilling wet by injecting water at water flow rates of 10 to 25 gpm, the sample was collected using a rotary wet splitter for ¼ to ⅛ splits. When drilling with large amounts of water (flow rates 50 to 100 gpm and up to 150 gpm), a desilter was used to collect the sample. A ⅛ sample split for a +10 mesh and a ⅛ sample split for the -10 mesh were collected. The samples were then dried at 110°F and weighed at the assay lab; the two samples averaged 20 lb combined.

No information on the sampling procedures for the five NC core holes drilled by Athena was available.

PLACER DOME

Placer Dome did not split or saw the drill core from the five HX holes for sampling (whole core sample).

For the RC program, the following procedures were in place.

- For all dry drilling intervals, a ¼ split of the chips returned from each 5 ft. drill increment was collected for assay.
- For inclined RC holes beneath the water table in, Hackworth Drilling collected a ¼ split from each 5 ft. interval using a rotary wet splitter.
- In vertical RC holes beneath the water table, Drilling Services circulated the drill cuttings and subsurface water through a desilter, extracted a coarse and fine fraction from the slurry, and usually retained a ¼ split of each size fraction for assay. The splits were assayed separately. Half splits were collected by drill contractor when sample recoveries were reduced. The entire sample splits were sent for assay.

Sample recovery in the core holes averaged 90%. Sample recovery for the first 10 RC holes averaged 64%. RC recoveries were calculated by weighing the dried sample and normalizing to 120 lb as 100% return for a 5 ft. interval.

PEGASUS

There is no description of the sampling procedures used by Pegasus.

MIRAMAR

There is no description of the sampling procedures used by Miramar.

NEWCREST

Newcrest chose PQ-(3.35 in) core in order to provide material for assay (¼ core), for metallurgical testing (½ core), and preserve ¼ as reference. Where practical, the core was quartered for assay, but when extreme shearing, fracturing and breaking made it dubious that the core could be quartered with integrity, the full core was submitted for assay. Full core was also sent for assay where sawing was too difficult as in portions of the massive quartz veins. When full core was sent for assay, representative specimen core was archived.

11.1.2 SAMPLE PREPARATION, ANALYTICAL PROCEDURES, AND SECURITY

HOMESTAKE

There are no records about sample preparation or security for the diamond drill program (Thomssen 1978).

The initial samples were sent to Hunter Mining Laboratory (Hunter Mining) in Sparks, Nevada, for assaying. When turn-around time became an issue, Homestake switched to Union Assay Office (Union Assay) in Salt Lake City, Utah, for the remaining assaying. No significant differences between results from the two labs were noted. A total of 556 fire assays for gold and silver were received, out of which duplicate and triplicate fire assays were run on 70 samples with an additional seven run by atomic absorption. Nine samples were also analyzed for lead, zinc and sulphur.

The detection limits for gold and silver for both the Hunter Mining and Union Assay labs were 0.001 and 0.1 oz/ton respectively.

SUPERIOR

There are no records about sample preparation or security for the diamond drill and rotary drill programs.

The rotary samples were analyzed for gold and silver using fire assay. Danley (1999a) proposed that the assays from the rotary holes should be considered highly suspect because it appeared that the laboratory Superior used had a high detection limit.

The core samples were sent to GD Resources for fire assaying, which had detection limits for gold and silver of 0.003 and 0.03 oz/ton respectively.

KENNECOTT

There are no records about sample preparation or security for the diamond drill program.

The samples from the first 11 core holes sent to Hunter Mining and Shasta labs for fire assay. The samples for the final six holes were sent to Shasta. The detection limits at Hunter Mining were 0.001 and 0.03 oz/ton for gold and silver, respectively, while at Shasta, were 0.001 and 0.01 oz/ton (Danley 1999a).

Van Nieuwenhuyse (1989) reported that Kennecott encountered discrepancies when comparing duplicate fire assays on sample splits. Some large discrepancies were noted between metallurgical calculated head grades and the original composite grades. The issue was investigated by Kennecott and resolved.

ATHENA

There are no records regarding security during the drill program.

For its 1985 RC drill program of 34 holes, samples to ALS Chemex laboratory in Sparks, Nevada, where all samples were analyzed using a then standard 10 g sample for fire assay with an atomic absorption finish. No information is available on sample preparation. The detection limit was 5 ppb for gold and 0.2 ppm for silver.

For its 1988 drill program of 121 RC holes, sample preparation was completed at an in- house preparation facility. The assay sample was crushed to -10 mesh, from which 750 g were split and pulverized using a disk pulverizer. The pulps were then taken to GD Resources for assay. All samples were assayed for gold and silver using a 50 g gravimetric fire assay. Detection limits were 0.001 oz Au/ton and 0.015 oz Ag/ton.

For its 50 RC holes in 1989, the in-house preparation facility was used and samples sent to GD Resources for assay. All samples were assayed for gold and silver using a 50 g gravimetric fire assay. For those samples collected with a desilter, the sample was initially assayed the +10 and -10 mesh fraction separately and calculated a weighted average for the interval. After not seeing any consistent relationship of assay results to size fractions, the samples later recombined the two fractions and homogenized them in the laboratory. The sample was then split in half with a riffle splitter. One half was pulped in its entirety using an impact mill (Lynx Pulverizer). Approximately 250 g were separated from the pulp to be used as an assay pulp.

Although Athena had conducted routine spot check sampling with check assays on pulps showing good consistency, during metallurgical testing it was noted that calculated head grades were consistently higher than the estimated composite grades. Studies indicated that a large sample volume and a metallic screen assay procedure provided a more representative result.

PLACER DOME

For five core holes, the entire core was sampled in three to ten foot intervals as defined by the geologist. Samples were sent to Bondar Clegg & Company Ltd. (Bondar Clegg) for sample preparation and assaying (Placer 1990). For intervals greater than ten feet that returned assay results greater than 0.02 oz Au/ton, were re-run. The 1,200 g splits from the -50 mesh reject were pulverized, and metallic sieve analyses were completed by ALS Chemex. Rejects from the -10 mesh fraction were sent to the Golden Sunlight mine for metallurgical testing.

Samples from Placer's first 10 RC holes were sent to Bondar Clegg for analysis (Placer 1990). The entire ¼ or ½ split was sent for assay. For the holes drilled by Drilling Services in which a desilter was used for samples from below the water table, the splits were assayed separately. As with Placer's core samples, for intervals with significant mineralization, 1,200 g splits from the -50 mesh reject were pulverized, and metallic sieve analyses were completed by ALS Chemex, according to Placer (1990).

During Placer's drill program at Talapoosa following their initial core and 10 RC holes, they used a sample preparation style modified from recommendations from Pitard (Placer 1990; Athena 1991).

The sample collected at the drill rig was dried at 130°F, weighed and crushed to -10 mesh. A ¼ split weighing at least 2.5 kg was ground to -40 mesh. From that, a 25 kg split was taken and ground to -100 mesh, from which a 30 g sample was taken for one- assay-ton fire assay. From the -40 mesh reject, a 1,200 g sample was split out, ground to -150 mesh and assayed by metallic sieve, if warranted; metallic sieve assays replaced fire assays, if performed.

PEGASUS

There are no records about sample preparation or security for the diamond drill and RC drill programs.

In the first two drilling phases, all RC samples as well as the core samples from hole PE-001 were sent to Barringer Labs in Reno for gold and silver assays. Barringer used a two- assay-ton fire assay method. All other core samples during this time were sent to American Assay Laboratories (AAL). Samples at AAL were analyzed for gold and silver by fire assay. McClelland's labs was used for metallurgical testing on core samples.

During the third phase of drilling, Bondar Clegg was used to prepare and assay the drill samples including all RC samples and all core samples not sent to McClelland labs for metallurgical testing. For the metallurgical samples from Phase III, core interval fire assays were completed at AAL.

During the Phase III drilling, Pegasus initiated two separate check assay programs. One tested "keeper" sample check assays for variability between two separate labs. AAL ran the "keeper" sample check assays to compare with original assays by Bondar Clegg. The gold assay comparisons between the two different labs showed the most variability, with the silver assay comparisons showing better correlation (Pegasus 1994).

The second program ran check assays on pulps at Bondar Clegg, the original lab. Random pulps were outlined, re-numbered and re-submitted to Bondar Clegg for assay. The pulp check assays correlated relatively well with the original assays, although there were variations (Pegasus 1994).

MIRAMAR

There are no records about sample preparation or security for the diamond drill and RC drill programs.

The primary assay lab used by Miramar was AAL, whose detection limits for gold and silver were 0.001 oz/ton and 0.02 oz/ton, respectively. Miramar also sent check samples to Barringer and Cone laboratories. About 10% of the delineation RC drilling samples were sent to Barringer for check analysis. Miramar concluded that overall the check assays compared with the original assays from AAL.

NEWCREST

There are no records about sample preparation or security for the diamond drill program.

Samples from the core drilling were submitted to ALS Chemex labs in Sparks, Nevada, for assay. A total of 753 core samples were assayed on even 5 ft. intervals, of which 594 samples were analyzed by metallic screen of nominal 1,300 g pulps with fire assay. The remaining 159 samples, generally barren rock, were analyzed by standard fire assay using a one-assay-ton (30 g) pulp. Both metallic screen and standard fire assays were run on 18 duplicate intervals. Two gram pulps were digested in aqua regia and analyzed by atomic absorption for silver.

Check metallic screen assays were run by Bondar Clegg on 31 samples whose assays from Chemex had ranged from 0.029 to 0.687 oz Au/ton. New nominal 1,200 g pulps were prepared from the -10 mesh rejects. Based on this limited population, the checks appeared to be acceptable, and there was no significant bias.

Newcrest implemented a quality control program to monitor sample preparation, precision, and accuracy at ALS Chemex labs. Control samples were inserted with each batch of samples at a frequency of 1 per 15 samples. A barren sample was used to monitor sample preparation and verify that there was no contamination between samples. Pulps with known values were inserted as controls. Rejects from earlier holes were re-submitted to verify accuracy and precision.

Metallurgical testing on 11 core samples was conducted by Oretest Labs of Perth, Western Australia.

11.2 GUNPOINT EXPLORATION LTD.

11.2.1 CORE SAMPLING

The following steps summarize the procedures Gunpoint had in place during the core sampling program in 2010 to 2011:

- Core was cut in half using a portable core saw. Water for the saw was recycled from a decanted pail (Figure 11.1).
- Both pieces of cut core were returned to the core box.
- Samples were collected from between run markers unless noticeable changes in alteration, structure or lithology was noted. Sample intervals were recorded on core splitting sheets to be later incorporated in to the database.
- Sample numbers were placed on both sides of poly bags.
- Half of the cut core in placed in the poly bag and sealed close with a zip tie.
- Quality assurance/quality control (QA/QC) samples were inserted into the sample stream at prescribed intervals. A full description of the QA/QC program is provided in Section 11.3.
- Up to four samples bags were placed in rice bags and a record was made of the sample number placed in each rice bag and secured with zip-ties. The rice bags were labelled with GUNEXP and the enclosed sample numbers.
- At the end of every day, the rice bags were transported from the core logging facility to Gunpoint's office located in Sparks, Nevada (Figure 11.2).
- A sample submission form was completed and the samples were transported to the ALS laboratory facility located in Reno, Nevada.

The remaining core is stored temporarily on site until transported to Gunpoint's office in Sparks, Nevada for storage (Figure 11.2).

Figure 11.1 - Core Saw

Figure 11.2 - Core Storage at the Gunpoint Office

11.2.2 SAMPLE PREPARATION

ALS USA Inc. in Reno, Nevada is a division of ALS Laboratory Group. ALS USA has geochemical accreditation that conforms to the requirements of CAN P-4E International Organization for Standardization/International Electrotechnical Commission (ISO/IEC) 17025:2005.

The following is a brief description of the sample preparation ALS USA laboratories prep code Prep-31B:

- Samples are received, sorted into numerical order and then dried.
- Once dried, the material was initially crushed to 70% passing 6 mm and then crushed to 70% passing 2 mm.
- The sample is then split to get a 1,000 g sample for pulverizing.
- The 1,000 g split sample is pulverized to 85% passing 75 µm.
- Pulverized material is screened from -100 to 106 µm.

11.2.3 ANALYTICAL METHODOLOGY

The following is a brief description of the analytical procedure for screen metallic assay (ALS USA laboratories analytical code Au-SCR21) which is typically referred to a screen metallic:

- A total of 1,000 g of the final prepared pulp is passed through a 100 µm (Tyler 150 mesh) stainless steel screen to separate the oversize fractions.

- Any +100 µm material remaining on the screen is retained and analyzed in its entirety by fire assay with gravimetric finish and reported as the Au (+) fraction result.
- The -100 µm fraction is homogenized and two sub-samples are analyzed by fire assay with atomic absorption spectroscopy (AAS) finish (Au-AA25 and Au- AA25D).
  - In the fire assay procedure, the sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required in order to produce a lead button.
  - The lead button, containing the precious metals, is cupelled to remove the lead and the resulting precious metal bead is parted in dilute nitric acid, annealed and weighed to determine gold content.
- The average of the two AAS results is taken and reported as the Au (-) fraction result.
- The gold values for both the +100 and -100 µm fractions are reported together with the weight of each fraction as well as the calculated total gold content of the sample.

In addition to the gold assay, a 33-element inductively coupled plasma atomic emission spectroscopy (ICP-AES) package was run (ALS code ME-ICP61)

At no time was a Gunpoint employee or designate of the company involved in the preparation or analysis of the samples.

11.3 QA/QC PROGRAM

11.3.1 BLANKS

Gunpoint inserted a blank sample into the sample stream at a frequency of about one every 30 samples. The blank samples were acquired from Shea Clark Smith, Minerals Exploration & Environmental Geochemistry based out of Washoe Valley, Nevada, and consisted of a low-gold rhyolite tuff.

A total of 53 blank samples were submitted during the 2011 drilling program for an insertion frequency of 5%. Figure 11.3 graphs the results for the gold samples, and Figure 11.4 graphs the results for the silver samples. One sample or 2% of the blank data is deemed a failure and should be investigated.

Figure 11.3 - Gold Blank Chart

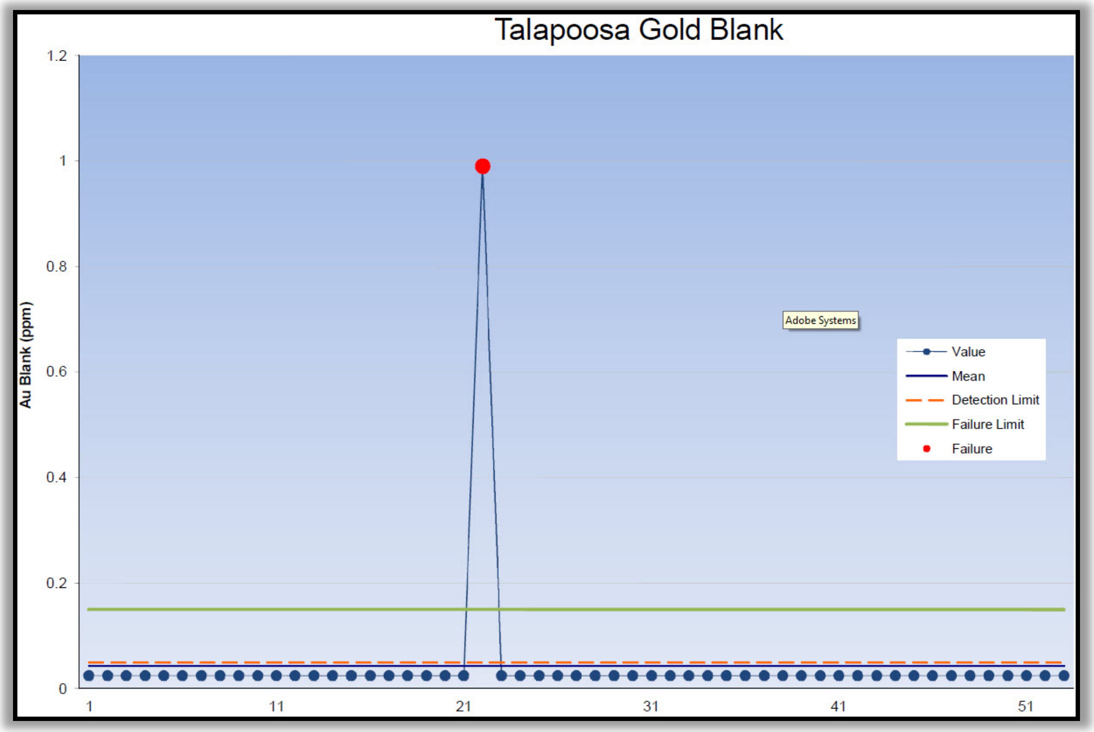
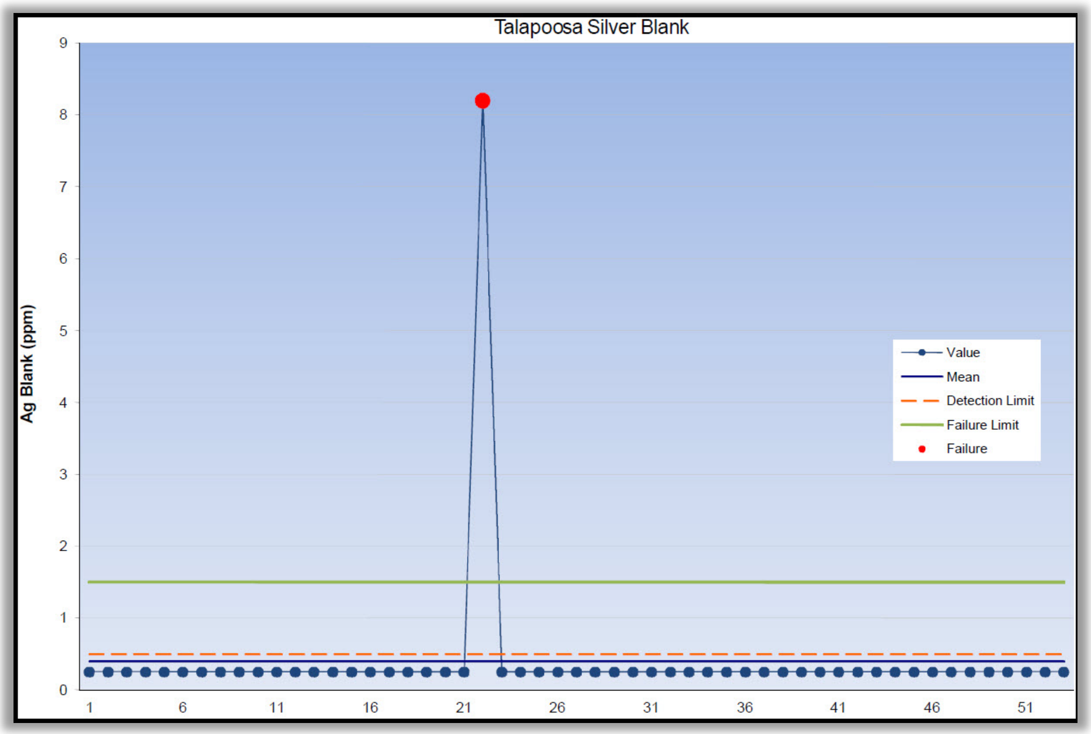


Figure 11.4 - Silver Blank Chart



11.3.2 DUPLICATES

Duplicate samples are inserted at a frequency of approximately one every 30 samples. A duplicate is ½ of a cut piece of core, which would be the equivalent of ¼ the size of a full piece of core.

A total of 32 duplicate samples were submitted. Figure 11.5 plots the gold assay duplicates. A majority of the duplicate sample fall outside the ±20% limits. This is indicative of coarse grained gold in drill core. In future drill programs, core duplicate should not be continued as part of the QA/QC program. Course rejects duplicates or pulp duplicates should be used in place of the core duplicates. It should also be noted that all duplicate samples above 1 ppm gold were biased high compared to the original.

Figure 11.5 - Gold Core Duplicate

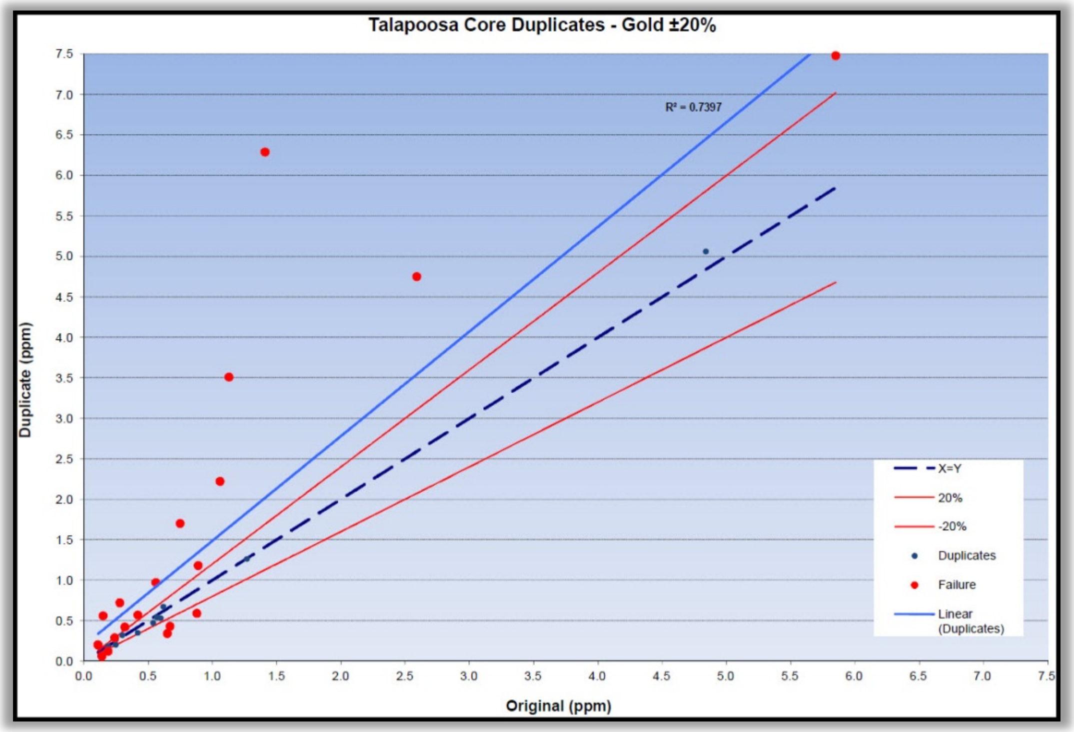
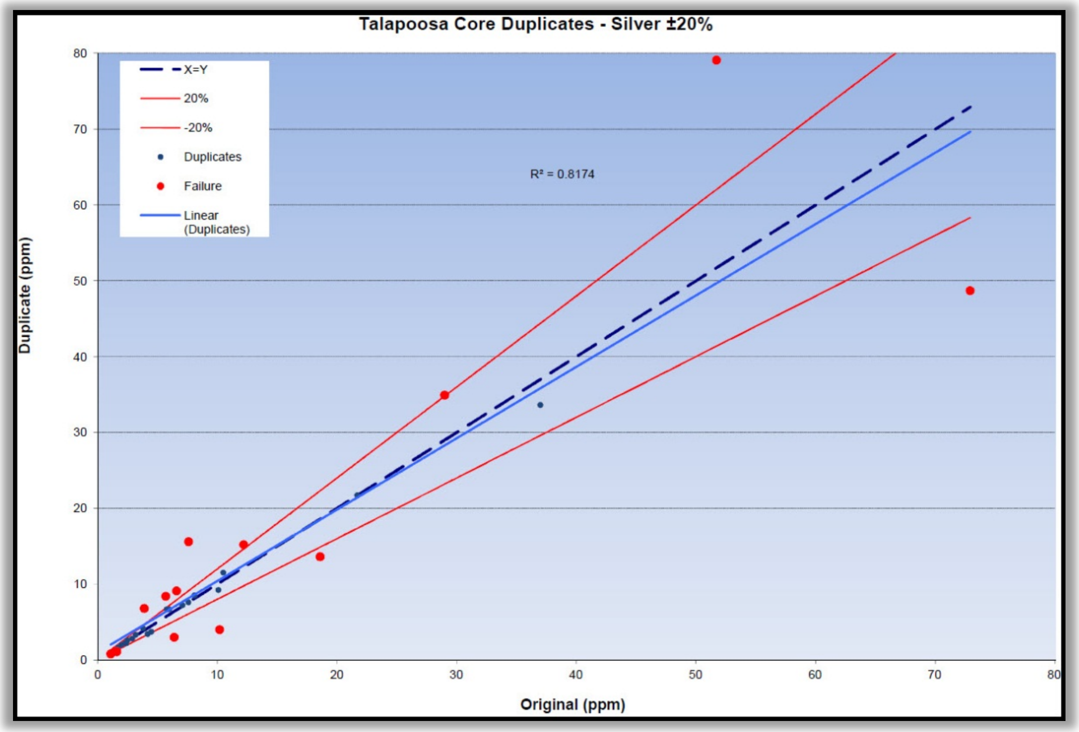


Figure 11.6 plots the silver assay duplicates.

Figure 11.6 - Silver Core Duplicate



11.3.3 STANDARD REFERENCE MATERIAL

Standard reference material (SRM) was inserted approximately one every 30 samples. A plastic block labelled with either STD-1 (Au.09.03), STD-2 (Au.09.04), STD-3 (Au.09.01), or STD-4 (S107004X) are placed in the poly sample bag during the logging and sampling process. The standards were placed in the poly sample bag at the Sparks office and then inserted in with the samples delivered from the Project site. Standards are acquired from Shea Clark Smith, Minerals Exploration & Environmental Geochemistry based out of Washoe Valley, Nevada

Table 11.1 shows the expected values of the SRM. Figures 11.7 to 11.14 plot the result of the SRM analysis. Since the standards are already in pulp form, they are only analyzed by fire assay. All of the other core samples and blanks are analyzed by both metallic screen analysis and fire assay. Although some failure exists, the size of the data set is not large enough to definitively indicate if an issue is present.

Table 11.1 - Standard Expected Values

	MEG-Au.09.01	MEG-Au.09.03	MEG-Au.09.04	MEG-S107004X
Au Mean (g/t)	0.7	2.1	3.4	1.2
Au Standard Deviation	0.07	0.166	0.204	0.07
Ag Mean (g/t)	9.6	17.2	26.3	8.0
Ag Standard Deviation	0.96	1.82	3.30	-

Figure 11.7 - SRM Au.09.01 – Gold Plot

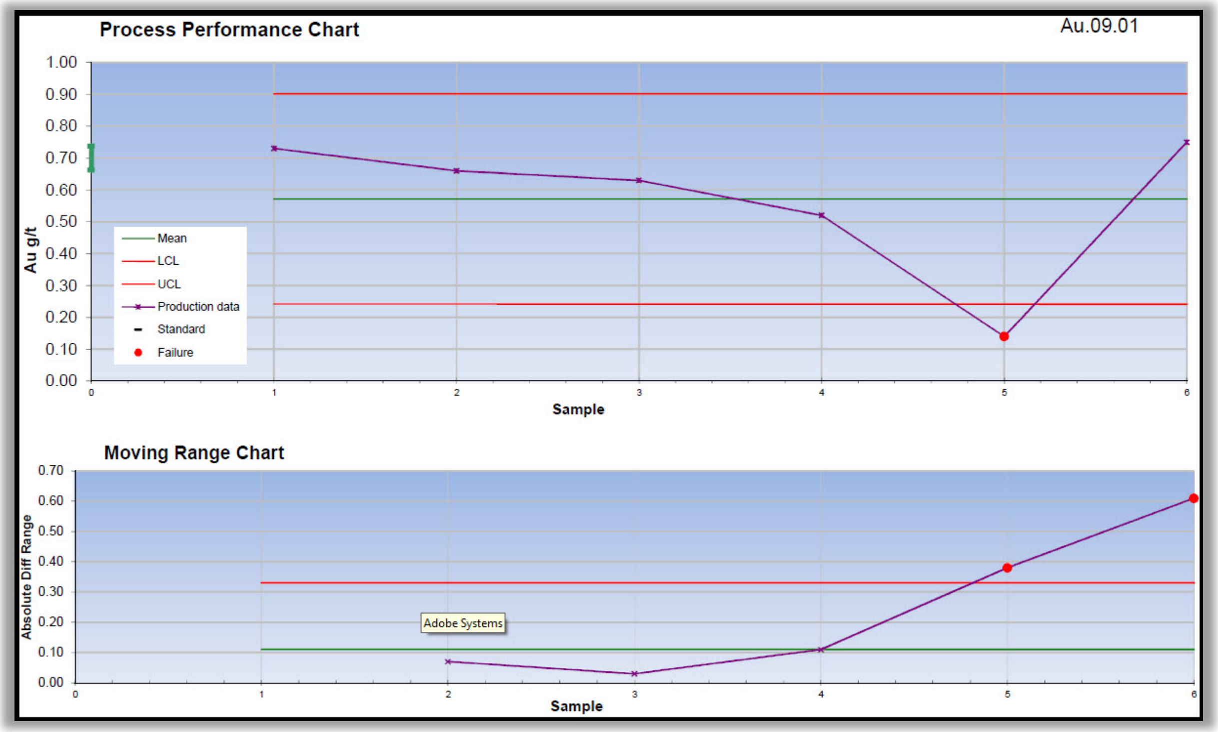


Figure 11.8 - SRM Au.09.01 – Silver Plot

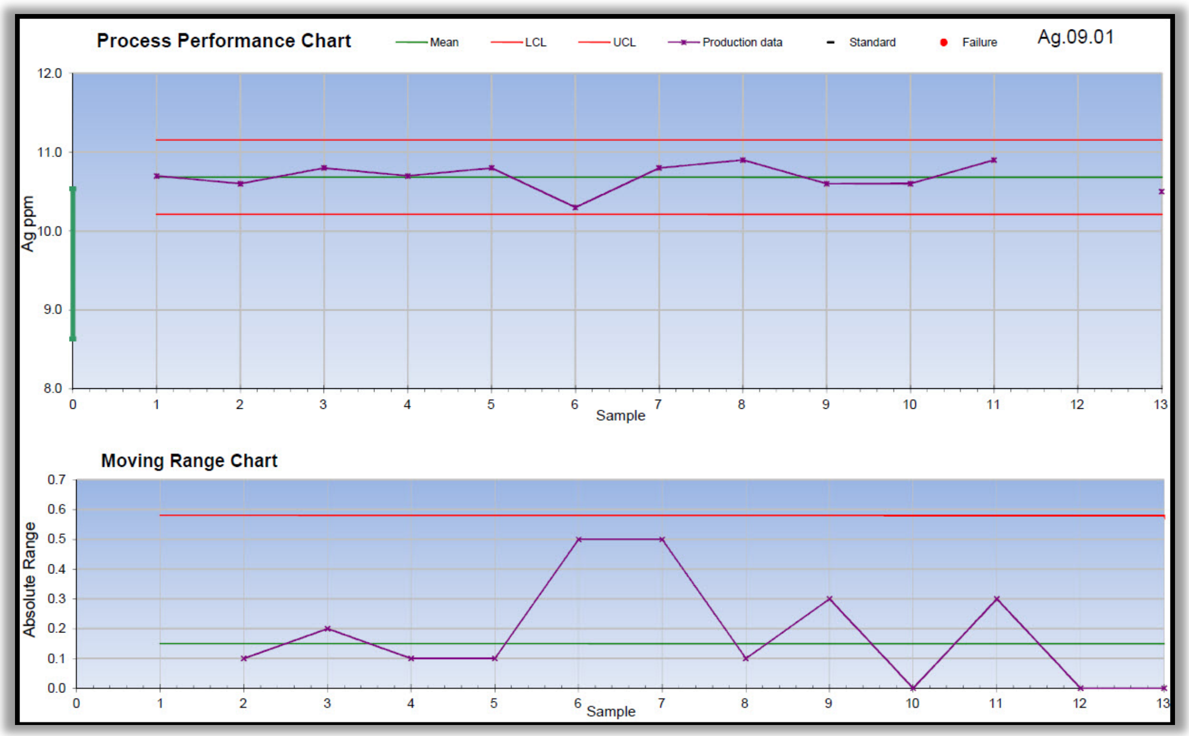


Figure 11.9 - SRM Au.09.03 – Gold Plot

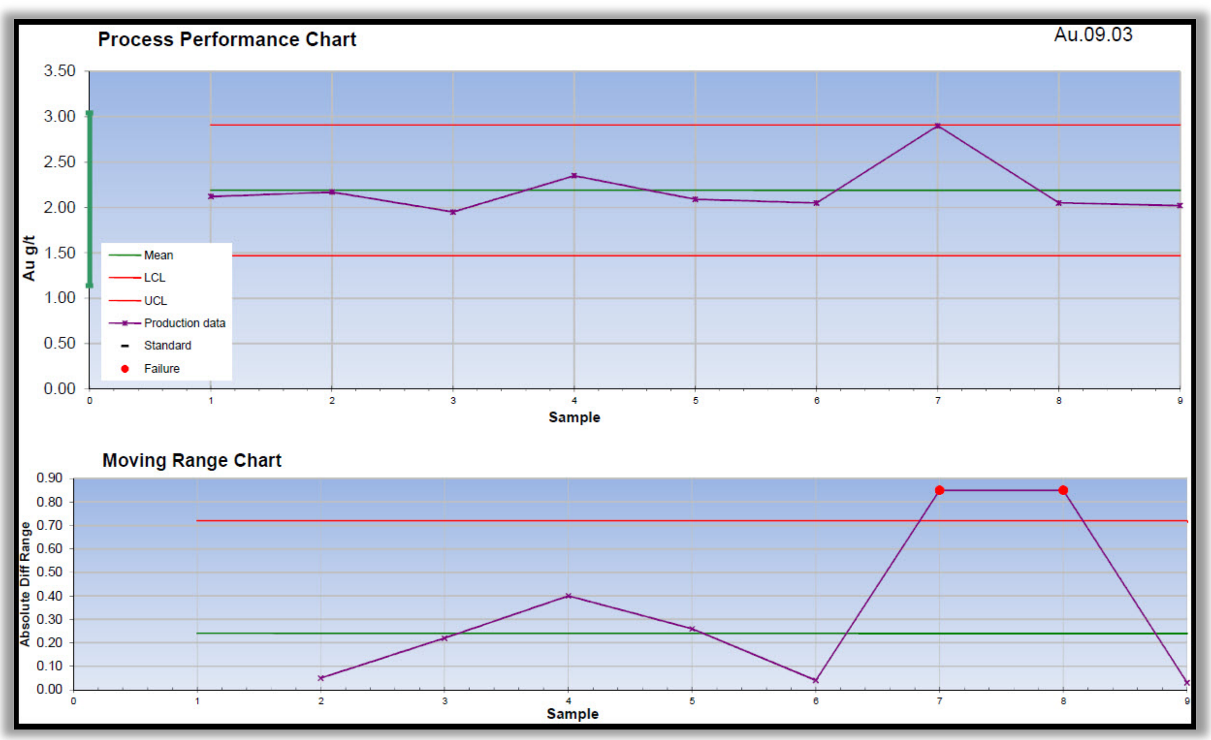


Figure 11.10 - SRM Au.09.03 – Silver Plot

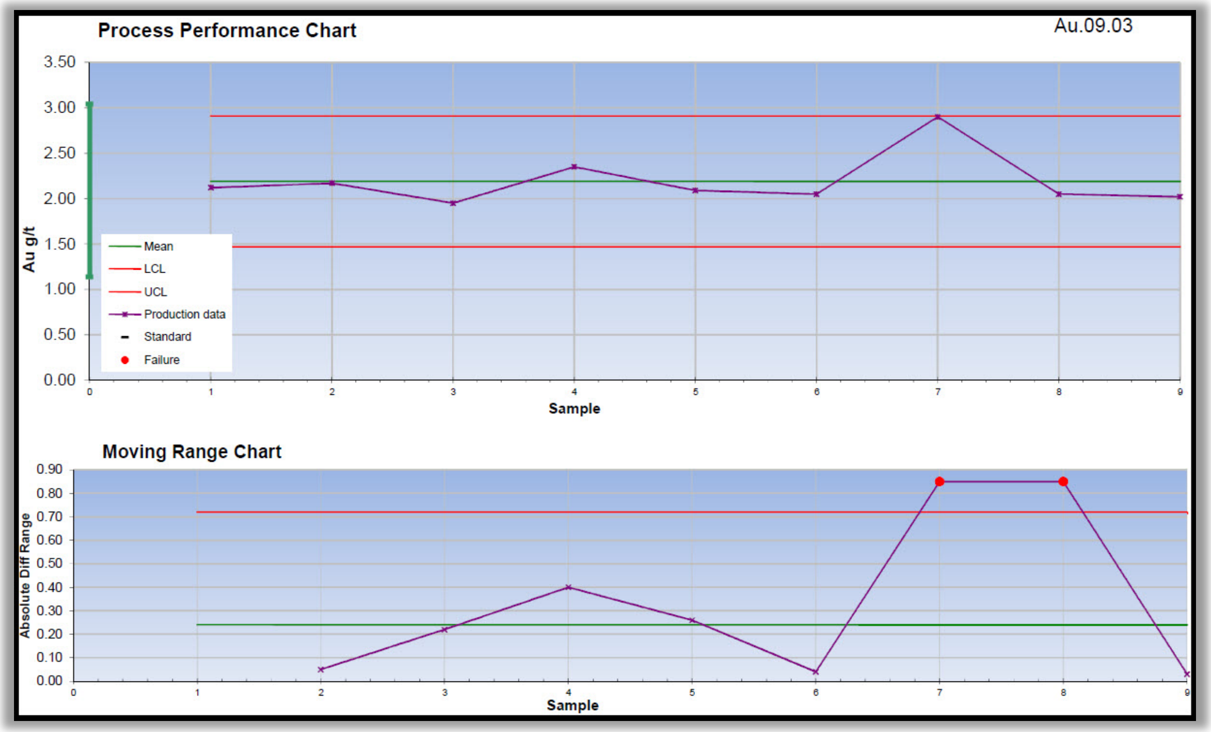


Figure 11.11 - SRM Au.09.04 – Gold Plot

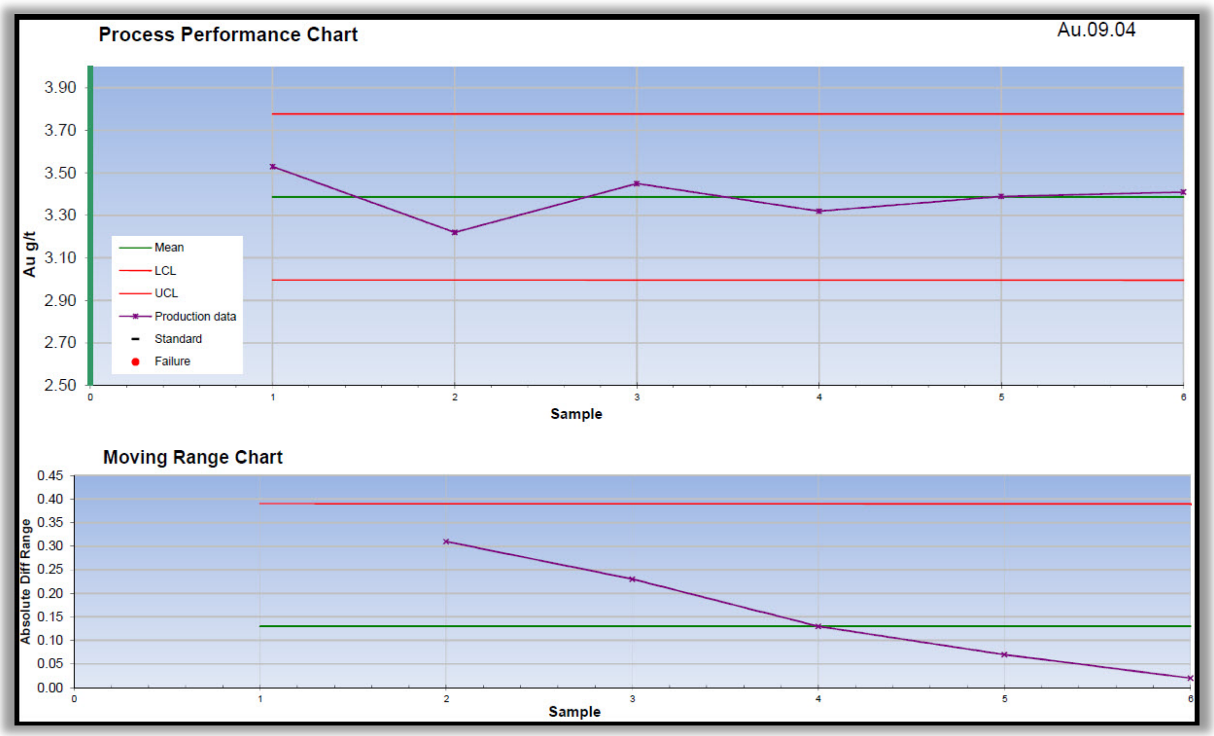


Figure 11.12 - SRM Au.09.04 – Silver Plot

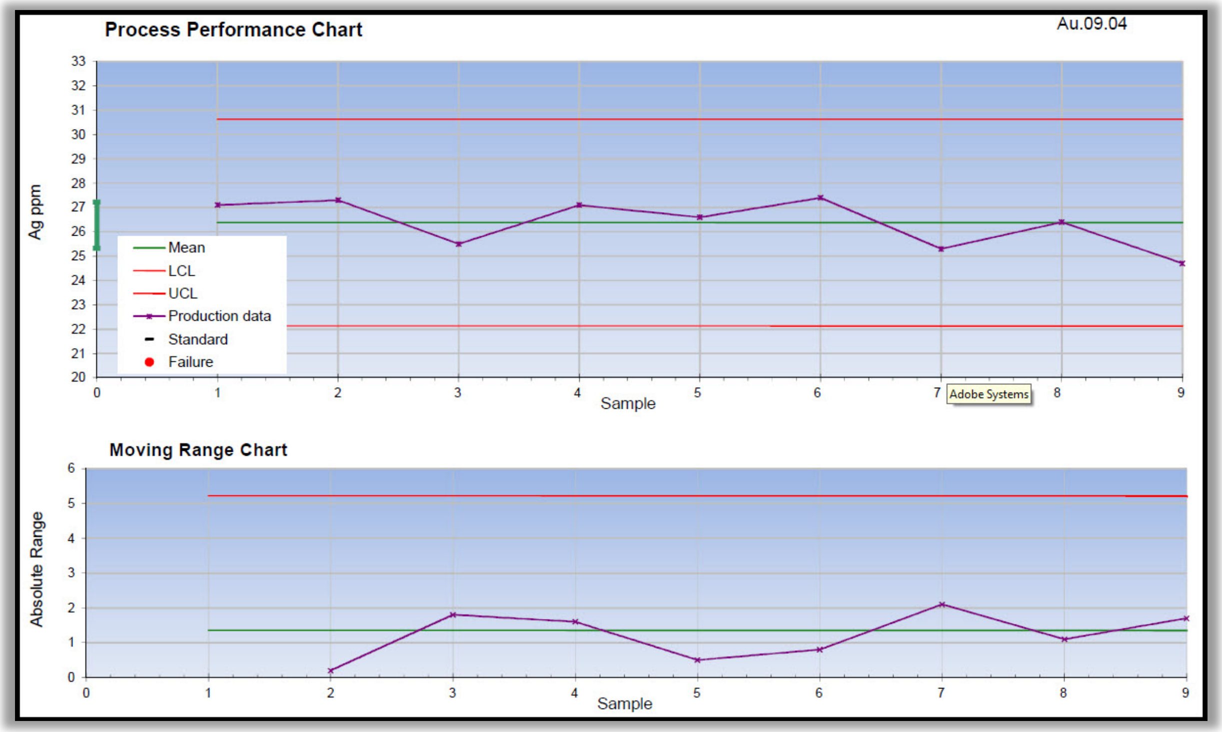


Figure 11.13 - SRM S107004X – Gold Plot

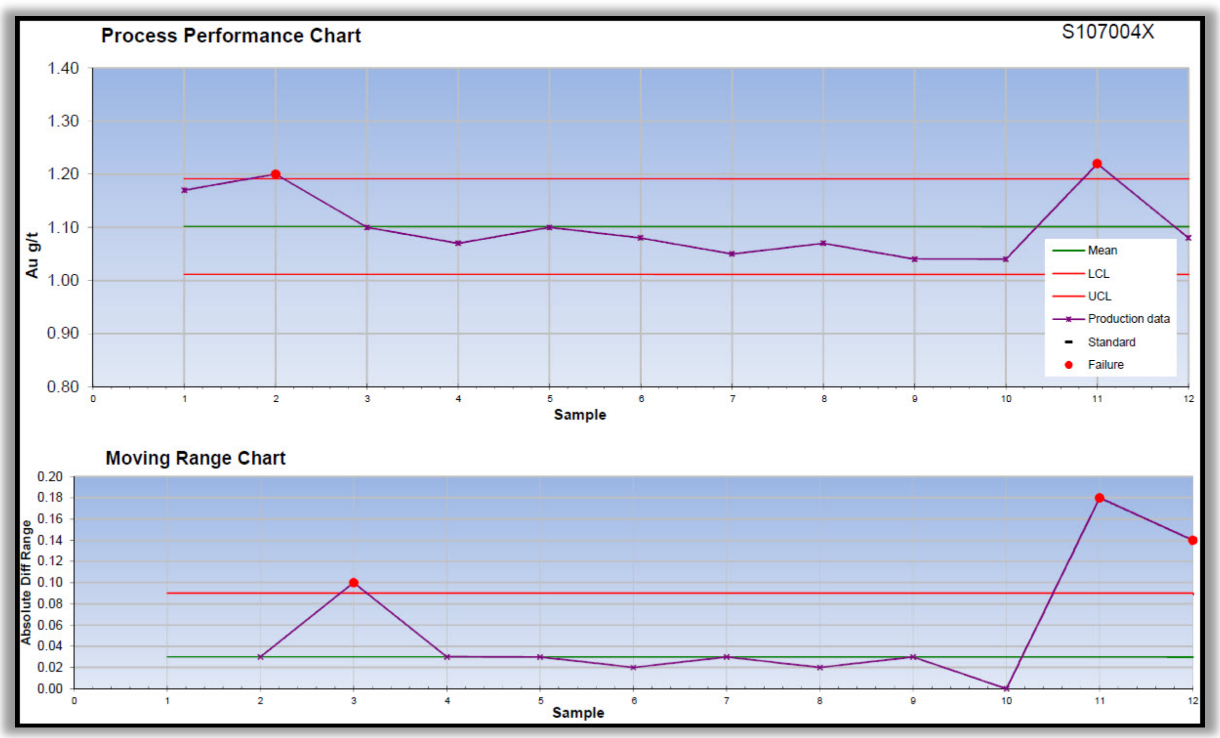
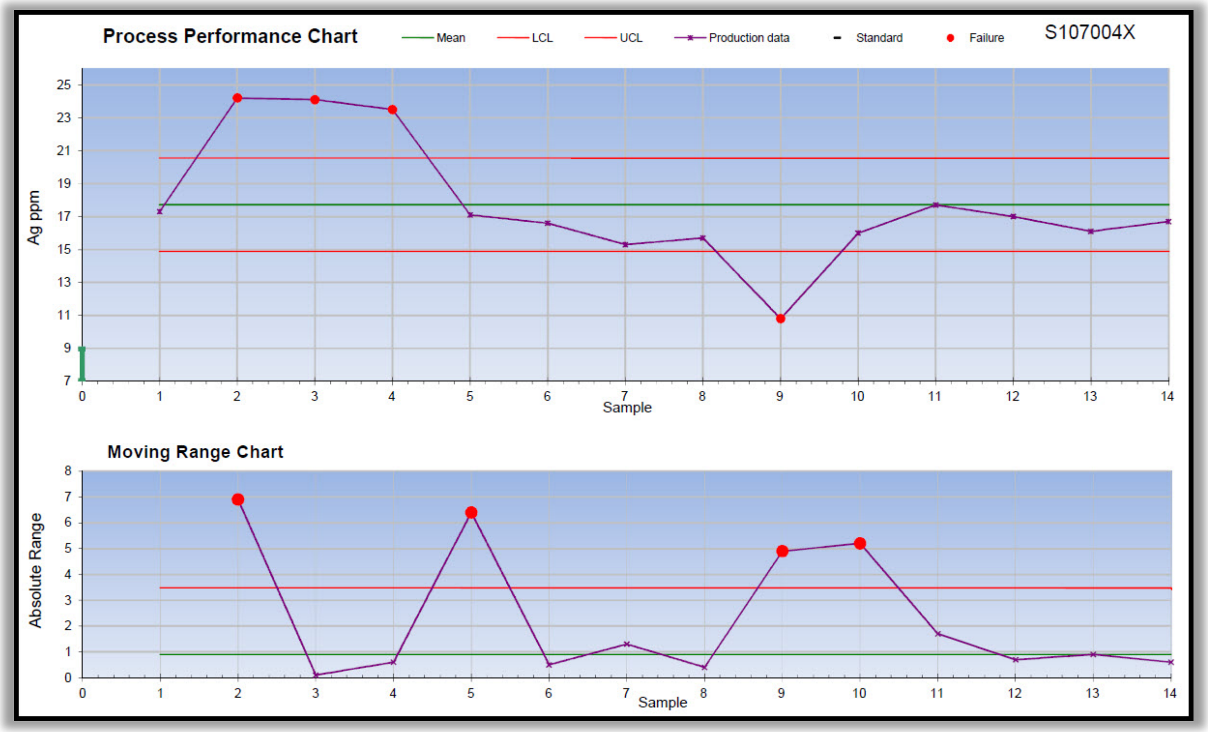


Figure 11.14 - SRM S107004X – Silver Plot



11.4 QP'S OPINION

It is WSP's opinion that the sample preparation and analytical procedures used in the Property meet acceptable industry standards and that the information can be used for geological and resource modelling.

12 DATA VERIFICATION

The QP, while employed by Tetra Tech, carried out an internal validation of the diamond drillhole file against the original drillhole logs and assay certificates. The validation of the data files was completed on seven of the Gunpoint boreholes completed during the 2011 drill program. Data verification was completed on collar coordinates, end-of-hole depth, down-the-hole survey measurements, and "from" and "to" intervals. No errors were encountered. A total of 100% of the assays data were validated against the original assay certificate. The error rate from this validation was 0%.

All assays entered in the database as being below detection limit with a "<" sign were converted to half the detection limit and were not considered to be errors in the data. All the data was converted to a consistent unit as over the year different units were used (Table 12.1).

Table 12.1 - Database Modifications

Element	Detection Limit Edits Made to Original Assay Data	Converted to
Au	<0.05	0.025
Ag	<0.5	0.25
Ag	>100	Used Alternative Ag_0G62 value
Au	(blank field)	Used Alternative Au-AA26 value

The QP, while employed by Tetra Tech, imported the drillhole data into the Datamine™ program, which has a routine that checks for duplicate intervals, overlapping intervals, and intervals beyond the end-of-hole. The errors identified in the routine were checked against the original logs and corrected.

The QP, while employed by Tetra Tech, visually observed the diamond drill setups on surface. Manual GPS validation was completed using a Garmin GPSMAP® 60Cx handheld device. Coordinates were collected using North American Datum (NAD) 27 Nevada State Plane (West). Table 12.2 summarizes the findings.

Table 12.2 - Drill Collar Validation

	Tetra Tech Data		
Borehole ID	Easting (m)	Northing (m)	Elevation (m)
GTI-001	304180	4369434	5,367
GTI-003	304247	4369288	5,334
GTI-004	304255	4369343	5,353

Seven independent samples of mineralized drill core and two standards were collected for check assaying representing typical mineralization grade ranges. The core was squared using a core saw and placed in plastic sample bag with sample numbers assigned by the QP. The samples were sent to ALS in Reno, Nevada for preparation and analysis. The same procedures used by Gunpoint for preparation and analysis were used by the QP.

ALS is accredited to international quality standards through ISO/IEC 17025 (ISO/IEC 17025 includes ISO 9001 and ISO 9002 specifications) with CAN-P-1579 (Mineral Analysis).

The results of the validation check samples for gold and silver indicate that the results of the check samples are mineralized and emphasize the highly variable nature of the grade distribution (Table 12.3). There are no gold results for the standards submitted by the QP, since the standards were already pulverized and could not be analyzed using the screen metallic procedure.

Table 12.3 - Check Sample Validation

BHID	Interval	Gunpoint Sample			Tetra Tech Sample		
		Sample No.	Au (g/t)	Ag (g/t)	Sample No.	Au (g/t)	Ag (g/t)
GTI-001	400-410	-	1.64	10.0	J350931	1.54	9.1
	550-560	-	1.59	9.0	J350932	1.35	13.4
	680-690	-	0.91	10.0	J350933	1.37	7.6
	MEG Au-09.01	-	0.69	9.6	J350934	-	9.9
GTI-003	320-330	-	0.68	17.0	J350935	0.62	15.1
	390-400	-	1.62	15.1	J350936	1.8	12.0
	520-530	-	1.48	6.7	J350937	0.73	5.3
	580-590	-	1.28	9.1	J350938	0.69	9.5
	MEG Au-09.04	-	3.40	26.3	J350939	-	25.1

The following QP completed a site visit of the Property:

→ Todd McCracken, P.Geo. visited the site from September 23 to 25, 2012.

12.1 HISTORICAL DATA

The QP, while employed by Tetra Tech, reviewed the work completed by Mine Development Associates (MDA) to rebuild the historical drill database. A summary of the work is described below and is derived from the MDA report (Ristorcelli, et al. 2010).

MDA re-constructed the database in 2008 by entering all available data into their corresponding fields.

All available hard-copy of assay certificates, collar coordinates, and downhole surveys were located and entered those data that did not already exist. The database reconstruction was organized by drilling campaigns so that the data could be more methodically evaluated.

Eighty-four percent of the gold and silver assays are backed up by original assay certificates or copies. Additionally, 68% of the collar coordinates and 100% of the down- hole surveys in the database are supported by original copies. The remainder of the data was compiled from older databases but could not be verified by originals or copies of certificates. A coding system was developed to reflect different levels of confidence and support in the entered data. The codes (Table 12.4) are based on the presence or absence of hard-copy assay certificates, as well as the presence or absence and results of assay quality control programs.

A second code, which assigns a use or no-use to sample-interval assay results, was also incorporated into the database. Only assays with Use codes were used for resource estimates. Assay labs or intervals with confidence codes of 0 were assigned No Use codes.

Table 12.4 - Confidence Code

CONFIDENCE CODE	USE / NO USE CODE	DESCRIPTION
3	1	High Confidence: assays supported by QA/QC program and hard copy assay certificates.
2		Moderate Confidence: assays supported by successful QA/QC program or hard copy assay certificates.
1		Low Confidence: program or product of lab that has produced poor QA/QC results in other campaigns, or assays takes from indirect sources.
0	0	No Confidence: no QA/QC program and unsupported by hard copy assay certificates.

12.2 QP'S OPINION

The Talapoosa data set is deemed to be valid and is acceptable for the use in resource estimation.

The QP agrees with the use of the Confidence Code and Use/No Use procedures implemented by MDA on the recent Gunpoint drilling data that was incorporated into the database. Data assigned a zero Use/No Use code was not included in the resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Multiple metallurgical testing programs were completed for the Property between 1981 and 1999. A detailed summary of those testing programs was presented as chapter 13 of the 2013 Tetra Tech technical report. Those summaries, along with the source documents referenced, were reviewed in detail by this author. That metallurgy summary contained in the2013 technical report was found by this author to be complete, and to accurately represent the metallurgical testing done on the property through 2013. Rather than rewriting those sections of the report concerning the metallurgical testing, they are repeated here (13.1 through 13.20).

A more recent metallurgical testing program was completed by McClelland Laboratories, Inc. in February 2015. That work was commissioned by Gunpoint Exploration Ltd., and was focused on evaluation of heap leaching of unoxidized materials from the Bear Creek Zone. A summary of that work has been added here as section 13.21.

13.1 KENNECOTT MINERALS COMPANY – JUNE 1981

Bear Creek Mining Company (Bear Creek) had sent 93 samples taken from surface grab samples and drill core to Kennecott Minerals Company - Process Technology (Kennecott). The rock types tested were shallow oxidized material taken from the 15 to 20 m intervals of drill core TA-3, and deeper primary mineralized material from the 100 to 150 m intervals of drillhole TA-4. TA-3 samples had a gold head grade of 0.96 g/t and 49.7 g/t silver. TA-4 had gold head grade of 1.23 g/t and 10.6 g/t silver.

The tests performed included:

- bottle roll tests on drill core samples crushed to 16 mm (5/8 in.) (performed at Dawson);
- bottle roll tests on grind samples of drill core (performed at Dawson);
- small column leach on drill core sample crushed to 16 mm (5/8 in.) (performed at Miller-Kappes Company).

The bucket leach tests were performed on grab samples taken from an adit at surface. The tests were done on a composite sample with a top size of 51 mm (-2 in) and another sample at 16 mm (-5/8 in). The 16 mm material had an average gold recovery of 61.4% and the 51 mm had an average gold recovery of 51.6%.

The results from the agitated leach of the 15 mm material are presented in Table 13.1. Results from the pulverized material can be found in Table 13.2.

Table 13.1 - Kennecott Bottle Roll Results on -15 mm Composites

Composite Sample	Calculated Head (g/t)		Extraction Percent (%)		NaCN (kg/t)
	Au	Ag	Au	Ag	
TA-3 1.5 to 21 m	0.96	41.1	40.9	17.1	1.74
TA-4 104 to 128 m	0.79	6.86	17.9	9.6	0.41
TA-4 128 to 152 m	0.72	6.86	9.2	9.3	0.73

Source: Bear Creek (June 1981)

Table 13.2 – Kennecott - Agitation Leach Test Results on Pulverised Samples taken from 1.5 m Interval Composites (-150 µm)

Sample	Calculated Head Assay (g/t)				Extraction Percent (%)						NaCN Consumption (kg/t)	
	Au		Ag		Dawson		Kappes					
Hole TA-3	Dawson	Kappes	Dawson	Kappes	Au (24h)	Ag (24h)	Au (24h)	Au (48h)	Ag (24h)	Ag (48h)	Dawson	Kappes
72147A	4.18	4.66	75.4	101.5	91.8	77.3	84.55	88.2	80.4	86.4	0.89	3.22
72149A	-	0.446	-	31.2	-	-	76.9	76.9	75.8	78.0	-	2.70
72151A	-	0.583	-	39.4	-	-	82.4	82.4	88.7	91.3	-	2.25
72153A	0.857	0.960	54.8	53.5	80.1	68.0	85.7	89.3	93.6	93.6	1.36	3.38
72155A	-	0.617	-	9.60	-	-	77.8	83.3	92.8	100.0	-	3.75
72157A	-	0.960	-	62.7	-	-	25.0	71.4	3.6	83.6	-	4.88
72159A	0.926	1.20	24.0	38.4	92.7	85.9	-	91.43	-	82.14	0.38	3.22
72307A	0.926	0.55	17.1	31.2	63.4	60.4	43.8	75.0	57.1	67.0	0.27	0.825
72315A	-	0.411	-	4.80	-	-	72.7	63.6	28.6	28.6	-	0.075
72320A	0.857	0.514	24.0	32.2	60.6	43.4	66.7	73.3	68.1	78.7	0.38	0.60
72324A	1.51	1.57	17.1	15.8	77.2	40.0	84.8	91.3	71.7	78.3	0.20	0.525
72328A	-	0.549	-	3.77	-	-	85.7	71.4	90.9	100.0	-	1.80
72331A	4.22	6.27	0.5	12.3	87.8	21.6	65.0	91.3	58.3	72.2	0.38	0.60
72337A	-	1.40	-	7.54	-	-	9.8	43.9	40.9	54.5	-	1.23

Source: Bear Creek (June 1981)

The agitation leach at 15 mm had poor gold and silver recoveries. The pulverized material (-150 µm) had markedly better gold and silver recoveries. These results lead to the conclusion that heap leach at sizes larger than 15 mm would not be feasible based on these samples. Samples which were ground to fine sizes which were subjected to agitated leach had higher recoveries. Drillhole TA-3 is described as representing the oxidized material and drillhole TA-4 the unoxidized material. The recoveries for the finer size oxidized material has better recovery than the finer unoxidized material with higher cyanide consumption.

A pulverized sample of drillhole SS-21 was subjected to bulk flotation and achieved 89.2% and 87.6% gold and silver recoveries respectively in the concentrate. The concentrate was leached and achieved 91% gold recovery and 83% silver recovery.

The possible process options which were theorized were as follows:

- Conventional agitation leach with carbon-in-pulp (CIP) recovery;
- Bulk flotation of a fine grind; the flotation concentrate could either be leached or smelted.

The results from this test work showed that heap leach may not be a feasible option due to low column leach precious metal recoveries.

13.2 HAZEN RESEARCH INC. – APRIL 1984

Hazen Research Inc. (Hazen) was contracted to determine the conditions required to create an efficient heap leach with oxidized and reduced samples from Talapoosa. Drillholes TA-3 and TA-4 were used to represent the oxidized and unoxidized samples respectively. The head assays for the samples are tabulated in Table 13.3.

Table 13.3 - Hazen – Head Assays – Drillholes TA-3 and TA-4

Drillhole	Au (g/t)	Ag (g/t)
TA-3	0.69	34.97
TA-4	1.03	8.91

Source: Hazen (April 1984)

The samples were subjected to attrition and then screened to different sizes prior to being subjected to bottle roll cyanide leach with a sodium cyanide dosage of 2 g/L for 96 hours. Samples were taken at 24, 48, and 72 hours and the entire pulp was filtered and washed at 96 hours. The results of these tests can be found in Table 13.4.

Table 13.4 - Hazen - Screened Feed Bottle Roll Leach Results

Composite	Size (µm)	Residual Gold g/t	Residual Silver g/t	Recovery Gold %	Recovery Silver %
TA-3	6,730	0.21	24.7	68	33 3
	2,380	0.10	25.4	81	38
	841	0.069	14.7	88	60
	230	0.034	9.60	95	72
TA-4	6730	0.48	8.57	68	24
	2,380	0.27	6.17	72	33
	841	0.069	5.83	89	46
	230	0.034	4.11	96	62

Source: Hazen (April 1984)

The results show that to achieve higher gold and silver recoveries the particle size must be reduced to below 2 mm (2,380 µm). An interesting point to note as well is that the oxidized and unoxidized samples behaved similarly for the gold solubilisation, but the oxidized sample had better silver solubilisation.

13.3 HEINEN-LINDSTROM CONSULTANTS – JANUARY 1986

The tests by Heinen-Lindstrom Consultants (HLC) were performed on samples from 15 drillholes and 4 bulk samples. Bottle roll tests were conducted at 40% solids using 1 kg/t of sodium cyanide. The measured head assays as compared to the calculated head assays are shown in Table 13.5. The measured and calculated head assays were in good agreement.

Table 13.6 presents the bottle roll leach results from the composite and individual drillhole samples. Table 13.7 is a summary of the screen analysis of the agitated cyanide leach residues for the four composites. These are the only particle size data available for the samples used in this test work.

Table 13.5 - HLC – Head Grade Comparison

Sample	Measured Head (g/t)		Calculated Head (g/t)	
	Au	Ag	Au	Ag
Composite M1	4.32	51.8	3.87	48.7
Composite M2	2.09	48.3	2.37	47.0
Composite M3	0.754	48.0	0.583	34.6
Composite M4-M5	1.02	11.6	1.16	13.0
T-RC-1: 6 to 12 m	1.99	28.4	1.54	22.3
T-RC-1: 18 to 24 m	3.36	42.5	3.98	39.8
T-RC-8: 9 to 12 m	4.46	280.8	4.35	256.0
T-RC-11: 4.5 to 10.5 m	0.69	7.54	0.617	6.51
T-RC-11: 22 to 29 m	1.85	18.8	1.68	17.8
T-RC-12: 13 to 19 m	2.40	103.2	2.40	94.3

Sample	Measured Head (g/t)		Calculated Head (g/t)	
	Au	Ag	Au	Ag
T-RC-13: 4.5 to 10.5 m	0.82	19.9	0.823	20.9
T-RC-13: 19 to 26 m	2.88	27.8	2.40	21.9
T-RC-15: 10 to 17 m	1.17	11.3	1.37	11.6
T-RC-16: 4.5 to 10.5 m	10.2	33.6	7.78	27.1
T-RC-31: 4.5 to 10.5 m	1.09	11.0	0.823	12.3
T-RC-31: 16 to 23 m	0.823	8.57	0.857	9.94
T-RC-32: 3 to 9 m	0.411	35.3	0.446	38.0
T-RC-33: 6 to 12 m	0.960	64.4	1.13	63.8
T-RC-33: 15 to 21 m	6.41	186.0	4.18	133.4

Source: HLC (January 1986)

Table 13.6 - HLC – Bottle Roll Leach Results

Sample	Au Extraction (%)						Ag Extraction (%)						Cyanide Consumed (kg/t)	Lime Added (kg/t)
	2 h	4 h	8 h	24 h	48 h	72 h	2 h	4 h	8 h	24 h	48 h	72 h		
M1	7.8	9.3	10.9	15.7	23.5	26.7	10.6	11.6	12.8	15.4	18.5	19.4	1.96	0.715
M2	7.7	10.2	10.3	17.9	20.7	25.8	1.4	1.9	2.4	4.4	4.3	7.5	0.88	0.625
M3	21.0	21.0	21.0	31.7	42.5	32.3	13.5	14.8	16.4	19.2	19.4	21.6	2.29	0.655
M4-M5	25.9	31.2	31.5	47.2	47.6	42.9	23.4	25.1	26.2	29.4	31.9	33.8	0.95	2.24
T-RC-1: 6 to 12 m	15.6	-	-	52.2	65.1	66.7	-	-	-	-	-	-	0.38	1.74
T-RC-1: 18 to 24 m	16.7	-	-	49.0	53.2	56.0	-	-	-	-	-	-	0.035	1.85
T-RC-8: 9 to 12 m	15.9	-	-	40.6	45.6	49.5	-	-	-	-	-	-	0.56	3.21
T-RC-11: 4.5 to 10.5 m	28.8	-	-	69.6	71.2	72.8	-	-	-	-	-	-	0.19	2.30
T-RC-11: 22 to 29 m	35.7 22 9	-	-	53.1	58.0	59.2	-	-	-	-	-	-	0.325	1.78
T-RC-12: 13 to 19 m	22.7	-	-	45.3	51.4	52.5	-	-	-	-	-	-	0.93	3.52
T-RC-13: 4.5 to 10.5 m	28.9	-	-	60.3	61.6	62.8	-	-	-	-	-	-	0.215	1.66
T-RC-13: 19 to 26 m	30.1	-	-	50.1	56.2	60.0	-	-	-	-	-	-	0.79	3.78
T-RC-15: 10 to 17 m	43.8	-	-	71.3	75.0	76.8	-	-	-	-	-	-	0.1	2.25
T-RC-16: 4.5 to 10.5 m	15.5	-	-	50.1	55.9	57.2	-	-	-	-	-	-	0.195	1.83
T-RC-31: 4.5 to 10.5 m	27.8	28.7	36.7	37.6	38.4	53.6	10.0	11.4	11.1	13.4	16.2	16.4	0.93	2.94
T-RC-31: 16 to 23 m	27.8	28.6	29.0	36.9	37.7	52.4	10.8	12.5	13.2	16.0	21.2	20.1	0.55	1.68
T-RC-32: 3 to 9 m	27.1	27.9	28.7	43.4	44.2	45.7	16.2	18.1	18.2	22.3	26.4	27.2	0.425	1.18
T-RC-33: 6 to 12 m	37.4	54.9	61.7	68.7	70.6	72.4	27.5	32.9 32	35.2	41.6	45.1	47.4	0.745	1.18
T-RC-33: 15 to 21 m	14.4	19.0	25.4	40.4	47.3	49.9	36.0	39.3	39.3	48.1	52.7	55.3	0.58	2.34

Source: HLC (January 1986)

Table 13.7 - HLC – Composite Screen Analysis of Agitated Cyanide Leach Residue

Size Fraction (µm)	Weight Percent (%)			
	M1	M2	M3	M4-M5
25,400	34.6	33.7	30.8	9.1
-25,400, +19,050	16.1	14.8	13.0	5.7
-19,050, +12,700	14.9	11.3	12.6	3.7
-12,700, +6,350	14.5	11.4	12.0	5.8
-6,350, +2,380	7.0	6.9	8.9	5.3
-2,380, +1,190	2.3	3.0	3.6	2.5
-1,190, +650	--	--	--	0.3
-650, +325	--	--	--	0.4
-325, +150	--	--	--	0.4
-150, +75	--	--	--	1.2
-75	--	--	--	65.6
Composite	100.0	100.0	100.0	100.0

13.4 BATEMAN LABORATORIES – NOVEMBER 1988

Bateman was asked to review the results from 14 column leach tests carried out by Athena, review historical test program data, and complete four bottle roll leach tests on samples supplied by Athena. The column leach samples consisted of four surface and five drill core samples. The surface samples were taken from previously sampled reverse circulation drillholes TRC-01, TRC-13, TAL-43, and a previously mined high grade deposit named “Glory Hole”. Surface samples were taken by trenching down 2.5 to 3 m (8 to 10 ft.) before collecting a sample. All samples were crushed to -9.5 mm except for the Glory Hole sample which was crushed to both -9.5 mm and -19 mm.

The results from the column leach test work are presented in Table 13.8. The gold recoveries varied from 32.5 to 80.3% and the silver from 15.5 to 69.2%. The final effluent in most cases was taken at 100 plus days of cyanide contact. There was a large variability in the recoveries which indicates that possibly there is a large variation in the mineralogy through the mineralized zone. The cyanide consumption was moderate to low. Overall the calculated and assayed heads are similar which indicates that the tests were performed correctly.

Four samples were sent to Bateman for use in bottle roll cyanidation tests. The results of these tests are presented in Table 13.9. The size fractions of the residues from the bottle roll leach are presented in Table 13.10. These are the only particle size data available for this portion of the test work. The gold recovery from the bottle roll tests varied from 16.7 to 58.8% and the silver recovery varied from 22.2 to 35.2%. To some extent, the bottle roll tests confirm the results from the column leach tests. They also indicate that the bottle roll tests could possibly achieve higher gold and silver recoveries at a finer particle size.

Table 13.8 - Athena / Bateman – Column Leach Results

Sample	Head Assay (g/t)				Extraction (%)										NaCN Consumption (kg/t)	Cement Added (kg/t)
	Calculated		Assayed		Au					Ag						
	Au	Ag	Au	Ag	5 d	10 d	30 d	60 d	Fe*	5 d	10 d	30 d	60 d	Fe*		
TRC-01 (9.5 mm)	2.23	35.3	2.02	36.7	14.0	20.3	32.4	47.2	56.9	6.3	10.2	16.9	25.2	31.8	1.24	10
TRC-01 A (6.3 mm)	2.33	29.1	1.85	33.2	56.7	62	67.3	70.4	71.9	22.5	25.4	29.0	31.3	32.5	1.19	10
TRC-01 B (6.3 mm)	2.19	30.8	1.61	31.2	51.8	57.7	63.4	67.0	68.6	22.4	25.2	28.4	30.8	31.4	1.16	10
TRC-01 B (6.3 mm)	2.47	36.7	1.68	31.5	31.9	44.3	62.0	71.6	73.7	3.1	13.7	33.4	39.0	41.0	1.70	10
T-01 (9 mm)	1.10	13.4	0.86	12.3	45.2	53.0	59.1	62.5	65.6	16.9	18.5	21.2	22.9	24.9	0.82	10
TA-10 (9 mm)	0.72	17.1	0.79	13.7	8.6	15.3	26.4	37.8	43.7	3.6	5.8	11.3	20.0	25.6	0.72	10
Glory Hole (19 mm)	0.82	31.5	1.23	28.4	13.5	22.5	32.0	35.4	37.8	8.0	13.8	19.9	21.5	22.8	0.65	10
Glory Hole (9.5 mm)	0.96	32.9	1.23	28.4	10.4	18.6	26.2	29.7	32.5	7.5	13.6	18.8	20.7	21.8	0.82	10
TA-2&3 (9.5 mm)	0.51	7.54	0.38	7.20	29.1	40.3	53.8	61.6	66.1	13.0	20.3	30.6	37.4	41.2	0.84	10
T-3 (9.5 mm)	0.51	8.91	0.45	19.2	67.6	70.5	74.2	77.1	80.3	53.3	56.5	60.0	65.2	69.2	1.08	10
TAL-43 (19 mm)	0.27	5.49	0.21	8.23	17.5	27.2	44.7	55.6	63.5	6.6	10.0	18.6	21.8	23.8	0.69	10
TAL-43 (9.5 mm)	0.31	3.43	0.21	8.23	28.0	38.2	51.9	59.6	66.6	22.1	30.3	40.6	45.5	49.0	0.70	10
TRC-13 (9.5 mm)	0.62	16.1	0.48	21.2	7.7	13.1	24.6	30.4	38.0	2.7	4.7	9.5	12.0	15.5	1.16	10
T-08 (9.5 mm)	0.45	6.17	0.34	7.54	18.0	26.6	34.5	40.1	45.4	10.9	13.6	18.5	21.8	23.8	1.22	10

Note: *FE = Final Effluent

Source: Athena / Bateman (1988)

Table 13.9 - Bateman – Bottle Roll Leach Results

SAMPLE	Head Assay (g/t)				Extraction (%)										NaCN Consumption (kg/t)	Lime Added (kg/t)
	Calculated		Assayed		Au					Ag						
	Au	Ag	Au	Ag	2 h	6 h	24 h	48 h	72 h	2 h	6 h	24 h	48 h	72 h		
TRC-01	2.43	32.9	2.16	16.1	28.4	37.1	47.9	55.9	53.5	17.6	20.2	31.5	31.9	31.2	0.715	6.63
T-08	0.27	3.43	0.069	2.74	5.2	5.3	21.1	11.2	16.7	4.3	5.6	10.9	16.3	22.2	0.105	2.38
TA-10	0.89	11.0	1.13	7.20	1.7	8.4	15.4	17.4	19.5	9.9	15.3	25.2	29.9	35.2	0.475	2.66
TA-2&3	0.34	12.7	0.34	15.8	27.0	50.3	47.1	61.8	58.8	15.7	19.3	24.5	28.8	31.6	0.405	6.44

Source: Bateman (1988)

Table 13.10 - Bateman – Residue Fraction Analysis from Bottle Roll Leach Tests

Zone	Flotation Recovery (%)		Concentrate Grade (g/t)		Concentrate Leach Recovery (%)		NaCN Consumption (kg/t)	Lime Consumption (kg/t)
	Au	Ag	Au	Ag	Au	Ag		
Main	40.3	54.1	5.35	328	97.4	96.0	2	15.15
Extension	92.4	93.3	15	480	72.0	81.10	4.1	17.4
Bear Creek	84.3	84.5	17.6	307	87.1	82.8	2.0	17.0

Source: Bateman (1988)

13.5 MINPROC ENGINEERS INC. – VIABILITY STUDY – AUGUST 1989

Minproc was contracted to produce a viability study for the Project. The report mentions the results of some column leach, direct cyanidation, and flotation tests which were completed on drill chip samples selected at the site from the “Main” and “Sulphide” (Bear Creek) Zones. The report does mention that the samples may not be representative of the proposed mineable area and the results from these tests should be considered as “scoping” results. The report also gives a description and costing for a proposed processing plant.

The results of the test work were described, although no tables or graphs of the work were presented. Column leach tests on the Main Zone sample gave 57.4% gold recovery on 13 mm (0.5 in.) material and 68.1% at 6 mm after 54 days of contact with cyanide. The report suggests that the sulphide mineralized material was expected to offer lower recoveries, and based on the Main and Sulphide Zones results, heap leaching was not believed to be selected as the process for this deposit.

Direct cyanidation gave 97% gold recoveries on Main Zone samples with 24 to 48 hours residence times. The results of a direct cyanidation of a sample assembled from Main Zone drill cuttings from 35 to 40 m (115 to 135 ft.) led the Minproc author to suggest that there was a correlation between the gold recovery and sample depth. The deeper sample gave a recovery of 63% gold. However, the deepest sample from 148 m (485 ft,) gave a gold recovery of 75%.

Flotation tests on material with a grind of P_{80} = -75 µm, gave gold recoveries to concentrate of 92.4 and 84.3% for Bear Creek samples and 40.3% for the Main Zone sample. Further optimization of the flotation yielded gold recoveries to concentrate of 95.9% and 98.1% for the Bear Creek samples, and 59.7% and 60.6% for the Main Zone.

Initial flotation concentrate leach gold recoveries were 72% and 87.1% with high cyanide consumptions of 2 to 4 kg/t. The gold recovery from the leach of the optimized flotation concentrate was in the neighbourhood of 80%, with even higher cyanide consumption.

The proposed process facility would utilize a semi-autogenous grinding (SAG) mill and ball mill to achieve a grind of P_{80} = -75 µm. The ball mill hydrocyclone underflow will feed a gravity circuit (i.e. Reichert cone/spirals/shaking table) to try and isolate and recover any electrum which might not be collected in flotation or may cause slower leach times of the flotation concentrate. The overflow will be subjected to column flotation. Flotation concentrate would be leached in leach tanks, dewatered using counter-current decantation, and the gold recovered from the pregnant solution in a packaged Merrill Crowe plant (i.e. zinc precipitation).

13.6 MCCLELLAND LABORATORIES INC. - 1989

13.6.1 FLOTATION / CYANIDATION TESTS

McClelland completed the test work utilized in the Minproc viability report. The laboratory reports give further details of the program. In the opinion of the McClelland reports author, heap leach is not a viable process option for the Project due to the varied mineralogies and poor recoveries in the test work. Gold recoveries were in the range of 50% and the silver recoveries were lower. The heap residence times would most likely be long to obtain viable heap leach precious metal recoveries.

A section of flotation work was completed on composite samples from the Bear Creek, Extension and Main Zones. These tests were given the Job No. 1299. The Main Zone composite was created from sections from drillholes TAL-151 and 154, the Extension Zone composite from TAL-141 and TAL-151, and the Bear Creek Zone composite from TAL-127, TAL-129 and TAL-157. The flotation tests were completed at a grind of P_{80} =-75 µm. The results from these tests have been tabulated in Table 13.11. Flotation was carried out at P_{80} = -75 µm and the concentrates were subjected to a grind at 100% -37 µm prior to intensive cyanidation.

Table 13.11 - McClelland 1989 – Flotation and Cyanide Leach Test Results (Job No. 1299)

Zone	Flotation Recovery (%)		Concentrate Grade (g/t)		Concentrate Leach Recovery (%)		NaCN Consumption (kg/t)	Lime Consumption (kg/t)
	Au	Ag	Au	Ag	Au	Ag		
Main	40.3	54.1	5.35	328	97.4	96.0	2	15.15
Extension	92.4	93.3	15	480	72.0	81.10	4.1	17.4
Bear Creek	84.3	84.5	17.6	307	87.1	82.8	2.0	17.0

Source: McClelland (1989)

The Main Zone does not appear to be quite as amenable to flotation as the Extension and Bear Creek Zones, but the Main Zone flotation concentrate was more amenable to cyanidation than the other two zone samples. The opposite was true for the Extension and Bear Creek which showed good flotation recoveries and lower cyanidation recoveries.

An additional set of flotation/cyanidation tests were performed which further optimized the flotation and cyanidation. These tests were given the Job No. 1373. Initial flotation work completed on the Main Zone drill cuttings samples showed poor flotation recoveries, but good cyanidation of the concentrate. The work on the Bear Creek drill cuttings sample showed the opposite with good flotation response but lower precious metal leach recoveries from the concentrate. Tests utilizing a bulk sulphide flotation/sulphidizing agent (sodium sulphide) (Main 1) and bulk sulphide float/sulphidizing agent/fatty acid (Main 2) were performed on the Main Zone sample to try and boost the precious metal recoveries. The flotation concentrates were then subjected to regrind to reduce the particle size to 100% -37 µm and a 96 hour intensive cyanidation. The results from these tests are presented in Table 13.12.

Table 13.12 - McClelland 1989 – Flotation and Cyanide Leach Test Results (Job No. 1373)

Zone	Flotation Recovery (%)		Concentrate Grade (g/t)		Concentrate Leach Recovery (%)		NaCN Consumption (kg/t)	Lime Consumption (kg/t)
	Au	Ag	Au	Ag	Au	Ag		
Main 1	55.5	63.2	10.2	234	96.3	95.8	37.05	52.05
Main 2	65.5	63.3	4.25	91.5	96.0	94.4	14.7	16.4
BC-1	96.2	93.6	30	218	80.4	82.7	16.25	7.7
BC-2	96.3	87.6	23.0	153	82.8	68.8	21.1	8.9

Source: McClelland (1989)

The overall gold recoveries for BC-1 and BC-2 were 77.2% and 80.5% respectively. The overall silver recoveries for BC-1 and BC-2 were 73.3% and 61.2% respectively. Mineralogical work on the rougher concentrate has shown the presence of electrum. The presence of electrum explains the slower leach kinetics on the flotation concentrates.

The average head assays for the Main Zone and two Bear Creek samples are presented in Table 13.13.

Table 13.13 - McClelland 1989 – Average Head Assays for Flotation Test Work

	Main Zone		BC-1		BC-2	
	Au	Ag	Au	Ag	Au	Ag
Average Head Assay (g/t)	1.27	21.6	1.61	19.9	1.78	12.0

Source: McClelland (1989)

13.6.2 DIRECT CYANIDATION

The first set of tests for direct cyanidation of the whole sample (without pre- concentration) was completed on samples of the Extension and Bear Creek Zones. The direct cyanidation was carried out in mechanically agitated baffled vessels for 48 hours. The samples were fed to the leach at a particle size of P_{80} = -53 µm. The results from these tests have been summarized in Table 13.14. The results were not as good as those from the previous flotation/cyanidation test work but the previous tests were done at a finer grind, longer residence time, and on a flotation concentrate.

A second set of direct cyanidation tests were completed on drillhole composite samples. Eighteen composites were created from six drillholes. The size was reduced to a nominal 75 µm. An additional four composites were also subjected to 96 hour direct cyanidation, but at their “as received” size. The results from these tests can be found in Table 13.15.

Gold and silver extraction rates were fairly fast for the majority of samples. The variance in direct cyanidation precious metal recoveries (i.e. gold went from 63.6 to 97.1%) indicates that there are different mineralogies across the samples tested. The cyanide consumption was consistently low and the lime addition varied and was quite high. The pH differed across the sample set.

The as-received samples also performed well. Results are presented in Table 13.16. Some of the samples were comparable to the recoveries of the finer particle composite from the same drillhole. Others performed far better (Composite 17) at a finer grind than the as-received sizing.

The next set of 96-hour direct cyanidation tests were conducted on drill cuttings samples at the “as received” size of nominal 6.35 mm. The 11 composites were created from 43 Bear Creek drill cuttings intervals. The results from these direct cyanidation tests are summarized in Table 13.17.

The cyanide requirements were low, and the lime requirements varied from moderate to high. The samples did not all appear readily amenable to direct cyanidation in this “as- received” size range.

The third set of 96 hour direct cyanidation tests were performed on two Talapoosa drill core composites (i.e. TC-2 and TC-4). Additional tests were performed using 5 kg/t of Portland cement and 10 kg/t sodium cyanide. Agglomerated (5 kg/t Portland cement) column leach tests were also performed on three composite samples from these drillholes. The results from the direct cyanidation are presented in Table 13.18 and the column leach tests in Table 13.19.

The results for the direct cyanidation showed that the two samples tested were not amenable to direct cyanidation, but that the pre-treatment with Portland cement and cyanide did increase the precious metal recoveries. Cyanide consumption was low and lime consumption was low to moderate.

The results from the column leach tests showed that the finer (6.35 mm) particle size column leach had much better precious metal recoveries than the coarser (12.7 mm) particle size. The cyanide consumption was low to moderate and the lime consumption was high.

Table 13.14 - McClelland 1989 – Direct Cyanidation Bottle Roll Leach Results, P80=53 µm – Part 1

Sample	Head Assay (g/t)				Extraction (%)												NaCN Consumption (kg/t)	Lime Added (kg/t)
	Calculated		Assayed		Au						Ag							
	Au	Ag	Au	Ag	2 h	4 h	8 h	12 h	24 h	48 h	2 h	4 h	8 h	12 h	24 h	48 h		
Extension	2.26	62.7	1.78	56.9	24.7	34.1	43.3	49.2	59.4	69.7	26.7	32.7	43.1	47.5	54.2	63.9	2.10	7.2
Bear Creek	1.44	28.8	1.41	26.7	38.8	46.9	55.0	60.5	66.2	76.2	32.0	37.9	48.0	507	54.4	49.5	2.11	7.25

Source: McClelland (1989)

Table 13.15 - McClelland 1989 – Direct Cyanidation Bottle Roll Leach Results, -75 µm – Part 2

Drillhole	Interval	Composite No.	Recovery (%)		NaCN Consumption (kg/t)	Lime Consumption (kg/t)
			Au	Ag		
TAL-5	0 to 4.5 m	1	97.1	84.4	0.085	9.4
	10 to 17 m	2	85.7	95.5	0.05	13.6
	22 to 29 m	3	84.9	71.9	0.09	13.0
	35 to 41 m	4	63.6	75.0	0.2	10.5
TAL-6	1.5 to 8 m	5	86.4	77.4	0.05	9.1
	13.7 to 20 m	6	79.3	68.4	0.05	18.1
	26 to 32 m	7	68.2	55.6	0.05	8.4 5
TAL-9	1.5 to 8 m	8	88.0	67.7	0.05	11.5
	13.7 to 20 m	9	85.0	85.7	0.05	10.0
	26 to 32 m	10	86.8	50.0	0.05	7.8
TAL-27	1.5 to 8 m	11	85.7	81.0	0.05	7.4
	20 to 26 m	12	78.6	78.9	0.08	10.0
	38 to 44 m	13	83.1	76.5	0.05	10
TAL-58	9 to 15 m	14	80.0	57.7	0.05	11.8
	21 to 27 m	15	77.8	78.6	0.05	15.0
TRC-1	0 to 6 m	16	76.3	64.0	0.05	9.6
	12 to 18 m	17	91.6	66.2	0.05	8.8
	24 to 27 m	18	65.1	56.9	0.05	14.2

Source: McClelland (1989)

Table 13.16 - McClelland 1989 – Direct Cyanidation Bottle Roll Leach Results 'As-Received' Sizes – Part 2

Drillhole	Interval	Composite No.	Recovery (%)		NaCN Consumption (kg/t)	Lime Consumption (kg/t)
			Au	Ag		
TAL-5	0 to 4.5 m	1	92.3	73.5	0.2	8.4
	22 to 29 m	3	85.5	67.8	0.12	12.0
TAL-6	26 to 32 m	7	72.7	41.7	0.42	17.8
TRC-1	12 to 18 m	17	56.3	23.5	0.45	13.2

Source: McClelland (1989)

Table 13.17 - McClelland 1989 – Direct Cyanidation Bottle Roll Leach Results Bear Creek Drill Cuttings 'As-Received' Sizes – Part 3

Drillhole	Interval	Recovery (%)		NaCN Consumption (kg/t)	Lime Added (kg/t)
		Au	Ag		
TAL-127	77.5 to 84 m	54.3	34.5	0.07	12.6
	108 to 114 m	42.6	--	0.05	3.8
	149 to 155 m	29.3	31.4	0.05	2.7
	175 to 181 m	60.0	36.8	0.05	5.0
TAL-129	56 to 62 m	36.1	21.8	0.20	4.6
	122 to 128 m	59.4	31.8	0.22	4.2
	141.5 to 148 m	75.4	45.7	0.09	8.6
TAL-130	73 to 79 m	30.9	21.9	0.17	3.2
	105 to 110 m	64.6	50.4	0.20	8.5
	157 to 161.5 m	26.0	25.0	0.05	2.6
TAL-148	41 to 49 m	27.0	19.2	0.12	4.4

Source: McClelland (1989)

Table 13.18 - McClelland 1989 – Direct Cyanidation Bottle Roll Leach Results

Sample	Head Assay (g/t)				Extraction (%)												NaCN Consumption (kg/t)	Lime Added (kg/t)
	Calculated		Assayed		Au						Ag							
	Au	Ag	Au	Ag	6 h	12 h	24 h	48 h	72 h	96 h	6 h	12 h	24 h	48 h	72 h	96 h		
TC-2	1.75	14.4	1.61	13.0	18.2	23.5	27.3	30.2	36.7	37.3	9.5	10.7	11.9	13.1	14.3	14.3	0.16	6.0
TC-2 Agg	1.61	13.4	1.61	13.0	31.9	34.3	36.6	39.8	44.0	44.7	10.3	10.5	11.5	12.8	14.1	15.4 5	0.07	0.05
TC-4	4.49	29.8	1.95	19.9	8.2	10.8	14.4	17.1	18.5	19.1	9.3	10.5	12.3	14.1	14.9	14.9	0.18	5.0
TC-4 Agg	1.78	25.4	19.5	19.9	41.3	42.5	43.5	44.4	45.6	46.2	15.8	16.2	17.2	17.6	18.5	18.9	0.37	0.2

Source: McClelland (1989)

Table 13.19 - McClelland 1989 – Column Leach Results – Part 1

Sample	Head Assay (g/t)				Extraction (%)												NaCN Consumption (kg/t)	Lime Added (kg/t)
	Calculated		Assayed		Au						Ag							
	Au	Ag	Au	Ag	10 d	15 d	20 d	27 d	37 d	54 d	10 d	15 d	20 d	27 d	37 d	54 d		
TC-2 (12.7 mm)	1.61	11.0	1.54	11.7	42.6	47.7	51.5	54.3	55.3	57.4	16.6	19.8	22.4	25.0	25.9	31.3	1.025	5
TC-2 (6.35 mm)	1.61	11.3	1.65	12.3	63.8	64.9	65.7	66.0	66.6	68.1	38.4	39.9	41.0	42.5	43.2	48.8	0.42	5
TC-4 (6.35 mm)	1.75	26.7	1.58	24.7	59.6	62.0	63.5	63.9	64.5	64.7	29.3	31.3	32.6	34.1	34.6	37.2	1.14	5

Source: McClelland (1989)

13.7 PLACER DOME U.S. INC. / GOLDEN SUNLIGHT MINES, INC. – REVIEW OF PLACER DOME'S INITIAL PHASE PROGRAM – 1990

Placer Dome completed a repeat of the direct cyanidation test work performed by McClelland to verify the results. Samples from the Main Zone and two samples from the Bear Creek Zone were used. The tests were run at 22 to 38% +150 µm and 26 to 29% +75 µm. The tests were done in duplicate, but reproducibility of the results was an issue. It was believed there are some issues with getting accurate assays with this deposit. The results given for the direct cyanidation were that 78 to 83% gold recoveries were achieved and in the grind size tested the size did not appear to have an effect on precious metal recovery. Cyanide consumption was low in the 0.5 to 0.6 kg/t range, and the lime consumption was moderate (1.9 to 2.3 kg/t) for the Bear Creek samples but high (5 kg/t) for the Main Zone leaches.

Flotation tests using the same parameters as the McClelland tests were also completed for samples from the Main Zone and Bear Creek. The flotation concentrates were not subjected to cyanidation. The BC-1 composite had a flotation recovery of 84% gold and the BC-2 had a gold recovery of 95%. The Main Zone flotation utilized the bulk sulphide/sulphidization/fatty acid flotation yielded 65.5% gold recovery. A subsequent test using a sulphuric acid scrub and copper sulphate activation yielded a 67% gold recovery in the concentrate. A Bond work index test was also completed on BC-2 giving a work index of 17.3 kWh/t (15.7 kWh/ton).

During its drilling campaign, Placer Dome defined the mineralogy in the Main Zone. There are two types of mineralization excluding the oxide mineralization. The two types are sulphide and hematite. The sulphide can further be subdivided into four subgroups and the hematite into clay rich, soft, and high grade.

Two composites were compiled for direct cyanidation and flotation/cyanidation test work by Golden Sunlight Mines, Inc. The same regrind step of the concentrate prior to cyanidation from the McClelland work was repeated in this work. The first composite was a Bear Creek composite to represent the sulphide mineralogy and the second was a hematite sample. Direct cyanidation was performed on samples at 30% +150 µm and 27% 75 µm grind sizes for each composite. The direct cyanidation gold recoveries were 71.8% and 77.9% respectively. The silver recoveries were 67.5% and 74.3%, respectively. The cyanide was 0.875 kg/t and 0.65 kg/t respectively. Lime consumption was about 1.5 kg/t in both cases. For the hematite sample the gold recoveries were 63.6% and 72.3% respectively. The cyanide consumption was about 0.65 kg/t and the lime consumption was 3.8 kg/t.

Flotation gold recoveries for grinds at 30% +150 µm and 27% 75 µm were 81.1% and 90.3%, silver recoveries of 68.1% and 96.9% respectively. The concentrates were subjected to regrind and 72-hour cyanidation of the flotation concentrates yielding overall gold recoveries of 63.5% and 74.5%, and overall silver recoveries of 61% and 51% respectively. These overall recoveries were lower than those achieved by McClelland while the flotation recoveries were similar.

The conclusions drawn were that finer grind gave better recoveries and that heap leach may not be a suitable option for the processing of this material, though they did suggest it should be investigated further.

Flotation followed by regrind and cyanidation of the concentrate was suggested as the process for the sulphide mineralogy.

13.8 ATHENA GOLD INCORPORATED – TALAPOOSA GOLD PROJECT: PROJECT INTRODUCTION REPORT – JULY 1991

In this report, the author discusses some column leach test work performed for Placer Dome by Barringer Laboratories. The test samples were oxides from the East Dyke, Dyke Adit and North Dyke zones. Possible column leach gold recoveries in the 75 to 80% range for the East Hill and North Dyke samples, and 65 to 70% gold recoveries for the Dyke Adit samples led to the suggestion that further work be done with respect to heap leach as a process option.

13.9 PEGASUS GOLD INC. – PHASE I TO III – 1993

Pegasus reviewed the historical test work, and decided that the best method for processing the material at Talapoosa would be heap leach. They came to this conclusion based on the mineralogical data suggesting the presence of electrum which dissolves slower in contact with cyanide, and the fact that heap leach facilities are generally lower capital and operating cost. They also investigated generating a precious metal bearing pyrite flotation concentrate which could possibly be smelted to obtain the precious metals, or which could be leached. Further investigation involved pre-treatment by bio-oxidation of column leach feed.

The metallurgical test work was carried out in three phases. Each phase created a new set of test composites and tested a different process option. All three phases have been summarized in the description below.

Phase I of the test work was carried out on five composites as presented in Table 13.20. It was believed these samples were representative of the respective zones. Composite 1 had 36 intervals (55 m) within the higher grade section of the Main Zone. Composite 2 utilized 33 intervals (50 m) within the lower grade section of the Main Zone. Composite 3 contains 131 intervals (200 m) and represents the low-grade section of the Bear Creek Zone. Composite 4 utilizes 59 intervals (90 m) to represent the lower grade section of the Bear Creek Zone. Composite 5 utilizes 69 intervals (105 m) to represent another high-grade section of the Bear Creek Zone.

Table 13.20 - Pegasus Phase I – Composites Recipes and Head Assays

Composite	Drillholes	Description	Au (g/t)	Ag (g/t)
1	PM-1,2,3	Main	1.95	16.8
2	PM-1,2,3	Main–LG	0.48	5.83
3	PM-4,5,6,6A,7,8	Bear Creek-LG	0.69	11.6
4	PM-4A,5,6A,7,8	Bear Creek Low SiO2	2.06	17.8
5	PM-4,4A,6A	Bear Creek High SiO2	2.19	22.3

Source: Pegasus (1992)

McClelland 1989 – Column Leach Results – Part 1. These composites were subjected to column leach and flotation tests. Composites 1 and 2 were completed in duplicate, and composites 3, 4, and 5 were done in single trials. Composites were agglomerated using 3.5 kg/t of lime. All samples were crushed to 100% 6.35 mm. The results from the column leach tests are presented in Table 13.21.

Table 13.21 - Pegasus Phase I – Column Leach Results

Composite	Weeks Leached	Status	Au Recovery (%)	Ag Recovery (%)
1A	59	Complete	74	49
1B	53	Complete	74	56
2A	5	Complete	75	39
2B	5	Complete	83	41
3	52	Complete	40	34
4	67	Incomplete	62	46
5	67	Incomplete	55	56

Source: Pegasus (1992)

These results confirm the results from previous test work that the Main Zone appears to be amenable to heap leach and the Bear Creek Zone does not show the same amenability. The leach times were very long which is typical for gold ores containing electrum.

Flotation was also tested in Phase I by Montana Tunnels Mining Laboratory (Montana Tunnels). The flotation tests were carried out on Composite 3, 4, and 5 samples. These tests were completed to determine if a marketable concentrate could be produced. The head assays for the composites have been compiled in Table 13.22.

Table 13.22 - Pegasus Phase I – Head Assays for Flotation Test Composites

Composite	Au (g/t)	Ag (g/t)	Fe (%)
3	0.55	12.7	2.48
4	3.70	14.7	2.52
5	1.99	32.6	1.50

Source: Montana Tunnels (1992)

A grind size of 80% passing -75 µm was targeted. The first set of flotation tests (Test #1) did not hit this target and were in the 67 to 58% passing -75 µm range. The remaining test were in the 79 to 80% passing -75 µm range. Below are descriptions of the details of each test; Table 13.23 is a tabulation of the flotation results for each test. Tests #3, #4, and #5 utilized a composite created from mixing Composites 3, 4, and 5.

- Test #1 – Bulk/scavenger flotation tests with low reagent dosages.
- Test #2 – Cleaner flotation tests low reagent dosages.
- Test #3 – Cleaning stage with longer flotation times and increased stage added reagent dosages.
- Test #4 – Similar to Test #3 but using regrind prior to cleaning.
- Test #5 – Same as Test #5 with shorter float times, shorter regrind time, and lower reagent dosages.

Table 13.23 - Pegasus Phase I – Montana Tunnels Flotation Results

Test	Rougher Recovery (%)				Cleaner Recovery (%)			
	Wt %	Au	Ag	Fe	Wt %	Au	Ag	Fe
#1*: Composite 3	3.5	78.0	73.1	35.9	2.0	67.3	61.1	21.5
#1*: Composite 4	2.8	69.8	55.8	31.3	1.4	55.1	38.9	17.6
#1*: Composite 5	2.2	82.0	75.9	30.8	1.2	65.3	61.1	19.4
#2: Composite 3	4.0	76.2	73.8	38.5	2.0	63.8	59.3	29.3
#2: Composite 4	3.5	60.0	59.4	33.7	1.6	49.5	45.8	25.6
#2: Composite 5	2.5	85.8	43.4	29.9	1.0	80.1	38.4	23.1
#3: Composite 3+4+5	11.1	94.0	72.4	52.8	2.6	85.2	59.8	37.5
#4: Composite 3+4+5	11.9	93.8	90.2	56.1	3.1	82.9	76.3	33.6
#5: Composite 3+4+5	7.9	90.0	68.0	47.5	1.5	75.7	54.3	23.0

Note: *Bulk + Scavenger Concentrate

Source: Montana Tunnels (1992)

Panning of the flotation concentrate yielded electrum. The suggestion was made that gravity concentration be tested to try and remove the electrum prior to leach or flotation. Microscopic analysis of the concentrate also revealed iron-silica and iron-gangue middlings which could act to lower the concentrate grade by being collected into the concentrate. From the flotation results it would appear that a reasonable primary grind for rougher flotation followed by a finer grind prior to cleaner flotation will offer the best recovery.

Inductively coupled plasma (ICP) analysis was run on the collected cleaner concentrates. The results are presented in Table 13.24.

Table 13.24 - Pegasus Phase I – Analysis of Flotation Cleaner Concentrates

Element	Assay	Method
Al	1.6%	ICP
Sb	310 ppm	ICP
As	9,000 ppm	ICP
Ba	230 ppm	ICP
Bi	<50 ppm	ICP
Cd	<5 ppm	ICP
Ca	0.15%	ICP
Cr	<25 ppm	ICP
Co	120 ppm	ICP
Cu	540 ppm	ICP
Fe	15%	ICP
Pb	290 ppm	ICP

Element	Assay	Method
Mg	0.25%	ICP
Mn	87 ppm	ICP
Mo	78 ppm	ICP
Ni	140 ppm	ICP
P	660 ppm	ICP
K	1.4%	ICP
Si	31%	ICP
Na	0.07%	ICP
Sr	28 ppm	ICP
Sn	<75 ppm	ICP
Ti	830 ppm	ICP
W	530 ppm	ICP
V	<5 ppm	ICP
Zn	1,200 ppm	ICP
As	0.56%	FLAA
Sb	0.03%	FLAA
Bi	<2 ppm	FLAA
Cd	<0.001%	FLAA
Hg	31 ppb	CVAA

Note: FLAA = Flameless Atomic Absorption; CVAA = Cold Vapour Atomic Absorption

Source: Montana Tunnels (1992)

The only penalty element in the concentrate is the arsenic which is at 0.56% for these samples. The American Smelting and Refining Company (ASARCO) in Helena, Montana, was asked if the cleaner concentrate was acceptable feed to their smelter and they indicated it would be acceptable. One of the conclusions drawn by the Montana Tunnels author is that the cost of a flotation mill and cyanide leach combination process may have been prohibitive at the time of the test program.

Three new composite samples were formed for Phase II test work. The Lower Bear Creek (LBC), Upper Bear Creek (UBC), and Main Zone (Main) composites as presented in Table 13.25 were created.

Table 13.25 - Pegasus Phase II – Composite Details

Composite	Origin	Description	Au (g/t)	Ag (g/t)
LBC	Core	312 m, 5 holes	1.17	22.6
UBC	Cuttings	136 m, 11 holes	1.34	35.6
Main	Bulk	At Surface	1.85	42.2

Source: Pegasus (1993)

Four column leaches at different crush sizes (i.e. 38.1 mm, 19.05 mm, 9.52 mm, and 6.35 mm) were set up with Main Zone composite to determine the maximum particle size that will offer the optimal precious metal recovery. One column was set up on the UBC composite with no size reduction to determine the "as-received" particle size precious metal recovery. Three columns were set up for the LBC composite. One column was -6.35 mm agglomerated, the second -6.35 mm non-agglomerated, and the third was agglomerated -3.36 mm. The -3.36 mm material was crushed in a Barmac impact crusher. A split of Composite #3 from Phase I was subjected to bio-oxidation and subsequent column leach. The results from these column leach tests are presented in Table 13.26.

Table 13.26 - Pegasus Phase II – Column Leach Test Results

Sample	Weeks Leached	Au Recovery (%)	Ag Recovery (%)
Main, 38.1 mm	48	57	37
Main, 19.05 mm	48	90	61
Main, 9.52 mm	48	79	62
Main, 6.35 mm	48	80	66
UBC, approx. 6.35 mm	17	58	56
LBC, -6.35 mm, Aggregate	33	46	42
LBC, -6.35 mm, No-Aggregate	33	47	44
LBC, -3.36 mm, Aggregate	33	52	48
Phase I, Composite 3, BIOX	11	49	50

Source: Pegasus (1993)

The second largest column leach particle size for the Main composite, 19.05 mm, gave a high gold recovery. The agglomeration of the LBC sample did not appear to have a significant effect on the precious metal recoveries. The finer crush to 3.36 mm did create a marked increase in the precious metal recoveries for the LBC sample. The pre-treatment by bio-oxidation also showed an improvement in the subsequent column leach recoveries for the Phase I, Composite 3 sample.

There were crusher tests also completed in the Phase II work. Allis Minerals Systems completed crusher impact tests and abrasion tests on Main Zone samples which the geologists had agreed was the hardest material on the Property. The crusher impact index (9.98 kWh/t) was average and the abrasion index was high (0.44). The high abrasion index indicates high wear of parts. The Nordberg HP series crusher and the Barmac from Rock Engineered Machinery Co. Inc. (REMCO) both proved that they could bring the particle size down to the goal size of 3.35 mm.

A new composite representing the LBC mineralization was assembled for Phase III. The composite was compiled from intervals from PE-30, PE-31, PE-32, and PE-37. Splits were taken to test -6.35 mm agglomerated column leach, -6.35 mm bio-oxidation then column leach, and a sample crushed to 3.36 mm by Barmac crusher, agglomerated, and then column leach. The results for these tests were not presented in the report, but it was indicated that again the sample crushed to 3.36 mm showed a marked increase in precious metal recovery.

Since the 3.36 mm crushed material resulted in higher recoveries, heap stability was investigated for this particle size range since this is smaller than the conventional crush size for a heap leach. Welsh Engineering Science and Technology (Westec) was asked to complete this investigation on behalf of Pegasus. Westec completed permeability and compression tests, as well as a site reconnaissance to determine that the material has heap stability to 30 m and can still maintain the permeability to drain the leach solution through the stack up to 90 m. This permeability also did not appear to deteriorate over a 10-day period.

A bottle roll test was completed on the LBC sample with 96 hour cyanide contact. The gold recovery was disappointing at 24.1%.

Finally flotation/cyanidation tests were completed on the LBC composite. The feed to flotation was 80% passing -75 µm. Samples were subjected to rougher flotation and the rougher concentrate was subjected to agitated cyanidation for 42 to 48 hours. The results from these tests have been summarized in Table 13.27.

Table 13.27 - Pegasus Phase III – Flotation / Cyanidation Results

Test	Flotation Recovery (%)		Cyanidation Recovery (%)		Overall Recovery (%)	
	Au	Ag	Au	Ag	Au	Ag
1	84.3	70.1	87.3	70.7	73.6	49.6
2	89.3	79.0	84.5	71.1	75.4	56.2

Source: Pegasus (1993)

These recoveries were low. It was believed these recoveries could not improve without oxidation of the flotation concentrate.

Projections of metal recoveries for the oxides and sulphides by heap leach with no oxidation techniques were made. The oxides were predicted to be 78% for gold and 55% for silver, and for sulphides 60% gold and 50% silver. To achieve these the top size for the oxides would need to be less than 19 mm and lime would need to be added to the heaps at 2.5 kg/t to maintain the proper alkalinity during leaching. The sulphides would need to be crushed to 100% -3.36 mm. The suggestion was that for significant improvements to the precious metal recoveries, oxidation techniques would need to be employed prior to leaching.

13.10 PEGASUS GOLD INC. / PITTSBURGH MINERAL AND ENVIRONMENTAL TECHNOLOGY INC. – MARCH 1993

Pegasus sent samples (through McClelland) for Pittsburgh Mineral and Environmental Technology Inc. (PMET) to analyze to determine the following:

- Overall mineralogical sample composition;
- Mode of occurrence of gold and silver;
- Particle size and gold distribution;
- Liberation/locking characteristics of gold and gold bearing sulphides;
- Determination of the reason for slow/low gold extractions in sulphide material types;

- Determination of factors critical to optimizing precious metal recovery (e.g. composition and amounts of slimes, cyanicides, scale-forming, minerals, potential mineral "preg robbing", reasons for refractories other than sulphide encapsulation).

The tests were done on Main Zone bulk material, Upper Bear Creek cuttings and Lower Bear Creek sulphide. The goals of the test work were achieved using x-ray diffraction, gravity separation, optical microscopy, scanning electron microscopy – energy dispersive x-ray spectroscopy (SEM-EDX), screen analyses and photomicrography.

The Main Zone sample was siliceous and had high iron oxidation. It also showed slightly elevated antimony levels (100 ppm). Gold and silver assays were also higher for these samples. The majority of the gold occurs in the +325 µm particle size range although there are high concentrations found in the -74 µm size range. The gold occurs as silver rich (approximately 20% silver) native gold. This gold/silver can also occur as electrum which often exhibits slower dissolution rates in cyanide. Some of the gold had iron oxide or copper sulphide coatings which would also deter dissolution by cyanide. It was estimated that 20% of the gold would not respond to leaching due to sulphide refractories. Another 30 to 40% may not respond due to siliceous gangue locking the gold particles away from the cyanide lixiviants. The Main Zone sample gravity pre- concentrations test work showed that this sample was not amenable to pre- concentration by gravity separation.

The Upper and Lower Bear Creek samples showed elevated barium (500 to 1,000 ppm) and titanium (5,000 ppm) contents and slightly elevated manganese (100 ppm). There were also slightly high elevations of base metals (copper, lead and zinc) in the 100 to 300 ppm range. The gold and silver assays were also lower for these samples. Carbon concentrations were low for all three samples. The majority of the gold is still found in the +325 µm particle size range, but the gold in the -74 µm size range is higher for the Bear Creek samples than the Main Zone. The gold is finely disseminated within sulphide minerals. It was suggested that fine grinding will be required to extract the precious metals. The Bear Creek samples showed that pre-concentration by gravity techniques may be effective.

13.11 MCCLELLAND LABORATORIES INC. REPORT TO ATHENA – JULY 1994

McClelland submitted a letter report along with tables of results for metallurgical test work completed on five Talapoosa composites. The letter report also contained the results for 400 HQ diamond drill core interval samples submitted for bulk density analysis, interval crushing, and assaying. The five metallurgical composites were created from these samples submitted for bulk density analysis. Two composites which were representative of the Main Zone were created as well as three composites representative of the Bear Creek Zone. The composites were all reduced in size to 100% passing -5 mm. Bottle roll and column leach test work was completed in duplicate on the Main Zone composite. The Bear Creek Composite 3 was subjected to a single bio-oxidation / heap leach cyanidation. The remaining Bear Creek samples were subjected to bottle roll and column leach.

The bottle roll results are presented in Table 13.28. The column leach results for the Main Zone composites can be found in Table 13.29. The column leach results for the Bear Creek Zone composites are shown in Table 13.30. Table 13.31 presents the results of the direct cyanidation and bio-oxidized/cyanidation. The direct cyanidation results are the same as the Bear Creek Composite 3 results from Table 13.30. They are reiterated for comparison.

The Main Zone results show that the samples are somewhat amenable to agitated cyanidation. They also had moderate lime and cyanide consumptions. The Bear Creek samples did not appear to be amenable to direct cyanidation. The reagent consumption was low for these samples.

Similarly for the column leach results, the Main Zone composites appeared to be more amenable to column leach than the Bear Creek samples. Reagent consumptions were high for both Main and Bear Creek samples.

The bio-oxidation pre-treatment of Bear Creek Composite 3 did show an increase in precious metal recovery. The lime consumption remained the same but there was a significant increase in the cyanide consumption for the bio-oxidation pretreated column leached sample.

Table 13.28 - McClelland 1994 – Bottle Roll Leach Results

Sample	Head Assay (g/t) Calculated Au	Calculated Ag	Assayed Au	Assayed Ag	Extraction (%) Au 2 h	Au 6 h	Au 24 h	Au 48 h	Au 72 h	Au 96 h	Ag 2 h	Ag 6 h	Ag 24 h	Ag 48 h	Ag 72 h	Ag 96 h	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
Main Zone 1	1.71	18.5	1.95	17.5	27.2	37.6	47.6	54.6	57.6	58.0	12.0	14.8	19.1	20.6	22.6	24.1	0.84	3.7
Main Zone 1 (Duplicate)	1.89	19.5	1.95	17.5	27.1	36.5	46.5	50.7	54.0	56.4	12.3	14.9	19.6	21.4	22.6	24.6	0.81	3.8
Main Zone 2	0.41	6.51	0.48	5.83	36.7	45.8	63.3	66.7	66.7	66.7	14.2	16.8	21.6	23.7	25.3	26.3	0.35	3.35
Main Zone 2 (Duplicate)	0.48	6.17	0.48	5.83	34.3	42.9	55.0	61.4	62.1	64.3	15.0	17.8	22.2	25.0	26.7	27.8	0.525	3.3
Bear Creek Composite 3	0.86	12.7	0.69	13.4	7.2	10.8	14.4	18.4	22.4	28.0	6.5	8.9	16.5	20.5	23.0	24.3	0.45	1.05
Bear Creek Composite 4	3.12	19.2	2.19	18.5	3.8	6.8	20.0	29.2	36.8	41.8	5.2	7.3	13.9	18.4	21.4	23.2	0.51	1.15
Bear Creek Composite 5	2.02	22.6	2.19	22.3	5.9	11.4	22.7	33.7	36.9	37.3	5.2	7.6	15.3	20.3	23.3	25.8	0.29	0.95

Source: McClelland (1994)

Table 13.29 - McClelland 1994 – Main Zone Composites Column Leach

Sample	Head Assay (g/t) Calculated Au	Calculated Ag	Assayed Au	Assayed Ag	Extraction (%) Au 5 d	Au 34 d	Au 50 d	Au 200 d	Au 300 d	Au 398 d	Ag 5 d	Ag 34 d	Ag 50 d	Ag 200 d	Ag 300 d	Ag 398 d	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
Main Zone 1	1.85	18.9	1.95	17.5	48.1	63.1	65.4	70.4	72.0	72.2	22.0	34.0	35.8	42.4	44.2	45.5	3.6	3.5
Main Zone 1 (Duplicate)	1.92	17.8	1.95	17.5	50.5	66.1	67.7	71.6	72.9	73.2	24.6	37.9	39.8	46.7	48.5	50.0	3.8	3.5
Main Zone 2	0.41	6.17	0.48	5.82	63.3	75.0	75.0	-	- -	75.0	27.2	38.3	38.9	-	-	38.9	1.1	3.5
Main Zone 2 (Duplicate)	0.41	5.83	0.48 559	5.83	66.7	83.3	83.3	-	-	83.3	28.2	40.6	41.2	-	-	41.2	1.2	3.5

Source: McClelland (1994)

Table 13.30 - McClelland 1994 – Bear Creek Zone Composites Column Leach

Sample	Head Assay (g/t)				Extraction (%)												NaCN Consumption (kg/t)	Lime Consumption (kg/t)
	Calculated		Assayed		Au						Ag							
	Au	Ag	Au	Ag	5 d	15 d	101 d	200 d	400 d	601 d	5 d	15 d	101 d	200 d	400 d	601 d		
Bear Creek Composite 3	0.65	13.0	0.67	13.4	16.8	27.4	35.8	36.8	-	36.8	10.8	19.5	32.1	36.1	-	39.5	2.1	3.5
Bear Creek Composite 4	2.16	16.4	2.19	18.5	19.8	34.8	51.0	54.8	59.0	61.9	13.3	21.9	35.0	39.4	42.9	45.8	3.8	3.56
Bear Creek Composite 5	2.54	23.0	2.54	23.0	18.6	28.2	38.1	40.5	43.4	44.6	10.7	21.0	38.1	45.2	51.2	59.7	3.6	3.5

Source: McClelland (1994)

Table 13.31 - McClelland 1994 – Bear Creek Zone Composite 3 – Direct Cyanidation and Bio-oxidation / Cyanidation

Sample	Head Assay (g/t)				Extraction (%)												NaCN Consumption (kg/t)	Lime Consumption (kg/t)
	Calculated		Assayed		Au						Ag							
	Au	Ag	Au	Ag	5 d	40 d	73 d	115 d	245 d	301 d	5 d	40 d	73 d	115 d	245 d	301 d		
Direct Cyanidation	0.65	13.0	0.67	13.4	16.8	33.7	35.8	35.8	36.8	36.8	10.8	25.8	30.3	33.2	37.1	39.5	2.1	3.5
Bio-oxidized Residue	0.69	12.3	0.69	13.4	32.0	46.0	51.0	54.5	55.0	55.0	33.0	44.3	46.8	48.8	52.1	52.8	3.5	3.56

Source: McClelland (1994)

13.12 DAWSON METALLURGICAL LABORATORIES – REVIEW OF PREVIOUS TEST WORK AND SUGGESTIONS FOR NEW TEST WORK – 1994

Dawson was asked to complete a review of the previous test work and suggest new test work that should be completed. The author reviewed the previous test work results completed by both Dawson and others. The author then made a few suggestions for future test work. The focus was on heap leaching and trying to determine the best process options for the Bear Creek Zone since it has proven to be the most difficult to process in test work to date. Below is a summary of some of the suggested testing options.

MAIN ZONE

- → Complete baseline column leach tests on 19 mm crush size.
- → Determine if unleached gold from the Main Zone is associated with sulphides.
- → Evaluate the use of cyanide and leach aids in the agglomeration of column leach feed to improve leach kinetics.
- → Determine if the cyanide consumption could be reduced through the use of a lower dosage of cyanide for the test work.

BEAR CREEK

- → Complete baseline column leach tests on 6.35 mm and 3.35 mm crush sizes.
- → The degree of gold sulphide association should be determined through further mineralogical work.
- → Evaluate the use of cyanide and leach aids in the agglomeration of column leach feed to improve leach kinetics.
- → Determine if the cyanide consumption could be reduced through the use of a lower dosage of cyanide for the test work.
- → Further investigations into bio-oxidation to improve leach kinetics and maximize precious metal recoveries.
- → Test work on a split flow process where the crushed ore is screened to remove the slimes (300 to 500 µm). The coarse fraction could then be heap leached, and the fine fraction could be either agitated leached or subjected to flotation. The flotation concentrate could then be subjected to a finer grind and then put in for agitated cyanidation to extract the precious metals. This would only be viable if the sulphides are found to be in the fine fraction.

13.13 SUMMIT VALLEY EQUIPMENT & ENGINEERING INC. – FEASIBILITY STUDY 1995

Summit Valley Equipment & Engineering Inc. (Summit Valley) created a feasibility study which costed out a heap leach facility. The heap leach would produce 11,350 L/min of pregnant solution. The pregnant solution would be clarified and deaerated prior to a zinc precipitation to extract the gold. The precipitate would then be acid washed, filtered on a filtered press, mercury retorted, and then fed to the doré furnace.

The design is based on the review of previous metallurgical work completed by Dawson (1994) focusing on the work by Pegasus. The costing, sizing, calculations and flowsheets are included in the feasibility report.

13.14 DAWSON METALLURGICAL LABORATORIES INC. – 1995

Dawson was contracted to reconfirm the column leach results from the Pegasus column leach work and to optimize and improve the gold leach kinetics. Specific focus was given to the Bear Creek Zone. Fresh sample from new drillholes were employed for the test work. Specifically for the Main Zone, the work included:

- Confirm previous gold extractions at crush sizes of 19.05 mm, 12.7 mm, and 6.35 mm, using a lower dosage of cyanide solution at 0.25 kg/t.
- Improve gold leach kinetics by agglomerating the feed with cyanide prior to the column leach.

For the Bear Creek Zone, the objectives were as follows:

- Confirm the 3.36 mm crush requirement to achieve a 50% gold recovery from a 0.9 to 1.2 g/t gold head grade.
- Find an appropriate device to reduce the feed to the required 3.36 mm.
- Determine if agglomerating with cyanide will increase the leach kinetics.
- Investigate leach aids that may increase the gold leach kinetics.

Miramar advised Dawson that a 55% gold recovery should be the target based on a 1 g/t head grade. New drill core specifically for the metallurgical test program were drilled. There were three cores from the Main Zone and five from the Bear Creek Zone. The head assays for the composites are presented in Table 13.32.

Table 13.32 - Dawson 1995 – Head Assays Main Zone and Bear Creek Composites

Composite	Head Assay			
	g/t		Wt%	
	Au	Ag	Sulphide	Fe
Main Zone	0.93* 1.13** 0.79 – 1.34***	12.3	0.08	2.19
Bear Creek No. 1	0.89* 0.79** 0.58 – 1.1***	5.49	1.36	2.88
Bear Creek No. 2	1.03* 0.99** 1.20***	9.94	1.37	3.32

Notes: * Calculated form individual footages.
** Assayed head.
*** Range of back-calculated head assays from test work.

Source: Dawson (1995)

Some gravity concentration work was completed on the Main Zone and Bear Creek composites at a grind of 67% passing -75 µm. The tests indicate that approximately 18% of the Main Zone sample is available as free milling gold and 28% of the Bear Creek sample.

The results from the Main Zone composite column leach tests have been summarized in Table 13.33.

Table 13.33 - Dawson 1995 – Main Zone Composite Column Leach Results

Crush Size (mm)	Leach Days	Au Recovery* (%)	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
19.05	59	49.1	0.42	5.8
12.7	59	39.8	0.4	5.2
6.35	59	47.5	0.40	5.2

Note: * Estimate based on 1.13 g/t gold head grade.

Source: Dawson (1995)

The gold extraction appears to be independent of the crush size based on the results from these samples. The leaches were completed at lower dosages of cyanide and lime so the consumptions of these reagents were also lower.

The column leach test results for the Bear Creek No. 1 composite are presented in Table 13.34. These tests utilized different pieces of equipment to achieve the crush sizes tested. The finer crush size did show an improvement in gold recovery in this case. The 6.35 mm crush size gave a recovery of 42.5% gold, and all 3.36 mm crush samples had recoveries over 49.2% gold.

Table 13.34 - Dawson 1995 – Bear Creek Composite 1 Column Leach Results – Different Size Reduction Equipment

Crush Size	Crush Type	Leach Days	Au Recovery (%)	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
6.35 mm	Jaw	28	42.5	0.58	1.15
3.36 mm	Fast Rolls*	63	51.2	0.595	1.15
3.36 mm	VSI	40	49.2	0.59	1.15
3.36 mm	HPGR	40	56.6	0.625	1.15
3.36 mm	HPGR + LA	40	64.3	0.615	1.15

Notes: * At Dawson Metallurgical Laboratories.
VSI = Vertical Shaft Impact crusher; LA = Leach Aid manufactured 3M.

Source: Dawson (1995)

The HPGR gold leach recovery increased by 7.7% through the use of 0.08 kg/t of the 3M Specialty Chemicals leach aid.

Bottle roll tests were completed on Main Zone composite samples to review the gold extraction kinetics at different crush/grind sizes. The results from these tests are presented in Table 13.35.

Table 13.35 - Dawson 1995 – Main Zone Composite Bottle Roll Results – Varied Crush / Grind Sizes

Test No.	Crush / Grind Size	Leach Days	Au (g/t)		Au Recovery (%)	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
			Residue	Head			
1	-25.4 mm	5	0.82	1.13	29.0	0.26	1.85
2	-19.05 mm	5	0.86	1.234285	29.4	0.26	1.85
3	-12.7 mm	5	0.93	1.337	31.1	0.32	2.1
4	-6.35 mm	5	0.31	0.960	67.0	0.31	2.2
A	-841 µm	3	0.41	1.029	61.3	1.29	1.9
22	67% -74 µm	3	0.21	0.857	77.8	0.36	2.05

Source: Dawson (1995)

Bottle roll kinetic analysis of the coarse crush sizes (-12.7 mm and larger) showed that the gold extraction was slow and that the gold was still dissolving at the end of the fifth day of leaching. The recoveries for these samples were also quite low. The gold leach kinetics were quicker for the finer crush or grind sizes and the gold recoveries were also over 60%. The lime and cyanide consumptions were also quite low.

Similar bottle roll tests were carried out on the Bear Creek No.1 composite at crush sizes of -6.35 mm, -3.36 mm, and -2.0 mm. The leach kinetics were very slow with poor gold extractions (25% or less gold recovery). The gold was still leaching after 120 hours. A second set of tests were performed which agglomerated 0.5 kg/t of sodium cyanide and 1 kg/t of lime. The agglomerates were allowed to cure for 72 hours. The results are presented in Table 13.36.

Table 13.36 - Dawson 1995 – Bear Creek No. 1 Composite Bottle Roll Tests – Varied Crush / Grind Size and Crush Equipment

Test No.	Crush/Grind Size	Leach Days	Au (g/t)		Au Recovery (%)	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
			Residue	Head			
14	-6.35 mm	10	0.41	0.58	27.9	0.89	1.2
15	-3.36 mm (DML)	10	0.58	0.79	27.2	1.155	1.25
19	-3.36 mm (HPGR-SP)	10	0.34	0.65	45.2	0.885	1.25
20	-3.36 mm (HPGR-DP)	10	0.38	0.75	51.4	0.915	1.25
17	-2.0 mm (DML)	10	0.41	0.69	38.1	1.015	1.3
B	-841 µm (DML)	3	0.31	0.72	54.8	0.795	0.9
21	67% -74 µm	6	0.27	0.72	60.8	0.48	1.05

Notes: DML= fast rolls at Dawson; HPGR-SP = high-pressure grinding rolls – single pass; HPGR-DP – high-pressure grinding rolls – double pass

Source: Dawson (1995)

The finer particle size resulted in higher gold recovery, but the equipment used to reduce the particle size also seems to play a role. Comparing tests 15, 19, and 20, it can be seen that the HPGR single pass offered a greater gold recovery than the fast rolls for the same particle size, but the double pass through the HPGR gave a further increase in gold recovery for the same particle size. The finer grind of 67% passing -74 µm still gave the highest gold recovery at 60.8%.

Mineralogical analysis to determine the gold associated minerals in the residues from the Main Zone and Bear Creek 1 composites were completed. The results indicate that about two thirds of the unleached gold can be attributed to gold associated with sulphide and the remainder encapsulated in silicates.

Tests were in progress for agglomerated feed from Main Zone and Bear Creek No. 1 composites. The preliminary results were presented. These have been summarized in Table 13.37 and Table 13.38. The recovery results in these tables are simply estimates based on the assay head. Since these columns were still leaching the calculated head could not be determined until the end of the leach when the residue assay was determined.

The agglomeration recipe for the Main Zone composite was 0.25 kg/t sodium cyanide, 0.5 kg/t Type II cement, 4.5 kg/t lime, and 80 kg/t of water. The agglomeration recipe for the Bear Creek Composite No.1 was 0.5 kg/t sodium cyanide, 0.5 kg/t Type II cement, 1.6 kg/t lime, and 115 kg/t water.

Table 13.37 - Dawson 1995 – Agglomerated Main Zone Composite – Column Leach Results

Test No.	Crush Size (mm)	Leach Days	Au Predicted Recovery (%)*	Ag Predicted Recovery (%)**	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
28	-19.05	59	49.1	32.3	0.42	5.8
29	-12.7	59	39.8	25.9	0.4	5.2
30	-6.35	59	47.5	30.1	0.41	5.2

Notes: * Based on 1.13 g/t gold head.
** Based on 12.3 g/t silver head.

Source: Dawson (1995)

Table 13.38 - Dawson 1995 – Agglomerated Bear Creek No. 1 Composite – Column Leach Results

Test No.	Crush /Grind Size (mm)	Leach Days	Au Predicted Recovery (%)*	Ag Predicted Recovery (%)**	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
45	6.35 (Jaw)	28	42.5	17.6	0.58	1.15
25	-3.36 (DML)	63	51.7	25.8	0.595	1.15
35	-3.36 (VSI)	40	49.2	27.2	0.59	1.15
37	-3.36 (HPGR)	40	56.6	36.9	0.625	1.15

Notes: * Based on 0.79 g/t gold head for Tests 45, 25, and 35; 0.89 g/t gold head for Test 37.
** Based on 8.9 g/t silver head.

Source: Dawson (1995)

The gold recovery for the Main Zone column leach are lower than previous test work. It was suggested that this could be due to the lower head grade used in these tests (i.e. 1.13 g/t gold) and the lower cyanide dosage (i.e. 0.25 kg/t versus previously used 1 kg/t). Further test work at a higher cyanide dosage is planned, and a Main Zone sample will also be subjected to size reduction by HPGR to -3.36 mm.

As seen previously, the gold extraction in the Bear Creek No.1 samples are dependent on the crush size and the equipment used to achieve the crush size. The best gold recovery (56%) was achieved for -3.36 mm with the HPGR, as shown in the bottle roll test work. The leach kinetics has also increased due to the addition of the sodium cyanide in the agglomeration. The effect of sodium cyanide agglomeration (and leach aid) were tested with the Bear Creek Composite No. 1 sample. The results are presented in Table 13.39.

Table 13.39 - Dawson 1995 – Effect of Agglomeration with Cyanide and Leach Aid on Column Leaching at -3.36 mm Bear Creek Composite No. 1

Test No.	NaCN Addition to Agglomeration (kg/t)	Estimated Au Extraction (%)*		
		6 d	31 d	40 d
36	0	25.4	43.5	48.1
37	0.5	45.7	55.7	56.6
38	0.5 + 0.08 L.A	53.1	62.7	64.3

Notes: * Based on 0.89 g/t Au head.
L.A = Leach Aid manufactured by 3M.

Source: Dawson (1995)

Both the leach aid and the sodium cyanide seem to work together to increase the gold leach kinetics and the gold recovery. The samples tested were -3.36 mm which were reduced in size by HPGR.

The HPGR has shown that it can improve the gold leach kinetics for these samples as well. This is possibly due to microfracturing of the material as well as the generation of fines. HPGR generated 46% passing -150 µm, compared to 20% from fast rolls and 13% from the VSI. Screen analyses of the leach product from these pieces equipment are shown in Table 13.40.

Table 13.40 - Dawson 1995 – Screen Analysis of -3.36 mm Leach Products

Test No.	Crusher Type	Estimated Au Extraction (%)		
		-2.0 mm	-500 µm	-150 µm
25	DML fast rolls	71.6	31.1	20.2
35	VSI	67.8	23.9	12.6
37	HPGR	87.0	60.7	45.5

Source: Dawson (1995)

13.15 JBR ENVIRONMENTAL CONSULTANTS – ENVIRONMENTAL IMPACT STATEMENT – 1996

In 1996, the final environmental impact statement (EIS) which was issued and contained a description of the proposed process facility for Talapoosa. The proposed process was a valley fill high density polyethylene lined leach pad, which would have the capacity for 38 Mt of material. The solution ponds would be double lined and ponds and pad would have leak detection. An overflow pond would be situated down grade from the pregnant solution pond and all surface flow would be directed around the heap.

Crushed material (four stages of crushing) would be mixed with lime, cement, and dilute cyanide solution and placed on the pads via conveyor. Run-of-mine would be direct dump by truck. The heap would be leached with dilute cyanide solution drip irrigated onto the heap. The pregnant solution would be collected in the pregnant solution pond for storage prior to processing for extraction of the gold from solution (Merrill Crowe plant – zinc precipitation). Once the gold was extracted from solution, the barren solution would be returned to the barren solution pond for storage prior to being reintroduced to the heap.

13.16 FLUOR DANIEL WRIGHT 1996 – TECHNICAL ECONOMIC REVIEW

Fluor Daniel Wright completed a technical review for the Project based on metallurgical testing by Dawson in 1996. The designs were based on mining and processing only the oxide zone material. The development of the more difficult sulphide zones would follow. The plan was to reuse equipment from the Golden Eagle Mine at the Talapoosa operations to lower capital costs. The planned production rate was 14,500 t/d. Power would be generated by diesel generator sets.

The flowsheet is set up to be crush (two vertical shaft impactor crushers), valley fill heap leach (4.5 to 8 m lifts), pregnant solution, emergency and barren solution ponds. The gold will be removed in a Merrill Crowe process plant. This is the same process described in the EIS.

13.17 DAWSON METALLURGICAL LABORATORIES – FEBRUARY 1997

Dawson was contracted to complete further test work on Talapoosa samples. RC drill cutting samples were taken from the UBC (0.86 g/t gold head assay) and Dyke Adit zones (1.13 g/t gold head assay). The test work consisted of head assay, ICP scan, bottle roll tests, gravity concentration tests and column leach with agglomeration. For the UBC zone, 80 interval samples were taken from four drillholes (i.e. TAL-328, 329, 300, and 331). For the Dyke Adit, 39 interval samples were taken from ten drillholes.

For the column leach tests the samples were stage crushed to -6.35 mm. The UBC and Dyke Adit samples were 21% and 27% passing 150 µm respectively. The results from the column leach tests are presented in Table 13.41. The column feeds were agglomerated with 0.25 kg/t sodium cyanide, 2.5 kg/t Type II cement, and tests 93 and 94 were agglomerated with lime. Column residue screen analysis revealed that little of the gold in the -6.35 mm +3.36 mm particle size range was leached. Half of the gold in the -3.36 mm +150 µm particle size range leached out, and the majority of the gold in the - 150 µm range was leached. Electrum was identified in the UBC samples during characterization work on the head samples. There was also twice the free gold observed in the Dyke Adit samples as compared to the UBC.

The Dyke Adit composites had better gold recoveries than the UBC composite. Due to agglomeration with sodium cyanide and lime, the consumption of these items were low to moderate.

The next tests were 72-hour bottle roll tests at a crush size of -841 µm. The results are presented in Table 13.42. These were leaches done with carbon-in-leach (CIL) at a 1 kg/t sodium cyanide solution. The Dyke Adit composite again had higher gold recoveries as compared with the UBC sample.

Samples of Dyke Adit and UBC were subjected to a ball mill grind to -100 µm and were panned and amalgamated. The results from these tests can be found in Table 13.43. Some free milling electrum was found in the UBC composite in the 250 to 75 µm size range. Amalgamation measured 15% of the gold and 6% of the silver as free milling electrum. The sulphides which associated with the gold in the pan concentrates were mainly pyrite, but there was some bornite and galena. The Dyke Adit composite measured 27% of the gold and 2% of the silver as free milling electrum found in the same 250 to 75 µm size range with the same associated pyrite, bornite and galena.

The UBC and Dyke Adit composites were subjected to 240 hour bottle roll tests. The results from these tests are presented in Table 13.44. Since these samples seemed to have slow leach characteristics the test samples were agglomerated with 0.5 kg/t sodium cyanide, lime, and cured for three days prior to the tests. The Dyke Adit samples achieved higher gold recovery than the UBC samples. The UBC and Dyke Adit residues from these bottle roll tests were screened to determine what size ranges the gold was being leached from. The screen analysis with assays can be found in Table 13.45. As is expected, the higher gold extractions are at the finer particle sizes.

The residues from the column leach tests were also screened and compared in the same manner. The results of this analysis can be found in Table 13.46. Again, the majority of the gold was leached in the finer fraction of material.

Table 13.41 - Dawson February 1997 – Column Leach Summary

Test No.	Composite	Leach Days	Calculated Head (g/t)		Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumption (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
93	UBC	81	0.79	9.94	0.41	4.11	48.2	57.8	0.38	5.83	0.6	3.65
94	Dyke Adit	81	1.13	16.8	0.38	9.23	65.2	43.8	0.75	7.54	0.7	3.65
95	Dyke Adit	81	1.10	14.4	0.41	7.54	63.1	47.4	0.69	6.86	0.8	0.5

Source: Dawson (February 1997)

Table 13.42 - Dawson February 1997 – CIL Matrix Testing at 841 µm

Test No.	Composite	Leach Hours	Assay Head (g/t)		Calculated Head (g/t)		Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumption (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
H	UBC	72	0.86	5.49	0.72	9.23	0.24	0.67	69.8	92.6	0.48	8.57	0.59	2.06
I	Dyke Adit	72	1.03	14.1	1.17	15.8	0.31	4.8	73.6	70.1	0.86	11.0	1.17	2.06

Source: Dawson (February 1997)

Table 13.43 - Dawson February 1997 – Ball Mill Grind Product Gravity Hand Panning and Amalgamation Results

Test No.	Composite	Product	Wt %	Assay Head (g/t)		Distribution (%)	
				Au	Ag	Au	Ag
88	UBC	Amalgam Concentrate	-	-	-	15.3	6.1
		Amalgam Tail	2.1	1.34	29.83	3.0	6.7
		Gravity Concentrate	2.1	8.23	57.257	18.3	12.8
		Gravity Tail	97.9	0.79	8.571	81.7	87.2
		Total Calculated	100	0.96	9.600	100	100
		Total Assay	100	0.86	5.486	100	100
91	Dyke Adit	Amalgam Concentrate	-	-	-	27.1	2.5
		Amalgam Tail	3.1	N/A	N/A	-	-
		Gravity Concentrate*	3.1	~8.81	~13.7	27.1	2.5
		Gravity Tail	96.9	0.75	16.8	72.9	97.5
		Total Calculated	100	~0.99	~16.8	100	100
		Total Assay	100	~1.03	~14.1	100	100

Note: * The gravity concentrate for Test 91 is approximated. Bead from Test 91 Amalgam Tail was lost.

Source: Dawson (February 1997)

Table 13.44 - Dawson February 1997 – Bottle Roll Test at 6.35 mm Crush

Test No.	Composite	Leach Hours	Assay Head (g/t)		Calculated Head (g/t)		Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumed (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
89	UBC	240	0.86	5.49	0.93	15.4	0.45	7.89	53.5	49.4	0.48	7.54	0.66	2.86
92	Dyke Adit	240	1.03	14.1	1.2	19.9	0.358	11.0	68.2	45.2	0.82	8.91	1.0	2.51

Source: Dawson (February 1997)

Table 13.45 - Dawson February 1997 – Screen Analysis of Bottle Roll Test Residues

Size Fraction	UBC Test 89			Dyke Adit Test 92		
	Head (g/t)	Residue (g/t)	Extracted (%)	Head (g/t)	Residue (g/t)	Extracted (%)
-6.35 mm +3.36 mm	0.89	0.72	20	0.99	0.79	21
-3.36mm +2.0 mm	0.93	0.62	35	1.13	0.34	69
-2.0 mm +0.841 mm	0.86	0.45	49	0.89	0.41	54
-0.841 mm +0.5 mm	0.69	0.38	46	0.82	0.27	65
-0.5 mm +0.149 mm	0.86	0.24	71	0.89	0.31	64
-0.149 mm	1.13	0.17	85	1.30	0.27	80
Total	0.93	0.45	-	1.06	0.38	-

Source: Dawson (February 1997)

Table 13.46 - Dawson February 1997 – Screen Analysis of Column Leach Test Residues

Size Fraction	UBC Test 93			Dyke Adit Test 94			Dyke Adit Test 95		
	Head (g/t)	Residue (g/t)	Extracted (%)	Head (g/t)	Residue (g/t)	Extracted (%)	Head (g/t)	Residue (g/t)	Extracted (%)
-6.35 mm +3.36 mm	0.89	0.69	23	0.99	0.55	46	0.99	0.51	48
-3.36mm +2.0 mm	0.93	0.38	59	1.13	0.58	47	1.13	0.62	45
-2.0 mm +0.841 mm	0.86	0.51	42	0.89	0.41	53	0.89	0.38	57
-0.841 mm +0.5 mm	0.69	0.24	66	0.82	0.38	54	0.82	0.51	40
-0.5 mm +0.149 mm	0.86	0.27	67	0.89	0.34	62	0.89	0.45	49
-0.149 mm	1.13	0.17	86	1.30	0.17	86	1.30	0.21	85
Total	0.93	0.41	-	1.06	0.38	-	1.06	0.41	-

Source: Dawson (February 1997)

ICP scans of the UBC and Dyke Adit composites were completed. The ICP results are presented in Table 13.47.

Table 13.47 - Dawson February 1997 – ICP Scans of UBC and Dyke Adit

Element	Lower Detection Limit (%)	UBC Oxide Composite (%)	Dyke Adit Composite (%)
Silver	0.005	0.18	0.05
Aluminum	0.02	5.5	5.5
Arsenic	0.02	n.d.	n.d.
Boron	0.005	0.01	0.008
Barium	0.005	0.06	0.04
Beryllium	0.005	n.d.	n.d.
Bismuth	0.02	0.12	0.09
Calcium	0.005	0.06	0.20
Cadmium	0.005	0.03	n.d.
Cobalt	0.005	n.d.	n.d.
Chromium	0.005	0.02	0.01
Copper	0.005	0.11	0.07
Iron	0.005	2.4	2.3
Potassium	0.005	7.9	6.3
Lanthanum	0.02	n.d.	n.d.
Magnesium	0.005	0.12	0.18
Manganese	0.005	2.1	0.04
Molybdenum	0.005	n.d.	n.d.
Nickel	0.005	n.d.	n.d.
Phosphorus	0.10	n.d.	n.d.
Lead	0.02	n.d.	n.d.
Palladium	0.02	n.d.	n.d.
Platinum	0.02	n.d.	n.d.
Sulphur	0.005	0.79	0.55
Antimony	0.02	n.d.	n.d.
Selenium	0.02	n.d.	n.d.
Silicon	0.005	>10.0	>10.0
Tin	0.02	n.d.	n.d.
Strontium	0.005	0.01	0.01
Titanium	0.005	0.11	0.11
Thallium	0.02	n.d.	n.d.
Vanadium	0.005	n.d.	n.d.
Zinc	0.005	0.02	0.03
Zirconium	0.02	n.d.	n.d.

Note: n.d. = not detected above stated detection limit.

Source: Dawson (February 1997)

13.18 DAWSON METALLURGICAL LABORATORIES – MARCH 1997

This test program was carried out using different types of samples from the Bear Creek Zone since it is the majority of the mineralized material. A Main Zone composite was also tested. The samples were subjected to column leach tests. Cyanide agglomeration was examined to increase leach kinetics. Crush size and the equipment types were also investigated.

The Bear Creek sample was prepared from drillholes TC-11, 12, 13, 14, and 15. Bear Creek No.1 and No.2 were created as well as high-and-low-grade Bear Creek composites (HG and LG respectively). A listing of the head assays is presented in Table 13.48.

Table 13.48 - Dawson March 1997 – Head Assay Comparison

Head Analysis	Units and Elements	BC No. 1	BC No. 2	HG	LG
Interval Calculation	Au (g/t)	0.89	1.20	3.53	0.38
	Ag (g/t)	6.51	9.26	35.3	2.06
Assay Head	Au (g/t)	0.79	0.99	-	-
	Ag (g/t)	5.49	9.94	-	-
Average Back Calculation	Au (g/t)	0.79	1.17	3.22	0.41
	Ag (g/t)	6.86	8.57	30.2	3.43
Total Sulphur	%	1.37	1.45	0.85	1.18
Sulphide Sulphur	%	1.36	1.37	-	-
Copper	%	0.008	0.010	-	-
Iron	%	2.88	3.32	-	-
Arsenic	%	0.056	0.051	-	-
Zinc	%	0.020	0.019	-	-
Mercury	ppb	420	315	-	-

Note: BC = Bear Creek; HG = High Grade; LG = Low Grade.

Source: Dawson (March 1997)

The gravity tests showed the presence of free milling electrum. The free electrum ranged in size from 500 to 88 µm and represented about 27% of the total gold. It was believed that the presence of electrum possibly created a “nugget effect” which could lead to inconsistencies in the head assays. These inconsistencies were observed mainly in the course size range for the heads and residues.

Column leach tests were run at different crush sizes utilizing different equipment. Results from these column tests are presented in Table 13.49.

Table 13.49 - Dawson March 1997 – Column Leach Results Using Various Crush Product Sizes and Types

Crusher Description	Leach Days	% -150 µm	Percent Extraction (%)		Consumption (kg/t)	
			Au	Ag	NaCN	Lime
Main Zone Composite						
Jaw Crush -19.05 mm	130	7.6	47	31	0.88	5.25
Jaw Crush -12.7 mm	122	7.5	49	16	0.66	5.55
Jaw Crush -6.35 mm	130	13.2	55	52	0.86	4.9
Upp. LABWAL -3.36 mm	119	45.3	69	80	0.96	3.95
Bear Creek Composite No. 1						
Jaw Crush -6.35 mm	99	9.6	45	34	0.8	1.15
Fast Rolls -3.36 mm	111	18.7	43	33	0.88	1.4
Cemco VSI -3.36 mm	89	16.4	47	32	0.76	1.15
Krupp LABWAL -3.36 mm	121	43.0	50	59	0.90	1.3
Bear Creek Composite No. 2						
Jaw Crush -15.9 mm	104	9.8	26	35	0.98	1.15
Fast Rolls -3.36 mm	104	24.0	44	58	1.00	1.115
Krupp REGRO -15.9 mm	108	33.3	46	53	0.91	1.3
Krupp 2-Stage -3.36 mm	99	42.6	53	84	0.88	1.55
Average Column Back-Calculated Head			Au (g/t)	Ag (g/t)	-	-
Main Zone			1.03	12.000	-	-
Bear Creek No. 1			0.823	6.514	-	-
Bear Creek No. 2			1.063	6.514	-	-

Source: Dawson (March 1997)

The higher the percentage of material less than -150 µm, the higher the gold and silver recoveries from the leach. The HPGR size reduction also gave higher gold and silver recoveries than the crushers.

Further test work was completed to determine why there was only a 50% gold extraction from the HPGR. Replicate column leach tests were completed and the residues were analyzed. The results of the column leach replicates are shown in Table 13.50. The leach residues were crushed to 0.50 mm and run over a gravity table to produce a rougher concentrate which contained the majority of the sulphides. The concentrate was amalgamated to collect the free gold and all table products were analyzed. Results are presented in Table 13.51.

The tests showed that there was only a small percentage of free gold in the residue (2.1% and 3.8% respectively). About 20% of the residual gold was associated with visible sulphides and approximately 76% reported to the gravity tails. Further mineralogical work on the gravity tails indicated that the majority of the residual gold was fine and encapsulated in sulphides in large gangue particles. The liberated sulphides had a dense texture which would make them refractory.

Size by size assays of the head and the residue were completed to determine what size range the gold was being extracted from. A comparison between jaw crush and HPGR size analysis was also completed. Test 64 which was jaw crushed to -15.9 mm was compared with Test 69 which used the HPGR to achieve the -15.9 mm. Test 64 had a gold extraction of 29% and Test 69 had a 49% gold extraction. Each was leached for 239 days. The results of this analysis can be found in Table 13.52.

The analysis revealed that the majority of the gold was extracted from particles sizes less than 0.5 mm. The gold extraction was even higher for this size range for the HPGR sample in Test 69. There was also a larger weight percentage of material which was less than 0.5 mm in size in the HPGR sample.

Column leach tests were run on the high-and-low-grade Bear Creek composites. The low grade sample was at the proposed cut-off grade. Both samples were reduced in size using the LABWAL HPGR. The results of these tests are presented in Table 13.53. The higher-grade material had a higher gold recovery (67.2%) than the lower-grade material (36%).

Table 13.50 - Dawson March 1997 – HPGR /Column Leach Duplicates

Test No.	Crush Size	Leach Days	Calculated Head (g/t)		Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumed (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
37	LABWAL 3.36 mm	111	0.93	9.60	0.48	6.17	49.2	36.6	0.45	3.43	0.94	1.15
38	LABWAL 3.36 mm	128	1.06	7.54	0.51	3.77	50.7	50.6	0.55	3.77	1.06	1.65

Source: Dawson (March 1997)

Table 13.51 - Dawson March 1997 – Column Leach Residue Test Results at 0.5 mm Crush

Product	Test 37 LABWAL 3.36 mm			Test 38 LABWAL 3.36 mm		
	Wt %	Au (g/t)	% Distance	Wt %	Au (g/t)	% Distance
Amalgam	-	0.01	2.1	-	0.02	3.8
Amalgam Tail	1.6	0.11	22.9	1.8	0.10	19.2
Table Concentrate	1.6	0.12	25.0	1.8	0.12	23.0
Table Tail	98.4	0.36	75.0	98.2	0.40	77.0
Residue	100.0	0.48	100.0	100.0	0.52	100.0

Source: Dawson (March 1997)

Table 13.52 - Dawson March 1997 – Screen Analysis of Column Leach Test Head and Residues

Size Fraction	Test 64: -15.9 mm Jaw Crush				Test 69: -15.9 mm REGRO Crush			
	Au Head (g/t)	Residue		Size Extracted (%)	Au Head (g/t)	Residue		Size Extracted (%)
		Wt %	Au (g/t)			Wt %	Au (g/t)	
-19.05 mm +12.7 mm	0.86	8.7	0.89	0	1.13	3.6	0.75	34
-12.7 mm +6.35 mm	0.99	44.4	0.72	28	1.10	9.2	0.86	21
-6.35 mm +3.36 mm	1.41	17.4	0.86	39	1.34	13.0	0.79	41
-3.36 mm +2.0 mm	0.82	8.8	0.55	33	1.17	9.9	0.75	36
-2.0 mm +0.84 mm	0.93	4.8	0.72	22	1.17	11.0	0.58	50
-0.84 mm +0.5 mm	0.82	2.9	0.62	25	1.27	8.2	0.41	67
-0.5 mm +0.15 mm	1.30	3.2	0.45	66	0.93	10.8	0.48	48
-0.15 mm	0.93	9.8	0.41	56	0.93	34.3	0.27	71
Total	1.03	100.0	0.72	-	1.10	100.0	0.51	-

Source: Dawson (March 1997)

Table 13.53 - Dawson March 1997 – High and Low Grade Bear Creek Composites Column Leach Results

Test No.	Crush Size	Composite	Leach Days	Calculated Head (g/t)		Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumed (kg/t)	
				Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
78	LABWAL 3.36 mm	LG	270	0.38	1.71	0.24	0.69	36.0	62.1	0.14	1.03	1.39	2.75
79	LABWAL 3.36 mm	HG	275	3.09	26.74	1.03	7.89	67.2	70.5	2.06	18.86	1.52	3.55

Source: Dawson (March 1997)

Some diagnostics tests were run on the Bear Creek No. 1 composite. A ball mill grind was subjected to hand panning and amalgamation to determine the quantity of free milling gold. A ball mill grind was also subjected to a bottle roll to try and determine the maximum possible gold extraction. The final diagnostic test was a CIL test at -841 µm, to examine the barren solution with atomic absorption to develop a standard.

Table 13.54 - Dawson March 1997 – Column Leach Residue Test Results at 0.5 mm Crush

Test No.	Grind Size	Product	Wt %	Assay (g/t)		Distribution (%)	
				Au	Ag	Au	Ag
11	67% -75 µm	Amalgam	-	-	-	27.2	3.6
		Amalgam Tail	0.86	7.71	119.3	6.2	14.2
	P80 = 101 µm	Gravity Concentrate	0.86	41.4	151.5	33.4	17.8
		Gravity Tail	99.14	0.72	6.17	66.6	82.2
		Total (Calculated)	100.0	1.06	7.20	100.0	100.0
		Total (Assay)		0.79	5.49		

Source: Dawson (March 1997)

The results of the gravity and amalgamation diagnostics are presented in Table 13.54. The amalgam concentrate indicates that approximately 27% of the gold in the feed sample was present as free milling gold. Electrum was also observed in the 0.5 mm to 88 µm range. The results of the bottle roll tests on the ball mill grind product are presented in Table 13.55.

As mentioned, the bottle roll test on the ball mill grind was completed to determine the maximum gold extraction for this composite. The results show that 61% of the gold and 65% of the silver are the maximum recoveries for this composite. CIL tests were also completed to generate a barren solution. The results from the CIL test are presented in Table 13.56. The gold extraction at the 841 µm crush size was 58% and 65% for silver.

Table 13.55 - Dawson March 1997 – Ball Mill Grind Bottle Roll Test Results

Test No.	Description	Composite	Leach Hours	Calculated Head (g/t)		Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumption (kg/t)	
				Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
21	Ball Mill Grind 67% -75 µm	BC No. 1	144	0.72	6.86	0.27	2.40	60.8	64.9	0.45	4.46	0.48	1.03

Source: Dawson (March 1997)

Table 13.56 - Dawson March 1997 – CIL Bottle Roll Test Results

Test No.	Description	Composite	Leach Hours	Calculated Head (g/t)		Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumption (kg/t)	
				Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
21	841 µm Crush CIL	BC No. 1	72	0.72	4.80	0.31	1.71	58.4	64.7	0.41	3.09	0.80	0.92

Source: Dawson (March 1997)

Bottle roll leach work was completed at different crush and grind sizes (i.e. 6.35 mm, 3.36 mm, 2.0 mm, 841 µm, and 67% 74 µm). The results are presented in Table 13.57. The precious metal extractions started to level out at 48 hours of leaching. They then began to increase again between 96 and 120 hours. The leach kinetics were slow, so for tests 14, 15, and 17 the samples were agglomerated with 0.5 kg/t of sodium cyanide and leached with 1 kg/t sodium cyanide solution. In most instances the increase in sodium cyanide consumption by agglomerating the sample results in higher gold extraction. The gold recoveries were also higher for the finer material.

A series of crush tests involving different crushing equipment were carried out on Bear Creek No. 1 samples. The general trend was that the precious metals distribution followed the weight distribution. These screened head assays were carried out for jaw crusher, fast rolls, VSI and HPGR. Subsequent column leach tests were carried out on the crush products from the different pieces of equipment. The column leach feeds were agglomerated with 1.6 kg/t lime, 0.5 kg/t cement, and 0.5 kg/t sodium cyanide (except Test 36). The results of the tests can be found in Table 13.58. The precious metals recovery seemed to trend with the generation of finer material except for Test 45. This test utilized a jaw crusher and had the lowest fines, but still had the highest gold recovery (i.e. 52%). The same crush size utilizing leach aid however gave the lowest gold recovery.

Since the HPGR products gave the best precious metal recoveries, further test work was completed using Bear Creek No. 1 sample HPGR product. The HPGR was set to 3.36 mm. The purpose was to test agglomeration with and without sodium cyanide as well as the use of leach aid. The results are presented in Table 13.59. The results show that the gold recoveries get better with sodium cyanide agglomeration and the maximum dosage of leach aid.

The previous test program at Dawson showed that the gold leaching continued and some cases the kinetics increased at a steady rate per month for long-term leach. That trend was not seen in this set of work. More in depth work looking at the screen assays of leach residues were completed. The trend was that more gold was extracted from the finer particle sizes.

Long-term column leach studies on a Bear Creek No. 2 samples were completed. The results showing equipment type and gold recovery are presented in Table 13.60. The long-term results show that the crusher type which creates the larger amount of fines below 150 µm achieves the higher long-term gold recovery.

A comparison of gold extraction from size fractions was carried out for the Bear Creek No. 2 samples for the different crusher equipment and similar to previous results, the gold extraction was higher for the finer particle sizes.

Table 13.57 - Dawson March 1997 – Bear Creek No. 1 Bottle Roll Crush Size Series Test Results

Test No.	Crush Size	Leach Hours	Calculated Au Head (g/t)	Au Residue (g/t)	Au Extraction		Consumption (kg/t)	
					g/t	%	NaCN	Lime
7	6.35 mm	120	1.10	0.82	0.24	23.7	0.62	0.95
14	6.35 mm	240	0.58	0.41	0.17	27.9	0.89	1.18
8	3.36 mm	120	0.82	0.58	0.24	27.7	0.57	1.04
15	3.36 mm	240	0.79	0.58	0.21	26.2	1.16	1.25
9	2.0 mm	120	0.82	0.62	0.21	26.4	0.66	1.44
17	2.0 mm	240	0.69	0.41	0.24	38.1	1.02	1.29
B	841 µm	72	0.72	0.31	0.45	58.4	0.80	0.92
21	67% -75 µm	144	0.72	0.27	0.45	60.8	0.48	1.03

Source: Dawson (March 1997)

Table 13.58 - Dawson March 1997 – Bear Creek No. 1 Column Leach Crusher Type Series Test Results

Test No.	Crush Type	% 75 µm Fines	Leach Days	Calculated Au Head (g/t)		Au Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumption (kg/t)	
				Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
45	6.35 mm Jaw Crush	9.6	99	0.63	5.28	0.30	3.43	52.2	34.9	0.33	1.85	0.80	1.15
46	6.35 mm Jaw +Leach Aid	9.6	99	0.58	5.52	0.36	3.77	37.7	31.8	0.22	1.75	0.81	1.15
25	3.36 mm DML Fast Rolls	18.7	111	0.81	7.68	0.46	5.14	43.2	33.0	0.35	2.54	0.88	1.40
35	3.36 mm CEMCO VSI	16.4	89	0.88	8.02	0.46	5.49	47.2	31.5	0.41	2.54	0.77	1.15
37	3.36 HPGR- LABWAL	43.0	111	0.91	9.70	0.46	6.17	49.2	36.6	0.45	3.57	0.94	1.15

Source: Dawson (March 1997)

Table 13.59 - Dawson March 1997 – Bear Creek No. 1 Column Leach Crusher Type Series Test Results

Test No.	3.36 mm Grind	Leach Days	Calculated Au Head (g/t)	Au Residue (g/t)	Au Extraction		Consumed (kg/t)	
					g/t	%	NaCN	Lime
36	No NaCN Agglomeration	128	0.87	0.44	0.43	49.6	0.68	3.28
37	NaCN Agglomeration	111	0.91	0.46	0.45	49.2	0.94	1.13
38	NaCN + Leach Aid (0.08 kg/t)	128	1.08	0.53	0.55	50.7	1.06	1.67
59	NaCN + Leach Aid (0.05 kg/t)	119	0.86	0.45	0.41	47.9	0.91	1.13

Source: Dawson (March 1997)

Table 13.60 - Dawson March 1997 – Bear Creek No. 2 Column Leach Crusher Type Series Test Results

Leach Time	T64 -15.9 mm Jaw		T66 -3.36 mm Fast Rolls		T69 -19.9 mm REGRO-HPGR		T81 -3.36 mm 2-Stage	
	Au (g/t)	Au (%)	Au (g/t)	Au (%)	Au (g/t)	Au (%)	Au (g/t)	Au (%)
1 week	0.17	17.1	0.35	34.7	0.37	34.5	0.47	41.6
1 month	0.23	22.9	0.40	40.0	0.45	41.8	0.55	48.7
3 months	0.25	25.8	0.44	43.7	0.49	45.4	0.59	52.7
8 months	0.28	28.6	0.47	46.2	0.52	48.1	0.61	54.4
Residue	0.70	71.4	0.54	53.8	0.56	51.9	0.51	45.6
Head	0.98	100.0	1.01	100.0	1.08	100.0	1.12	100.0
-150 µm	8%		20%		31%		42%	

Source: Dawson (March 1997)

Table 13.61 - Dawson March 1997 – Bear Creek High- and Low-Grade Column Leach Test Results

Test No.	Composite	Leach Days	Calculated Au Head (g/t)		Au Residue (g/t)		Extraction (%)		Extraction (g/t)		Consumed (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
78	LG	270	0.38	1.71	0.24	0.69	36.0	62.1	0.14	1.03	1.39	2.75
79	HG	275	3.09	26.74	1.03	7.89	67.2	70.5	2.06	18.86	1.52	3.55

Source: Dawson (March 1997)

High (3.09 g/t) and low grade (0.38 g/t) gold Bear Creek composites were created to determine if selective mining of the high grade portion of the Bear Creek Zone could be processed, and the low grade was selected at the cut-off grade used for the study. The samples were agglomerated with 0.25 kg/t sodium cyanide, 2.5 kg/t cement, and 1 kg/t of hydrated lime. The agglomerated samples were then column leached. The results from these tests can be found in Table 13.61. The gold kinetics were fast for the high- grade sample and slower for the low grade.

13.19 TALAPOOSA MINING INC. – TECHNICAL AND ECONOMIC REVIEW – DECEMBER 1997

Talapoosa Mining Inc. (TMI) created an internal technical and economic review. Conclusions drawn by this review were that the Project based on the mining and process (60% gold recovery) parameters for oxide material used in the study would have a break even at a US$420/oz (1997 dollars). At the average gold recovery of 50 to 55%, a US$460/oz gold price would be required to break even. A summary table of the different alteration types and the gold recoveries from these alterations is presented in Table 13.62.

Table 13.62 - TMI – Oxide Resource Inventory and Metallurgical Tests by Alteration Type

Alteration Type	Tonnes	Contained Grams	Bottle Roll Tests Completed	Column Tests Completed	Estimated Recovery (%)
Oxidized Silicic	4,912,000	4,547,300	40	21	47
Oxidized Sericitic	4,032,000	2,351,400	21	9	62
Oxidized Propylitic	100,700	65,300	None	None	62
Oxidized Argillic	508,900	469,600	1	None	65
Oxidized Sericitic-Silicic Mix	1,071,000	587,900	None	5	60
Total	10,624,600	8,021,500	62	35	55

Source: TMI

The author came to the conclusion that the finer grind sizes used in the bottle roll tests gave better recoveries than the coarse crush sizes used in the column leach tests. The test data suggests that the less siliceous material has higher recoveries.

13.20 ORETEST PTY LTD. – APRIL 1999

Ninety-eight drill core samples in 1.5 m intervals were sent to the Oretest Pty Ltd. (Oretest) metallurgical test work laboratory in Western Australia. The drill core was combined into 11 composites. The composites were tested to characterize each composite for gold head grade and response to gold recovery processes. The processes tested were heavy media separation, gravity separation, leaching and flotation.

Samples were sent for mineralogy and a subset was also sent for ICP. The primary concern with the ICP analysis was the mercury and selenium content, which were both at low concentrations in the samples tested (i.e. mercury less than 0.09 ppm and selenium less than 10 ppm). The following observations were made about the gold and silver mineralogy.

- The gold did not occur as free or native gold in the samples analyzed. It occurred mainly in gold/silver minerals such as argentian gold, acanthite and electrum. The electrum was present within pyrite as a fine particle (i.e. less than 30 µm). The gold particle sizes varied in size from 200 µm down to a few microns in size.
- Silver was present as acanthite native silver, electrum, and argentian gold.
- Pyrite with minor amounts of marcasite was the major sulphide mineral with one sample showing pyrrhotite. Other sulphides present were chalcopyrite, sphalerite, arsenopyrite, goethite and leucoxene/rutile.
- The predominant silicate minerals were quartz, but there was also contained sericite and clay.

The first set of test work carried out was heavy media separation. This work was done to get an indication of the liberation crush/grind size. Composites 1, 2 and 3 were crushed to -1,000 µm, -500 µm, and 250 µm. The samples were then deslimed at 38 µm. A summary of the test results can be found in Table 13.63. The results show that there is an increase in recovery with finer grind size, but even at the 250 µm size the maximum gold recovery was 65%. Since the gold recovery was low, the laboratory decided not to test the remaining eight composites. The -500 µm sinks from Composite No. 1 were sent for mineralogical analysis. All occurrences of gold and silver were electrum and were found in the 15 to 50 µm range.

Gravity separation tests were completed on the 11 composites. The composites were subjected to grinding to P_{80} of 150 µm. The composites were then feed to the laboratory scale Knelson concentrator and the Knelson concentrate was panned to create a pan concentrate. Results from this test work are presented in Table 13.64.

Table 13.63 - Oretest - Heavy Liquid Separation Results

Composite	Crush Size (µm)	+2.96 Specific Gravity (i.e. Sinks) Mass %	Au % Dist.	Ag % Dist.	S % Dist.	-38 µm Slimes Mass %	Au % Dist.	Ag % Dist.	S % Dist.	Possibly Liberated* Mass %	Au % Dist.	Ag % Dist.	S % Dist.
No. 1	-1,000	0.5	17.4	10.8	55.3	7.9	13.3	19.8	8.8	8.4	30.7	30.6	64.1
	-500	0.5	21.3	13.4	64.3	10.3	15.9	24.8	12.1	10.8	37.2	38.2	76.4
	-250	0.6	34.5	10.5	67.0	2.1	3.7	6.8	2.8	2.7	38.2	41.3	69.8
No. 2	-1,000	1.1	48.7	21.4	41.9	11.1	8.2	17.7	15.4	12.2	56.9	39.1	57.3
	-500	1.2	54.7	28.8	48.6	15.8	13.0	27.6	21.5	17	67.7	46.5	70.1
	-250	1.3	63.1	44.2	49.9	2.4	2.0	3.8	3.0	3.7	65.1	48.0	52.9
No. 3	-1,000	1.3	43.3	20.3	52.1	1.5	0.6	2.3	1.5	2.8	43.9	22.6	53.6
	-500	1.4	38.9	31.1	59.6	2.0	1.4	3.2	1.9	3.4	40.3	34.3	61.5
	-250	0.8	56.2	20.1	37.4	2.7	1.8	4.7	2.7	3.5	58	24.8	40.1

Notes: * Combining sinks with slimes fraction.
Dist. = Distribution.

Source: Oretest (April 1999)

Table 13.64 - Oretest – Summary of Gravity Test Results at 150 µm

	Composite No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7	No. 8	No. 9	No. 10	No. 11	Statistics Average	Standard Deviation
Head Assays													
Calculated Head Au (g/t)	8.46	2.53	3.86	6.89	3.51	7.83	3.55	2.29	3.58	1.26	0.93	4.06	2.44
Assay Head, Au (g/t)	9.15	1.95	3.77	7.16	1.42	8.75	2.31	1.91	3.31	0.85	0.68	3.75	2.98
Calculated Head, Ag (ppm)	119	15.6	30.2	92.9	9.9	21.4	16.2	18.1	52.1	10.8	4.4	35.5	35.7
Assay Head, Ag (ppm)	101	16.0	29.2	105	8.5	105	13.9	21.1	52.9	11.1	4.3	42.5	39.4
Calculated Head S (%)	0.34	0.85	0.90	0.39	1.62	1.08	0.85	0.74	2.17	1.51	1.30	1.07	0.52
Assay Head S (%)	0.32	0.83	0.97	0.37	1.71	1.12	0.92	0.82	2.02	1.40	1.16	1.06	0.49

	Composite											Statistics	
	No. 1	**No. 2**	**No. 3**	**No. 4**	**No. 5**	**No. 6**	**No. 7**	**No. 8**	**No. 9**	**No. 10**	**No. 11**	**Average**	**Standard Deviation**
Pan Concentrate													
Mass (%)	0.05	0.05	0.05	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.09	0.02
Au (g/t)	3,730	2,100	2,190	2,190	1,160	2,840	1,470	903	465	223	296	1,597	1070
Au Distribution (%)	21.2	43.3	30.6	32.1	33.2	37.1	42.1	40.2	13.4	18.4	33.2	31.3	9.39
Ag (ppm)	84,000	2,010	2,210	2,250	178	674	1,610	7.9	1390	307	418	1,769	2245
Ag Distribution (%)	3.4	6.7	3.9	2.4	1.8	3.2	10.1	0.0	2.8	2.9	9.8	4.3	3.08
S (%)	48.0	46.7	51.3	55.6	56.7	54.5	52.8	47.2	56.5	54.8	52.0	52.4	3.52
S Distribution (%)	6.7	2.9	3.1	14.6	3.5	5.1	6.3	6.5	2.7	3.8	4.2	5.4	3.2
Knelson Concentrator (i.e. Pan Concentrate + Pan Tail)													
Mass (%)	2.8	3.0	3.8	4.8	6.9	6.1	5.4	5.6	7.1	6.7	5.4	5.2	1.42
Au (g/t)	162	56.8	64.7	83.5	30.9	115	55.3	27.7	34	12.0	11.3	59.5	44.1
Au Distribution (%)	54.3	67.1	64.2	58.4	60.5	89.5	84.2	67.6	68.1	63.7	65.6	67.6	9.99
Ag (ppm)	57.0	6.6	11.0	42.3	6.9	8.3	8.1	7.7	43.4	7.6	2.3	18.3	18.4
Ag Distribution (%)	26.0	28.4	23.3	26.6	42.3	34.7	42.3	28.6	56.7	44.8	34.1	35.3	9.8
S (%)	0.68	1.06	2.01	0.73	2.42	1.84	0.92	0.99	2.55	2.09	1.72	1.55	0.66
S Distribution (%)	51.8	35.3	52.0	50.6	63.3	59.2	45.7	38.5	66.6	62.2	45.2	51.9	9.7
Knelson Tailing													
Mass (%)	97.2	97.0	96.2	95.2	93.1	93.9	94.6	94.4	92.9	93.3	94.6	94.8	1.42
Au (g/t)	3.98	0.86	1.44	3.01	1.49	0.88	0.60	0.79	1.23	0.49	0.34	1.37	1.08
Au Distribution (%)	45.7	32.9	35.8	41.6	39.5	10.5	15.8	32.4	31.9	36.3	34.4	32.4	9.99
Ag (ppm)	90.5	11.5	24.1	71.6	6.1	14.9	9.9	13.7	24.3	6.4	3.1	25.1	27.4
Ag Distribution (%)	74.0	71.6	76.7	73.4	57.7	65.3	57.7	71.4	43.3	55.2	65.9	64.7	9.8
S (%)	0.17	0.57	0.45	0.2	0.64	0.47	0.49	0.48	0.78	0.61	0.75	0.51	0.19
S Distribution (%)	48.2	64.7	48.0	49.4	36.7	40.8	54.3	61.5	33.4	37.8	54.8	48.1	9.7

Source: Oretest (April 1999)

Repeats of the gravity tests for the first three composites were completed and the pan concentrate sent for mineralogical analysis. The analysis revealed that no free gold was present in the pan concentrates and the gold was associated with silver as electrum or acanthite. Further analysis also revealed a strong correlation between gold and silver in the tailings stream. The Knelson and pan tailings from the first three composites were combined to create feed for the subsequent leaching and flotation tests described later.

Sodium isobutyl xanthate (SIBX) and Aerophine 3418A were used as collectors in "sighter" tests to determine their effectiveness for flotation test work with these samples. The SIBX outperformed the Aerophine and was used for all subsequent flotation tests. Composites 1, 2, and 3 were subjected to a grind to produce a P_{80} of 75 µm. Results from these tests can be found in Table 13.65. All 11 composites were also subjected to flotation tests at a grind P_{80} of 150 µm. The results of these tests can be found in Table 13.66.

Table 13.65 - Oretest – Summary of Flotation Results at P80=75 µm Grind for Composites 1, 2, and 3

Test No.	Composites			Statistics	
	No. 1 JA1487	No. 2 JA1489	No. 3 JA1485	Average	Standard Deviation
Head Assays					
Calculated Head, Au (g/t)	7.65	2.12	2.32	4.03	2.56
Assay Head, Au (g/t)	9.13	1.95	3.77	4.95	3.05
Calculated Head, Ag (ppm)	119	16.3	29.4	54.9	45.7
Assay Head Ag (ppm)	101	16.0	29.2	48.7	37.3
Calculated Head S (%)	0.34	0.84	0.86	0.68	0.24
Assay Head S (%)	0.32	0.83	0.97	0.71	0.28
First Concentrate					
Mass (%)	2.6	4.9	3.8	3.78	0.97
Au (g/t)	271	39.0	50.3	120	107
Au Distribution (%)	91.1	91.1	82.5	88.3	4.04
Ag (ppm)	4,250	278	668	1,732	1,788
Ag Distribution (%)	91.8	84.5	86.4	87.6	3.09
S (%)	12.8	16.2	21.9	17.0	3.75
S Distribution (%)	96.8	95.5	97.4	96.6	0.80
Total Concentrate (i.e. 1st and 2nd Concentrates)					
Mass (%)	3.9	8.0	6.0	5.94	1.68
Au (g/t)	183	24.8	33.4	80.3	72.5
Au Distribution (%)	92.5	93.5	85.9	90.6	3.35
Ag (ppm)	2,871	178	443	1,164	1,212
Ag Distribution (%)	93.4	87.6	89.8	90.2	2.40
S (%)	8.54	10.3	14.2	11.0	2.36
S Distribution (%)	97.2	97.8	98.9	98.0	0.71

Test No.	Composites			Statistics	
	No. 1 JA1487	No. 2 JA1489	No. 3 JA1485	Average	Standard Deviation
Flotation Tailings					
Mass (%)	96.1	92.0	94.0	94.1	1.68
Au (g/t)	0.60	0.15	0.35	0.37	0.18
Au Distribution (%)	7.5	6.5	14.1	9.38	3.35
Ag (ppm)	8.2	2.2	3.2	4.53	2.62
Ag Distribution (%)	6.6	12.4	10.2	9.76	2.40
S (%)	0.01	0.02	0.01	0.01	0.00
S Distribution (%)	2.8	2.2	1.1	2.04	0.71

Source: Oretest (April 1999)

Table 13.66 - Oretest – Summary of Flotation Test Results at P80=150 μm Grind

Test No.	Composite											Statistics	
	No. 1 JA1486	No. 2 JA1488	No. 3 JA1484	No. 4 JA1526	No. 5 JA1527	No. 6 JA1528	No. 7 JA1529	No. 8 JA1530	No. 9 JA1532	No. 10 JA1533	No. 11 JA1534	Average	Standard Deviation
Head Assays													
Calculated Head Au (g/t)	8.20	2.03	2.26	7.33	1.53	8.08	2.77	1.47	4.54	0.96	0.74	3.63	2.78
Assay Head, Au (g/t)	9.13	1.95	3.77	7.16	1.42	8.75	2.31	1.91	3.31	0.85	0.68	3.75	2.98
Calculated Head, Ag (ppm)	95.7	15.3	30.4	94.8	9.54	26.4	15.7	20.2	53.5	10.6	4.5	34.2	31.4
Assay Head, Ag (ppm)	101	16.0	29.2	105	8.50	25.8	13.9	21.1	52.9	11.1	4.3	35.3	34.3
Calculated Head S (%)	0.35	0.80	0.89	0.41	1.60	1.03	0.87	0.72	2.05	1.42	1.18	1.03	0.48
Assay Head S (%)	0.32	0.83	0.97	0.37	1.71	1.12	0.92	0.82	2.02	1.40	1.16	1.06	0.49
First Concentrate													
Mass (%)	2.7	4.8	4.0	1.9	3.2	2.8	2.8	2.6	17.9	8.3	6.9	5.25	4.42
Au (g/t)	270	35.8	40.4	306	37.8	269	86.7	33.8	17	8.5	9.3	101	113
Au Distribution (%)	87.5	85.1	71.2	78.5	77.6	94.2	86.2	58.8	68.6	73.3	86.5	78.8	9.81
Ag (ppm)	3,180	243	623	4,090	196	773	460	502	251	94.3	52.1	951	1,297
Ag Distribution (%)	88.3	76.9	81.7	81.1	64.7	82.8	80.7	63.4	83.8	74.2	80.8	78.0	7.4
S (%)	12.8	15.1	21.0	9.60	18.0	20.0	14.7	10.0	9.0	15.7	16.0	14.7	3.88
S Distribution (%)	96.8	90.7	94.2	44.0	35.5	55.2	46.6	35.8	78.3	91.9	94.0	69.4	24.6
Total Concentrate (i.e. 1st and 2nd Concentrates)													
Mass (%)	3.8	9.7	6.6	4.1	6.8	6.1	5.4	5.9	25.8	12.9	10.0	8.8	6.0
Au (g/t)	195	19.1	25.5	150	19.5	129	46.2	19.1	16	5.9	6.7	57.5	64.1
Au Distribution (%)	89.5	91.0	74.6	84.1	86.3	96.5	90.6	76.8	91.8	78.7	90.9	86.4	6.7
Ag (ppm)	2,297	133	395	2,016	115	388	256	260	194	67.7	38.3	560	763
Ag Distribution (%)	90.3	84.0	85.9	87.1	81.4	89.0	88.6	75.8	93.3	82.7	85.9	85.8	4.6
S (%)	9.08	8.04	13.0	9.55	21.1	16.0	13.6	8.93	7.5	10.9	11.41	11.73	3.85
S Distribution (%)	97.3	96.6	96.8	95.3	89.5	94.5	84.8	73.7	93.9	98.8	96.9	92.6	7.1

Test No.	Composite											Statistics	
	No. 1 JA1486	No. 2 JA1488	No. 3 JA1484	No. 4 JA1526	No. 5 JA1527	No. 6 JA1528	No. 7 JA1529	No. 8 JA1530	No. 9 JA1532	No. 10 JA1533	No. 11 JA1534	Average	Standard Deviation
Flotation Tailings													
Mass (%)	96.2	90.3	93.4	95.9	93.2	93.9	94.6	94.1	74.2	87.1	90.0	91.2	6.0
Au (g/t)	0.90	0.20	0.62	1.22	0.23	0.30	0.28	0.36	0.51	0.24	0.08	0.45	0.33
Au Distribution (%)	10.5	9.0	25.4	15.9	13.7	3.5	9.4	23.2	8.2	21.3	9.1	13.6	6.7
Ag (ppm)	9.6	2.7	4.6	12.7	1.9	3.1	1.9	5.2	4.8	2.1	0.7	4.5	3.5
Ag Distribution (%)	9.7	16.0	14.1	12.9	18.6	11.0	11.4	24.2	6.7	17.3	14.1	14.2	4.6
S (%)	0.01	0.03	0.03	0.02	0.18	0.06	0.14	0.20	0.17	0.02	0.04	0.08	0.07
S Distribution (%)	2.7	3.4	3.2	4.7	10.5	5.5	15.2	26.3	6.1	1.2	3.1	7.4	7.1

Source: Oretest (April 1999)

As illustrated by the flotation results, the final grind size has an impact on the gold recovery to the concentrate. The average flotation grade at the 150 µm grind was 86.4% gold and at 75 µm it was 90.6%. This is the gold recovery to concentrate and not the final gold recovery to some other medium (i.e. cyanide solution or smelting matte). The tailings from the 75 µm grind flotation tests were analyzed to determine the mineralogy of the lost gold. The gold was present as argentian gold or possibly electrum.

Further analysis of the results showed a similar correlation between silver and gold recovery as was seen in the gravity separation test work. This would suggest that to recover more gold, you should recover more silver. The majority of samples showed little correlation between gold and sulphur recovery, but some composites did show a relationship.

Test work was completed to determine if pre-concentration of the feed by gravity separation prior to flotation could possibly improve the overall gold recovery. Composites 1, 2, and 3 were subjected to a grind P_{80} = 150 µm and fed to gravity separation. The gravity tail was then the feed to flotation. The results are presented in Table 13.67.

Table 13.67 - Oretest – Gravity and Flotation Test Results

Composite	Gold Recovery (%)		Silver Recovery (%)		Sulphur Recovery (%)	
	Flotation	Gravity + Flotation	Flotation	Gravity + Flotation	Flotation	Gravity + Flotation
No.1	89.5	86.6 (21.2)	90.3	83.8 (3.4)	86.3	97.0 (6.7)
No. 2	91.0	91.6 (43.3)	84.0	78.4 (6.7)	81.4	87.5 (2.9)
No. 3	74.6	83.7 (30.6)	85.9	73.5 (3.9)	89.5	89.3 (3.1)

Note: Bracketed figures are the gravity component.

Source: Oretest (April 1999)

As the results show, neither the gold or silver recoveries were dramatically improved.

All composites were subjected to bottle roll cyanidation tests. The composites were tested at P_{80} = 150 µm and P_{80} = 75 µm. The results of the bottle roll tests are presented in Table 13.68 and 13.69. The finer grind size resulted in increased gold and silver recoveries. The average gold recovery for the coarser grind was 74.5% and 84.3% for the finer. Silver was 49.7% and 55.4% respectively.

Composites 9, 10, and 11 had viscosity problems due to the presence of clay and sericite. The viscosity did not vary with the grind size.

Table 13.68 - Oretest – Summary of Bottle Roll Cyanide Leach Test at P80=150 µm

Test No.	Composite											Statistics	
	No. 1 JA1481	No. 2 JA1483	No. 3 JA1491	No. 4 JA1517	No. 5 JA1519	No. 6 JA1521	No. 7 JA1523	No. 8 JA1525	No. 9 JA1535	No. 10 JA1536	No. 11 JA1525	Average	Standard Deviation
Gold													
Calculated Head, Au (g/t)	9.60	2.59	4.30	8.25	1.74	7.96	3.68	1.21	4.34	1.44	1.11	3.72	2.76
Assay Head, Au (g/t)	9.13	1.95	3.77	7.16	1.42	8.8	2.31	1.91	3.31	0.85	0.68	3.30	2.83
Extracted Au (ppm)	8.60	2.12	3.90	7.12	1.37	7.29	3.46	0.80	2.43	0.84	0.72	3.00	2.58
Recovery (%)	89.6	81.8	90.7	86.2	78.8	91.6	94.0	66.4	56.0	58.1	65.0	74.5	14.1
Residue, Au (g/t)	1.00	0.47	0.40	1.14	0.37	0.67	0.22	0.41	1.91	0.61	0.39	0.71	0.52
Silver													
Calculated Head, Au (g/t)	122	16.6	33.8	108	11.6	27.4	17.1	20.3	75.1	14.8	5.3	35.0	34.3
Assay Head, Au (g/t)	101	16.0	29.2	105	8.5	25.8	13.9	21.1	52.9	11.1	4.3	30.3	31.6
Extracted Au (ppm)	45.3	8.3	18.9	44.0	6.4	16.9	11.8	9.9	26.1	6.2	2.4	15.5	12.8
Recovery (%)	37.0	49.8	55.9	40.7	55.3	61.7	69.0	48.7	34.8	41.9	45.1	49.7	10.8
Residue, Au (g/t)	77.0	8.3	14.9	64.1	5.2	10.5	5.3	10.4	49.0	8.6	2.9	19.5	21.9
Reagent Consumption													
NaCN (kg/t)	0.36	0.18	0.21	0.29	0.19	0.24	0.14	0.13	0.32	0.23	0.24	0.22	0.06
Lime (kg/t)	0.19	0.71	0.54	0.21	0.25	0.44	0.49	0.40	1.22	1.46	0.58	0.63	0.43

Source: Oretest (April 1999)

Table 13.69 - Oretest – Summary of Bottle Roll Cyanide Leach Test at P80=75 µm

Test No.	Composite											Statistics	
	No. 1 JA1480	**No. 2 JA1482**	**No. 3 JA1490**	**No. 4 JA1516**	**No. 5 JA1518**	**No. 6 JA1520**	**No. 7 JA1522**	**No. 8 A1524**	**No. 9 JA1565**	**No. 10 JA1566**	**No. 11 JA1567**	**Average**	**Standard Deviation**
Gold													
Calculated Head, Au (g/t)	10.1	2.36	3.44	7.34	1.55	7.55	2.28	1.90	3.53	0.96	0.69	3.79	2.97
Assay Head, Au (g/t)	9.13	1.95	3.77	7.16	1.42	8.75	2.31	1.91	3.31	0.85	0.68	3.75	2.98
Extracted Au (ppm)	9.72	1.97	3.21	6.89	1.32	7.33	2.11	1.46	2.52	0.68	0.47	3.43	2.96
Recovery (%)	96.2	83.3	93.3	93.9	85.2	97.1	92.6	76.6	71.4	70.7	67.4	84.3	10.6
Residue, Au (g/t)	0.38	0.40	0.23	0.45	0.23	0.22	0.17	0.45	1.01	0.28	0.23	0.37	0.22
Silver													
Calculated Head, Au (g/t)	135	22.2	32.9	112	10.9	31.9	15.6	22.2	62.4	12.6	4.9	42.0	41.3
Assay Head, Au (g/t)	101	16.0	29.2	105	8.5	25.8	13.9	21.1	52.9	11.1	4.3	35.3	34.3
Extracted Au (ppm)	50.0	14.6	19.7	49.3	6.8	21.6	10.7	11.6	30.6	6.8	2.4	20.3	15.7
Recovery (%)	37.1	65.7	59.8	43.9	62.4	67.7	68.5	52.2	49.0	53.8	49.0	55.4	9.8
Residue, Au (g/t)	84.6	7.6	13.2	63.0	4.1	10.3	4.9	10.6	31.8	5.8	2.5	21.7	26.1
Reagent Consumption													
NaCN (kg/t)	0.24	0.21	0.21	0.23	0.12	0.21	0.26	0.26	0.24	0.19	0.12	0.21	0.05
Lime (kg/t)	0.34	0.73	0.62	0.23	0.47	0.27	0.38	0.54	1.80 90	1.80	0.60	0.71	0.54

Source: Oretest (April 1999)

Leach work involving leach enhancement agents were completed. The enhancements were the use of lead addition (as lead oxide (PbO)), oxygen addition and pre-concentration by gravity separation. Only Composites 1, 2 and 3 were used for these tests. The tests were completed in agitated vats as opposed to the bottle roll leach.

Oxygen was added as a blanket above the leach slurry and the dissolved oxygen levels were kept in excess of 20 ppm for these oxygen addition tests. Results from these tests are presented in Table 13.70.

Table 13.70 - Oretest – Results of Oxygen Addition to Vat Leach

Composite	Gold Recovery (%)		Silver Recovery (%)		NaCN (kg/t)		Lime (kg/t)	
	Leach	Leach + O2	Leach	Leach + O2	Leach	Leach + O2	Leach	Leach + O2
No. 1	89.6	94.3	37.0	50.7	0.36	0.75	0.19	0.13
No. 2	81.8	79.3	49.8	50.9	0.18	0.42	0.71	0.50
No. 3	90.7	83.0	55.9	52.0	0.21	0.39	0.54	0.31

Source: Oretest (April 1999)

The gold leach rates seemed to increase with the addition of oxygen and there was some slight increase in sodium cyanide consumption, but overall there were no significant increases in recoveries.

Lead was added to the leach vats at 500 g/t lead oxide. An oxygen blanket was also maintained to keep the dissolved oxygen levels above 20 ppm. Results from the lead addition test work are shown in Table 13.71.

Table 13.71 - Oretest – Results of Lead and Oxygen Addition to Vat Leach

Composite	Gold Recovery (%)		Silver Recovery (%)		NaCN (kg/t)		Lime (kg/t)	
	Leach	Leach + O2 + PbO	Leach	Leach + O2 + PbO	Leach	Leach + O2 + PbO	Leach	Leach + O2 + PbO
No. 1	89.6	96.0	37.0	80.7	0.36	0.87	0.19	0.17
No. 2	81.8	86.4	49.8	70.5	0.18	0.51	0.71	0.35
No. 3	90.7	90.0	55.9	74.5	0.21	0.33	0.54	0.30

Source: Oretest (April 1999)

The same tests were run with lead oxide addition at the same rate but no oxygen blanket. The results of these tests are presented in Table 13.72.

Table 13.72 - Oretest – Results of Lead Addition to Vat Leach

Composite	Gold Recovery (%)		Silver Recovery (%)		NaCN (kg/t)		Lime (kg/t)	
	Leach	Leach + PbO	Leach	Leach + PbO	Leach	Leach + PbO	Leach	Leach + PbO
No. 1	89.6	95.0	37.0	78.2	0.36	1.02	0.19	0.20
No. 2	81.8	77.0	49.8	72.3	0.18	0.60	0.71	0.60
No. 3	90.7	81.2	55.9	79.1	0.21	0.78	0.54	0.34

Source: Oretest (April 1997)

The lead oxide addition significantly improved the silver leach kinetics and the final silver recoveries. It gave the gold a slight increase in recovery as well, but also increased the sodium cyanide consumption. The results of the lead oxide addition both with the oxygen blanket and without are similar.

The final leach enhancement tested was to try pre-concentration by gravity separation prior to the leach. The gravity tails produced from the Knelson and panning were leached. Results can be found in Table 13.73.

Table 13.73 - Oretest – Results of Gravity Pre-concentration Prior to Val Leach

Composite	Gold Recovery (%)		Silver Recovery (%)		NaCN (kg/t)		Lime (kg/t)	
	Leach	Gravity + Leach	Leach	Gravity + Leach	Leach	Gravity + Leach	Leach	Gravity + Leach
No. 1	89.6	87.2 (21.2)	37.0	30.8 (3.4)	0.36	0.18	0.19	0.44
No. 2	81.8	83.6 (43.3)	49.8	53.3 (6.7)	0.18	0.67	0.71	1.08
No. 3	90.7	85.3 (30.6)	55.9	58.2 (3.9)	0.21	0.39	0.54	0.40

Source: Oretest (April 1999)

The gravity pre-concentration did not appear to increase the overall precious metal recoveries although it did slightly increase the kinetics. It was still believed that the gravity pre-treatment should be explored in further test work for material known to have larger gold (gold/silver) particles.

The conclusions the authors drew from this test work is that grinding to 75 µm could possibly be justified for high gold content (more than 3 g/t gold), but the lower grade material (less than 2 g/t gold) should be coarse ground (P_{80} = 150 µm) and a flotation pre-concentration should be done prior to leach. Any intermediate grades (2 to 3 g/t gold) should be cyanide leached or subjected to flotation. For comparison sake, the average recoveries tested of the eleven composites for each process were placed in Table 13.74.

Table 13.74 - Oretest – Average Results of all Composites for each Process

Process	Mass (%)	Gold Recovery (%)	Silver Recovery (%)	Sulphur Recovery (%)
Gravity	5.2	67.6	35.3	51.9
Flotation	8.8	86.4	85.8	92.6
Leaching	-	74.5 (84.3)	49.7 (55.4)	-

Note: Figures in brackets are recoveries for 75 µm.

Source: Oretest (April 1999)

13.21 MCCLELLAND LABORATORIES INC. – FEBRUARY 2015

A metallurgical testing program was undertaken by Gunpoint Exploration Ltd., in late 2013, to evaluate unoxidized ore types from the Talapoosa Bear Creek Zone for heap leach cyanidation amenability. The testing was conducted at McClelland Laboratories (Davis, 2015). Test work conducted included bottle roll and column leach tests, on unoxidized hangingwall and footwall composites from the Bear Creek Zone. These classifications were based on the refined interpretation of the controls of mineralization for the property, as described in the TetraTech Talapoosa resource summary (2013). A more limited scope of work was also conducted on oxidized composites from the Bear Creek Zone, and an unoxidized composite from the Main Zone.

Cyanidation testing was conducted on a total of seven drill core composites from the Talapoosa Resource Area. The composites were produced from 73 drill core samples, taken from a total of six drill holes. A summary of the composite make-up is shown in Table 13.75.

Table 13.75 - Composite Make-Up Information, Gunpoint 2013 Metallurgical Composites

Composite	Drill Hole	Interval, feet			Estimated Grade, gAu/mt
		From	To	Interval	
GUN_L3	GTI-002	342	496.5	154.5	
GUN_L3	GTI-003	257	328.5	71.5	
GUN_L3 (Bear Creek Hanging Wall Un-Oxidized)				**226**	**1.73**
GUN-L4	GTI-002	574	602	28	
GUN-L4	GTI-003	430	592	162	
GUN-L4	GTI-004	217	497	280	
GUN_L4 (Bear Creek Footwall Un-Oxidized)				**470**	**1.20**
GUN_L5	GTI-005	119	252	133	
GUN_L5	GTI-007	374	700	326	
GUN_L5 (Bear Creek Hanginwall Un-Oxidized)				**459**	**1.19**
GUN-L6	GTI-005	369	564	195	
GUN-L6	GTI-006	279	369	90	
GUN-L6	GTI-007	700	745	45	
GUN_L6 (Bear Creek Footwall Un-Oxidized)				**330**	**1.10**
GUN_L7	GTI-005	42.5	119	76.5	
GUN_L7 (Bear Creek Hangingwall Oxide)				**76.5**	**0.47**
GUN_L8	GRT-006	74	134	60	
GUN_L8 (Bear Creek Footwall Oxide)				**60**	**0.45**
GUN_L9	GTI-006	389	449	60	
GUN_L9 (Main Zone Un-Oxidized)				**60**	**0.83**

Six of the composites tested represented material from the Bear Creek zone. Those included two each representing unoxidized material from the hanging wall (GUN_L3 and GUN_L5) and footwall (GUN_L4 and GUN_L6), and one each representing oxidized material from the hanging wall (GUN_L7) and footwall (GUN_L8). The seventh composite represented unoxidized material from the Main Zone (GUN_L9. The Main Zone material was described as being mineralogically similar to the Bear Creek Hanging wall Zone (Baker, 2014).

Principal objectives of the testing program were to:

1. Evaluate and compare heap leach and agitated cyanidation gold and silver recoveries of samples from the unoxidized Bear Creek Hanging wall and Footwall Zones.
2. Evaluate the potential benefits of HPGR (high pressure grinding rolls) vs. conventional crush on both the Bear Creek Hanging wall and Footwall Zones.
3. Compare gold and silver recoveries between oxidized and unoxidized composites from the Bear Creek Hanging wall and Footwall Zones.
4. Confirm that the gold and silver recoveries from the unoxidized material from the Main Zone were similar to those from the Bear Creek Zone Hanging wall Zone.
5. The testing program included bottle roll tests on each of the seven composites at feed sizes of 80%-1.7mm and 80%-75µm. Additional bottle roll tests were conducted on the L-3 hanging wall composite at 6.3mm and 3.4mm (conventionally crushed and HPGR product) feed sizes. Column tests were conducted on four composites comparing conventionally crushed 6.3mm feed and HPGR product (3.4mm). In addition there were two column leach tests conducted with conventionally crushed feed that approximated the HPGR feed size (3.4mm).

Head and tail screen analyses were conducted on each column feed and residue. Before column testing, bench scale tests were conducted on the HPGR product from each composite to optimize agglomerating conditions. Load/permeability (hydraulic conductivity) tests were conducted on select column leached residues to evaluate permeability of the leached agglomerates under simulated commercial heap stack height compressive loadings.

Summary results from the bottle roll tests are shown in Table 13.76. Summary results from column leach tests are shown in Table 13.77.

Table 13.76 - Summary Results, Bottle Roll Tests, Gunpoint 2013 Metallurgical Testing

Composite	Feed Size, (P_{80})	Au		Ag		Reagent Requirements, kg/mt ore	
		Recovery, %	Calc'd Head, g/mt ore	Recovery, %	Calc'd Head, g/mt ore	NaCN Cons.	Lime Added
GUN_L3	6.3 mm	31.4	1.02	23.1	14.7	<0.07	1.0
GUN_L3	3.4 mm	48.5	1.03	36.7	14.7	0.13	1.2
GUN_L3	HPGR	50.0	1.04	36.4	14.3	0.15	1.4
GUN_L3	1.7 mm	59.3	0.91	41.5	13.0	0.13	1.3
GUN_L3	75 µm	82.4	0.85	58.9	12.9	0.10	1.5
GUN-L4	1.7 mm	53.8	0.80	30.2	6.3	0.13	1.6
GUN-L4	75 µm	73.4	0.94	54.1	7.4	<0.07	1.6
GUN_L5	1.7 mm	45.8	0.83	39.1	17.9	0.10	2.9

Composite	Feed Size, (P_{80})	Au Recovery, %	Au Calc'd Head, g/mt ore	Ag Recovery, %	Ag Calc'd Head, g/mt ore	Reagent Requirements, kg/mt ore NaCN Cons.	Reagent Requirements, kg/mt ore Lime Added
GUN_L5	75 µm	68.0	0.97	51.7	14.5	0.21	3.6
GUN-L6	1.7 mm	38.2	1.10	38.4	19.8	0.21	2.8
GUN-L6	75 µm	71.8	1.17	57.1	14.7	0.13	2.0
GUN_L7	1.7 mm	72.1	0.43	38.6	4.4	0.19	5.1
GUN_L7	75 µm	92.1	0.63	77.4	5.3	0.16	6.3
GUN_L8	1.7 mm	54.3	0.46	40.3	6.2	0.13	3.7
GUN_L8	75 µm	76.8	0.56	48.8	4.1	0.29	5.1
GUN_L9	1.7 mm	59.2	0.76	53.1	22.8	0.17	2.3
GUN_L9	75 µm	78.9	1.09	50.9	26.7	0.26	3.1

Bottle roll test results showed that gold recoveries at the 1.7mm feed sizes generally ranged from 45.8% to 59.3%, in 4 to 5 days of leaching. Gold recovery was lower (38.2%) for the GUN_L6 composite and higher (73.4%) for the GUN_L7 composite. As described above, GUN_L6 was one of the two Bear Creek Footwall un-oxidized composites. GUN_L7 was the Bear Creek hanging wall oxide composite. Correlation between the bottle roll (1.7mm feed size) and column leach test results was poor. Silver recoveries from the 1.7mm feed size bottle roll tests were low (30.2% - 53.1%).

Milling the composites to 80%-75µm generally improved gold recovery for agitated cyanidation (bottle roll testing) by between 19.6% and 23.1%. Improvement was higher (33.6%) for the GUN_L6 composite. Milling/cyanidation gold recoveries ranged from 68.0% to 92.1%. Silver recoveries for milling/cyanidation ranged from 48.8% to 77.4%.

Table 13.77 - Summary Results, Column Leach Tests, Gunpoint 2013 Metallurgical Testing

Composite	Feed Size, (P_{80})	Leach/Rinse Time, days	Au Recovery, %	Au Calc'd. Head, g/mt ore	Ag Recovery, %	Ag Calc'd. Head, g/mt ore	Reagent Requirements, kg/mt ore NaCN Cons.	Lime Added	Cement Added
GUN_L3	6.3 mm	205	60.3	1.41	35.8	14.8	2.08	1.3	----
GUN_L3	6.3 mm	5	31.4	1.02	23.1	14.7	<0.07	1.0	----
GUN_L3	3.4 mm	205	59.8	1.02	46.5	12.9	1.64	----	3.0
GUN_L3	3.4 mm	5	48.5	1.03	36.7	14.7	0.13	1.2	----
GUN_L3	HPGR	205	66.9	1.24	49.6	13.3	1.63	----	3.0
GUN_L3	HPGR	5	50.0	1.04	36.4	14.3	0.15	1.4	----
GUN-L4	6.3 mm	205	39.2	1.02	28.6	10.5	1.36	1.6	----
GUN-L4	3.4 mm	205	55.8	0.95	35.1	9.7	1.52	----	3.0
GUN-L4	HPGR	205	56.3	0.96	37.3	10.2	1.42	----	3.0
GUN_L5	6.3 mm	206	42.2	1.02	54.8	14.6	1.40	2.9	----
GUN_L5	HPGR	204	58.8	1.02	66.1	16.5	1.79	----	3.5
GUN-L6	6.3 mm	205	44.3	1.22	38.0	17.1	1.45	2.2	----
GUN-L6	HPGR	204	57.5	1.13	48.4	15.9	1.85	----	3.5

Column test gold recoveries from the four conventionally crushed 6.3mm feeds were 60.3% (GUN_L-3), 42.2% (GUN_L-5), 39.2% (GUN_L-4) and 44.3% (GUN_L-6), in about 205 days of leaching and rinsing.

A comparative column test was conducted on composites GUN_L-3 and GUN_L-4, after conventional crushing to a nominal 3.4mm feed size. Gold recoveries obtained from those tests were 59.8% and 55.8%, respectively. Gold recoveries for composite GUN_L-3 were approximately the same at the 6.3mm and 3.4mm (conventionally crushed) feed sizes. Gold recoveries for the 3.4 mm composite GUN_L-4 feed was approximately 17% higher than obtained at the 6.3mm feed size.

Comparative column tests were conducted on HPGR product (3.4mm feed size) for all four of the composites tested at the 6.3mm size. Gold recoveries obtained from the composite GUN_L-3 and GUN_L-4 HPGR products were 66.9% and 56.3%, respectively, in 205 days of leaching and rinsing. The composite GUN_L-3 gold recovery was approximately 7% higher than obtained from the conventionally crushed 3.4mm feed. The composite GUN_L-4 gold recovery after HPGR grinding was essentially the same as obtained after conventional crushing to the same nominal feed size.

Composites GUN_L-5 and GUN_L-6 were not tested at a conventionally crushed 3.4mm feed size, so it was not possible to directly compare recoveries from conventionally crushed and HPGR product, at a 3.4mm feed size. Gold recoveries obtained from the composite GUN_L-5 and GUN_L-6 HPGR products were 58.8% and 57.5%, respectively, in about 205 days. Although these gold recoveries were 13% to 17% higher than obtained after conventional crushing, the conventionally crushed feeds were significantly coarser (nominal 6.3mm) than the HPGR products (nominal 3.4mm). These results indicate that gold recoveries from composites GUN_L-5 and GUN_L-6 were improved substantially by reducing the feed size, and/or by preferential rock breakage during HPGR grinding. Further testing would be required to determine the degree to which each improved gold recovery.

Gold recovery rates (column tests) were fairly slow, and generally not substantially different for the conventionally crushed and HPGR products. Gold extraction was progressing at a very slow rate when leaching was terminated for all column feeds. Very long commercial heap leach cycles will be required to maximize gold recoveries.

Overall, comparative results between column tests on conventionally crushed and HPGR product samples were inconclusive with respect to the benefits of HPGR grinding. Evaluation of the recovery by size fraction data (head and tail screen analyses) from those tests, were also inconclusive, but tended to indicate that the improvement in gold recoveries obtained by HPGR grinding resulted more from the finer particle size distributions of the HPGR products, than from a preferential breakage of rock particles during HPGR grinding. Test results did show, however, that gold recoveries obtained at a 3.4mm feed size were substantially higher than obtained at a 6.3mm feed size.

Comparison between simulated heap leach and milling/cyanidation test results show that, for the two Bear Creek hanging wall unoxidized composites (GUN_L3 and GUN_L5), milling/cyanidation treatment (75µm) resulted in a gold recovery that was 9% to 15% higher than obtained by column testing (considering the maximum CLT recovery). In the case of the two Bear Creek footwall composites (GUN_L4 and GUN_L6), the improvement in gold recovery was somewhat higher (14% - 17%). Column leach tests were not conducted on the oxide and Main Zone composites, so a similar comparison was not possible. Crushed ore bottle roll testing on composites GUN_L-3, GUN_L-4, GUN_L-5 and GUN_L-6 showed that those tests were of limited value for predicting column test gold recoveries, as the bottle roll test recoveries consistently underestimated the column test recoveries. Consequently, the available bottle roll data from the oxide and Main Zone composites probably won't be useful for predicting crushed ore heap leach gold recoveries.

Three of the four HPGR column leached residues displayed solution ponding problems when rinsed with fresh water after column leaching. Of the HPGR product column tests, only the composite GUN_L-3 test did not display ponding problems. That column residue was selected for load/permeability testing after leaching. Load/permeability testing on the composite GUN_L-3 residue showed a hydraulic conductivity of 7.8 x 10^{-3} cm/sec. at a simulated heap stack height of 25m. That was significantly higher than the equivalent solution application rate used during leaching (3.3 x 10^{-4} cm/sec.). This result indicates acceptable permeability (up to a 25 m stack height) for the one HPGR product tested. The poor permeability (solution ponding) during column testing of three out of four HPGR products indicate that further optimization of agglomerating conditions would be required if heap leaching of the HPGR product is to be considered.

Conclusions reached from the testing program were summarized as follows.

→ The Bear Creek composites were only moderately amenable to agitated cyanidation at a 1.7 mm feed size, but showed significant increases in gold recovery when milled to 80%-75µm.

→ The Bear Creek Hanging wall oxide composite was more readily amenable to agitated cyanidation at the 1.7 mm and 75 µm feed sizes with recoveries of 72.1% and 92.1%, respectively.

→ All of the Bear Creek Zone and the Main Zone composites tested were sensitive to feed size with respect to gold and silver recovery.

→ Gold recoveries from agitated cyanidation at the 1.7mm and 75µm feed sizes were similar for the Main Zone composite and the Bear Creek Hanging wall GUN_L-3 composite.

→ Agitated cyanidation gold recovery rates were rapid at the milled feed size (75µm) but much slower at coarser feed sizes.

→ The Talapoosa Bear Creek Zone composites were moderately amenable to simulated heap leach testing at relatively fine feed sizes.

→ A comparison between HPGR grinding and conventional crushing at the same nominal feed size was not conclusive with respect to the benefits of HPGR grinding as HPGR grinding improved gold recovery for only one of the two composites.

→ The observed improvement in gold recovery likely resulted in a moderately finer particle size distribution for the HPGR product, compared to conventional crushing to the same nominal feed size.

→ Three of the four HPGR column tests showed poor permeability during rinsing which indicates the need for further optimization of agglomerating conditions.

→ Cyanide consumptions, generally, were low for the agitated leach tests but were higher for the column tests due in large part to long leach cycles employed.

→ The lime or cement added during column test leaching in some cases, resulted in less than optimum pH control, indicating the need for higher base additions.

14 MINERAL RESOURCES ESTIMATE

14.1 INTRODUCTION

Tetra Tech completed a resource estimation of the Talapoosa deposit. Mr. Todd McCracken, P. Geo, the QP who completed the resource estimation with Tetra Tech remains as the QP with WSP. The effective date of the resource is March 24, 2015. WSP has validated the resource model and has determined that there is no material change to the resource model.

WSP has reviewed the annual and quarterly financial reports posted by Gunpoint on SEDAR. The expenditure amount on the Property was limited to some surface geological mapping and sampling, the metallurgical test work and salaries. Expenditure levels do not support any size of drilling program since the completion of the 2012 drilling by Gunpoint. The metallurgical test work completed in 2014 confirms the existing recoveries on the Project are valid and that there is no material change to the resources.

Historically, the Talapoosa deposit is made up of four different areas: Bear Creek, Main Zone, East Hill and Dyke Adit. The Bear Creek Zone has been subdivided into a Hanging- Wall and Footwall zones.

14.2 DATABASE

Gunpoint maintains all borehole data in a MineSight® data format containing header, survey, assays and lithology tables. A copy of the header, survey, lithology and assays were provided to Tetra Tech between July 4 and August 28, 2012.

The files provided to Tetra Tech contained the data for 602 boreholes. The dataset were for all surface boreholes on the Property. There are 40,723 gold assays and 36,601 silver assays within the database (Table 14.1). All boreholes that occurred outside the limits of the Talapoosa deposit were removed from the dataset in order to concentrate on Talapoosa.

Table 14.1 - Talapoosa Diamond Drill Database

	Talapoosa				
	Holes in Project Area	Holes Used in Resource			
No. of Drillholes	**602**	**545**			
Field	**N Samples**	**Minimum**	**Maximum**	**Mean**	**Standard Deviation**
Length (ft)	44707	0.5	1400	5.68	13.995
Au (oz/ton)	40723	-2	5.389	-0.108	0.478
Ag (oz/ton)	36601	-2	41.756	-0.201	1.024

The Talapoosa database was reconstructed from scratch in 2008 by MDA (Ristorcelli et al. 2010). MDA continues to maintain the database on behalf of Gunpoint and updated the database with the recent 2011 diamond drilling results completed by Gunpoint. Intervals within the database that were not assayed contained a -2 value. These values were replaced by Tetra Tech with an absent field.

The database has all significant data, and each sample interval is assigned an integer code representation that reflects that particular assays quality. Considerations for whether or not a sample could be used included demonstrated contamination during drilling, no QA/QC and no lab certificates, or obvious bias in the sample campaign. This "USE" code was "1" for usable and "0" for not usable. Of the total assays, 23,828 gold assays and 24,261 silver assays were considered usable.

The resource estimation was conducted using Datamine™ Studio 3 version 3.21.7164.0.

14.3 SPECIFIC GRAVITY

Gunpoint collected a total of 310 specific gravity measurements from various rock types, alteration types and quartz veining content. Gunpoint collected pieces of diamond drill core and weighted the material dry and then suspended in water to determine the specific gravity (Figure 14.1).

Figure 14.1 - Specific Gravity Measurement Scale

Table 14.2 summarizes the results of the specific gravity measurements collected by Gunpoint. A conversion factor of 0.031214 was used to convert the metric g/cm^3 to tons/ft^3. Analysis of specific gravity data was done in the context of lithology and alteration and oxidation.

Table 14.2 - Talapoosa Specific Gravity Summary

Rock Type	Specific Gravity	t/ft	Samples
Host Rock – Argillic Altered	2.32	0.072	181
Quartz Vein or Breccia	2.50	0.078	81
Oxidized Host Rock	2.14	0.067	48

A historic Talapoosa density database totaling 83 samples dates back to 2008 and was not considered in the determination of current specific gravity values. The coated immersion method was used for the measurements collected historically.

WSP recommends that Timberline continue to collect specific gravity measurements from the various rocks types and grade distributions in order to build up the data set. At a minimum, 2% of the data set should have specific gravity measurements. Currently, the specific gravity data set represents 1.3% of the gold assay used in the resource estimate.

14.4 GEOLOGICAL INTERPRETATION

Several 3D wireframe models of mineralization were provided by Gunpoint in AutoCAD format and imported into Datamine™ software by Tetra Tech. The basis for each wireframe included a minimum downhole width of 5 ft, a minimum waste inclusion of 1 ft downhole, and a minimum grade of 0.01 oz/ton gold. A second large wireframe surrounding the high grade vein systems was constrained by the structural faults of the Project. The higher-grade vein wireframes are located within the lower-grade wireframe and represent a discrete, higher-grade domain.

Sectional interpretations were in Datamine™ software, and these interpretations were linked with tag strings and triangulated to build 3D solids. Table 14.3 tabulates the solids and associated volumes. The solids were validated in the Datamine™ software and no errors were found.

Table 14.3 - Wireframe Summary

Zone	Wireframe Dimensions						Volume (ft^3)
	Minimum X	Maximum X	Minimum Y	Maximum Y	Minimum Z	Maximum Z	
Bear Creek Hanging-Wall Vein	303966.06	305833.46	1712178.51	1713217.08	4476.92	5370.38	104,010,965.5
Bear Creek Footwall Vein	303909.09	305859.86	1712363.46	1713563.77	4370.82	5354.27	64,305,299.2
Main Vein	304142.65	305956.21	1712550.81	1714229.73	4380.66	5543.61	79,122,762.1
East Hill Vein	306335.44	307862.28	1712233.14	1712983.95	4905.43	5532.91	23,591,416.9
Dyke Adit Vein	302713.06	304136.87	1713200.03	1714779.00	5077.74	5709.89	25,981,203.5
Bear Creek Hanging-Wall Zone	303591.27	305869.35	1712177.83	1713357.19	4462.62	5414.89	282,393,020.0
Bear Creek Footwall Zone	303763.11	305911.19	1712308.9	1713572.29	4303.16	5382.24	539,566,934.4
Main Zone	303946.05	306083.03	1712364.15	1714249.41	4250.55	5565.8	459,028,036.8
East Hill Zone	306200.96	307941.86	1712013.36	1713096.87	4795.56	5554.29	218,257,322.5
Dyke Adit Zone	302381.14	304253.95	1712738.04	1714894.06	4815.42	5851.86	685,016,156.6

The zones of mineralization interpreted for each area were generally contiguous however, due to the nature of the mineralization there are portions of the wireframe that have grades less than 0.01 oz/ton gold, yet are still within the mineralizing trend.

All wireframes were trimmed to the topography in order to avoid any estimation of material above surface.

The wireframes extend at depth and along strike beyond the last borehole. This is to provide target areas for future exploration. The resource model will not estimate grades into the full volume of the wireframes due to sheer size of the wireframes.

Figures 14.2 to 14.6 are oblique views of the higher-grade vein mineral wireframes while Figures 14.7 to 14.11 illustrate oblique views of the low-grade mineral wireframe.

Figure 14.2 - Oblique View Main Vein

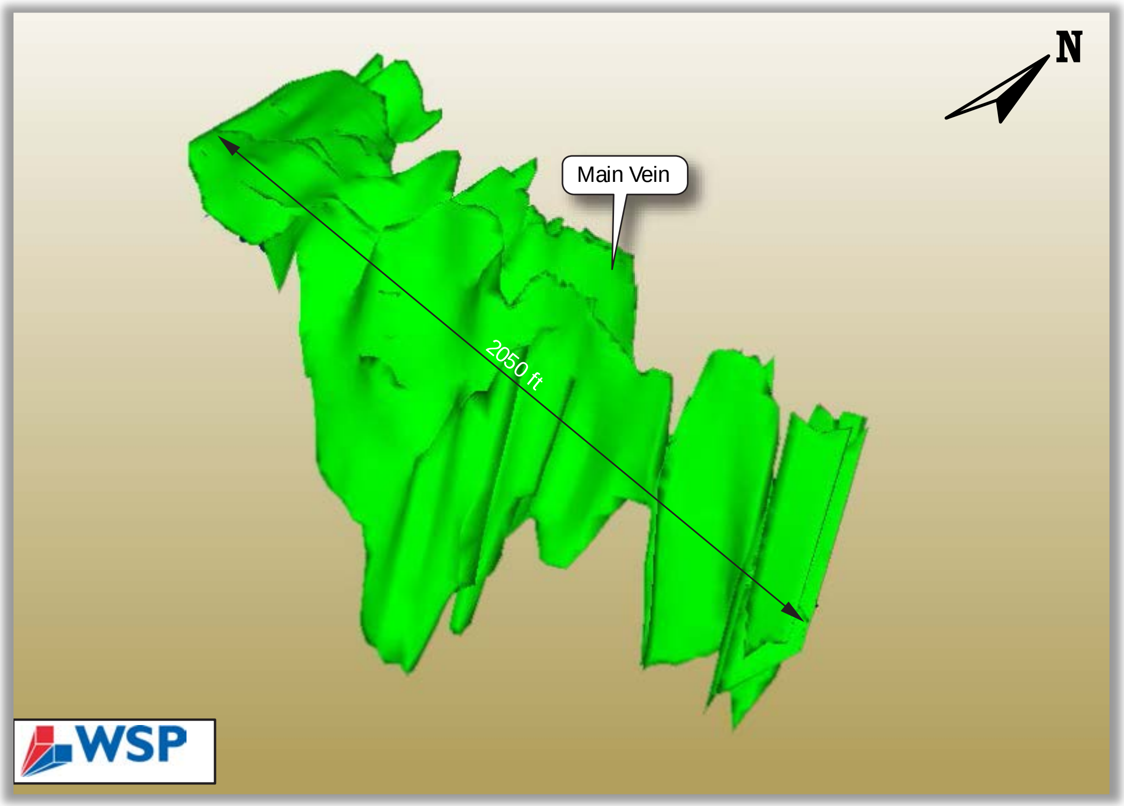


Figure 14.3 - Oblique View Bear Creek Footwall Vein

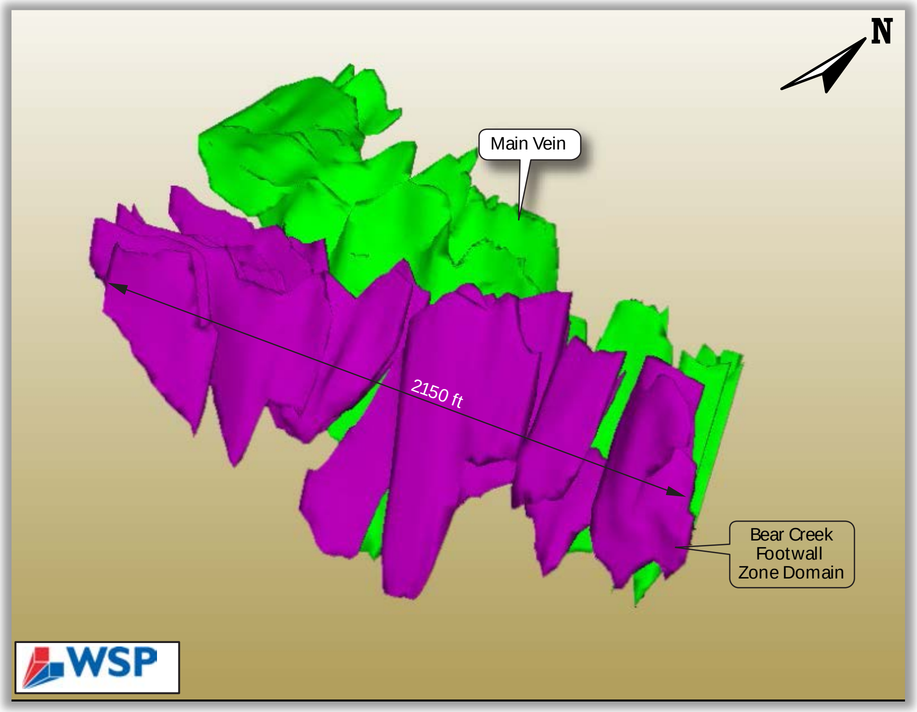


Figure 14.4 - Oblique View Bear Creek Hanging Wall Vein

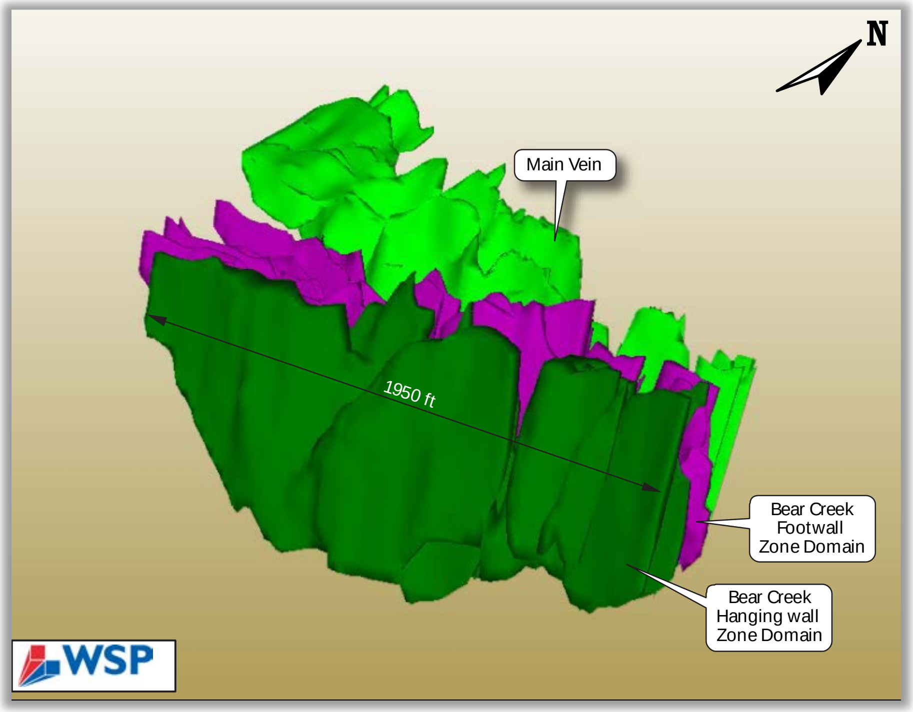


Figure 14.5 - Oblique View Dyke Adit Vein

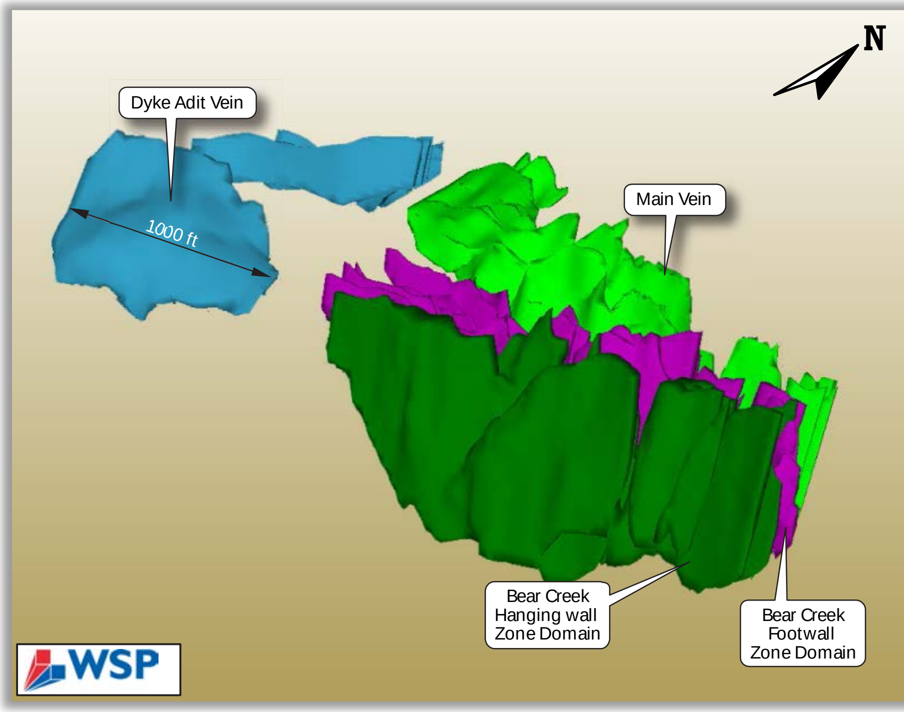


Figure 14.6 - Oblique View East Hill Vein

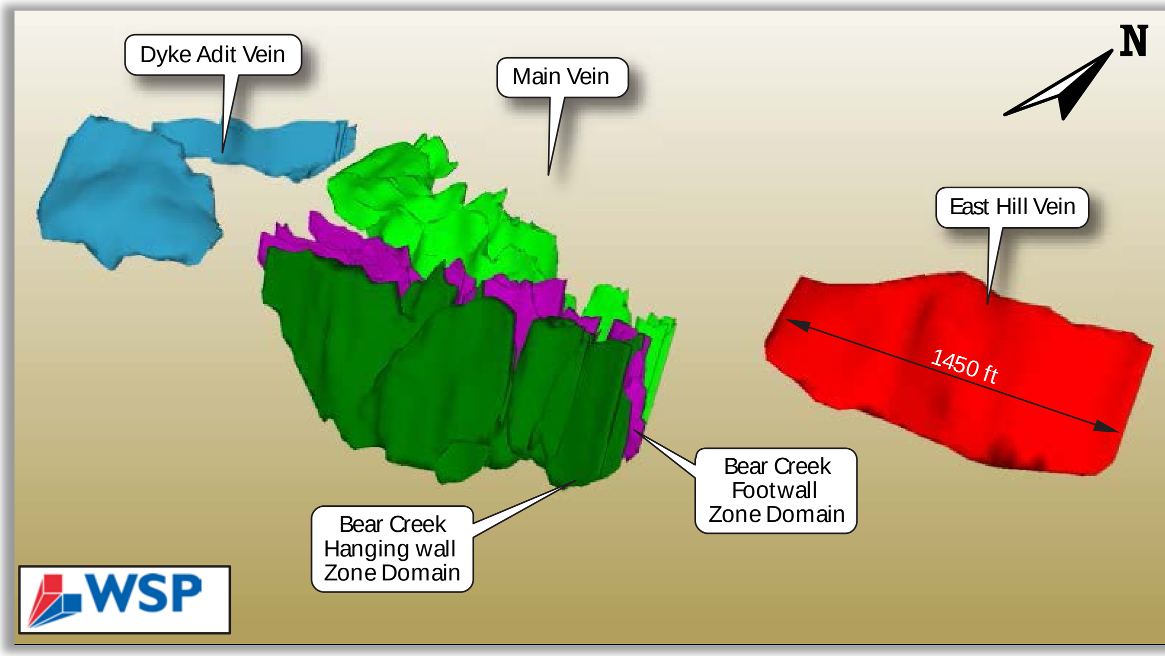


Figure 14.7 - Oblique View Main Zone

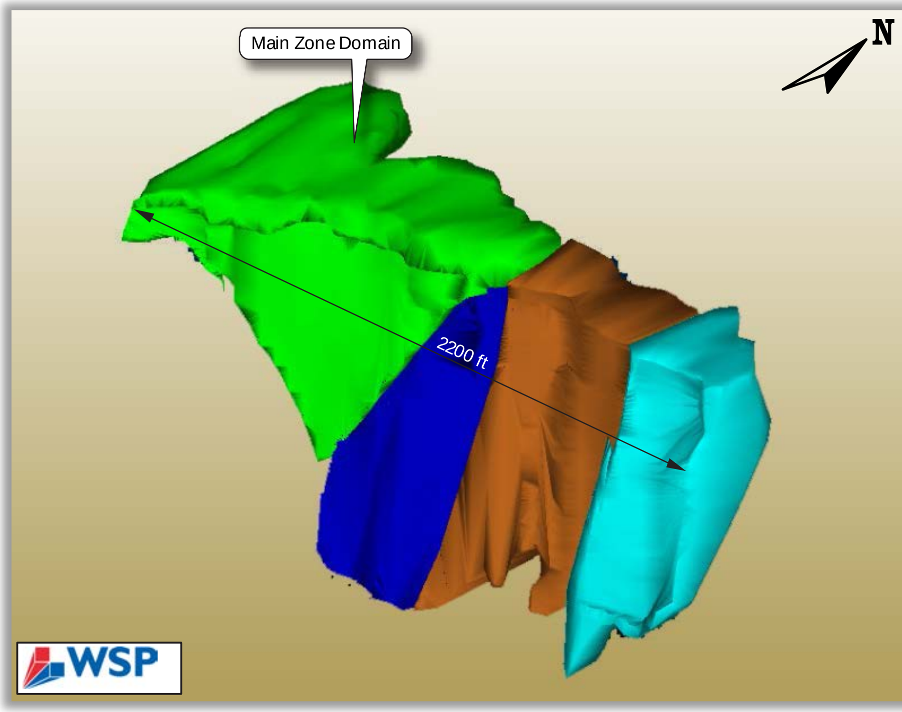


Figure 14.8 - Oblique View Bear Creek Footwall Zone

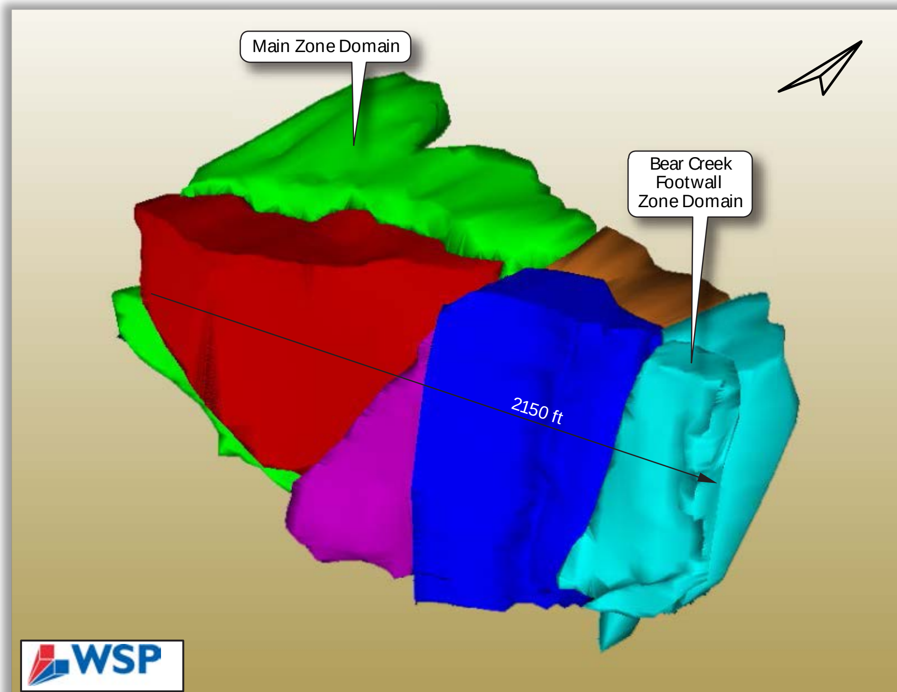


Figure 14.9 - Oblique View Bear Creek Hanging Wall Zone

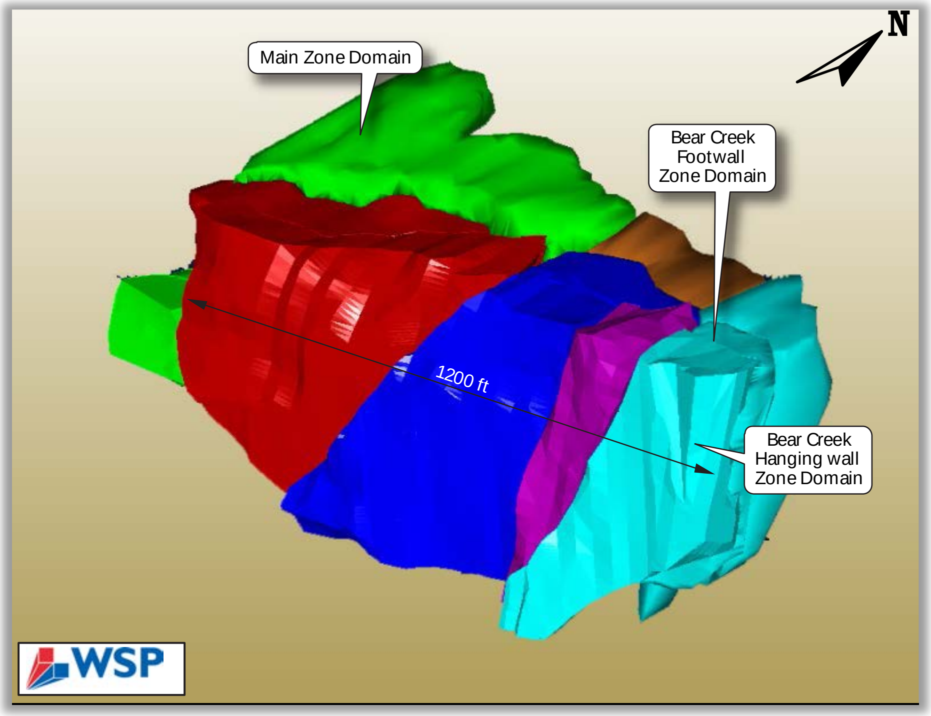


Figure 14.10 - Oblique View Dyke Adit Zone

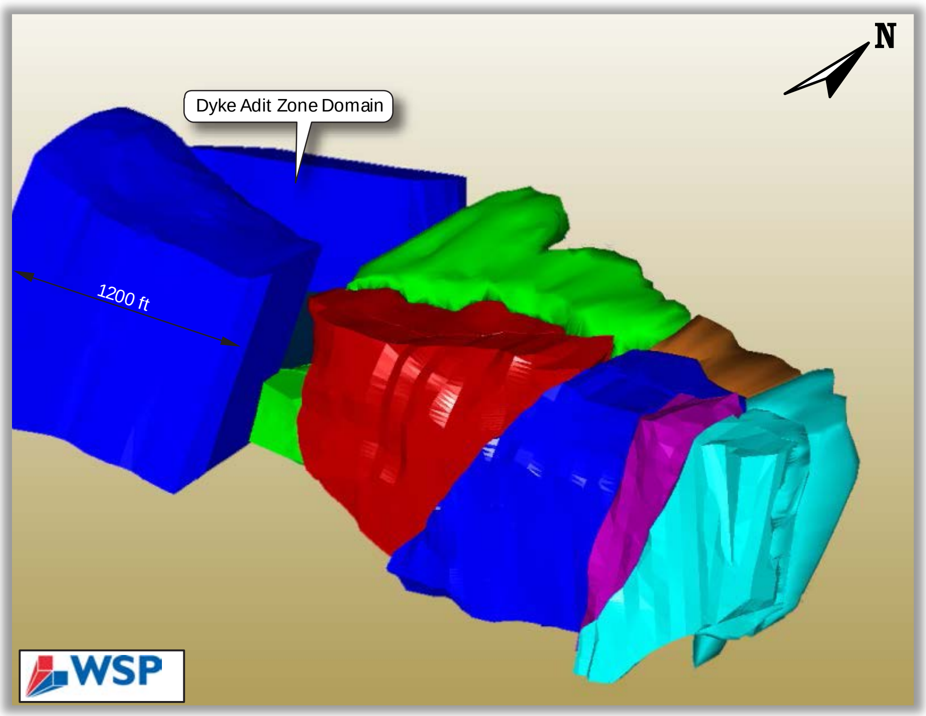
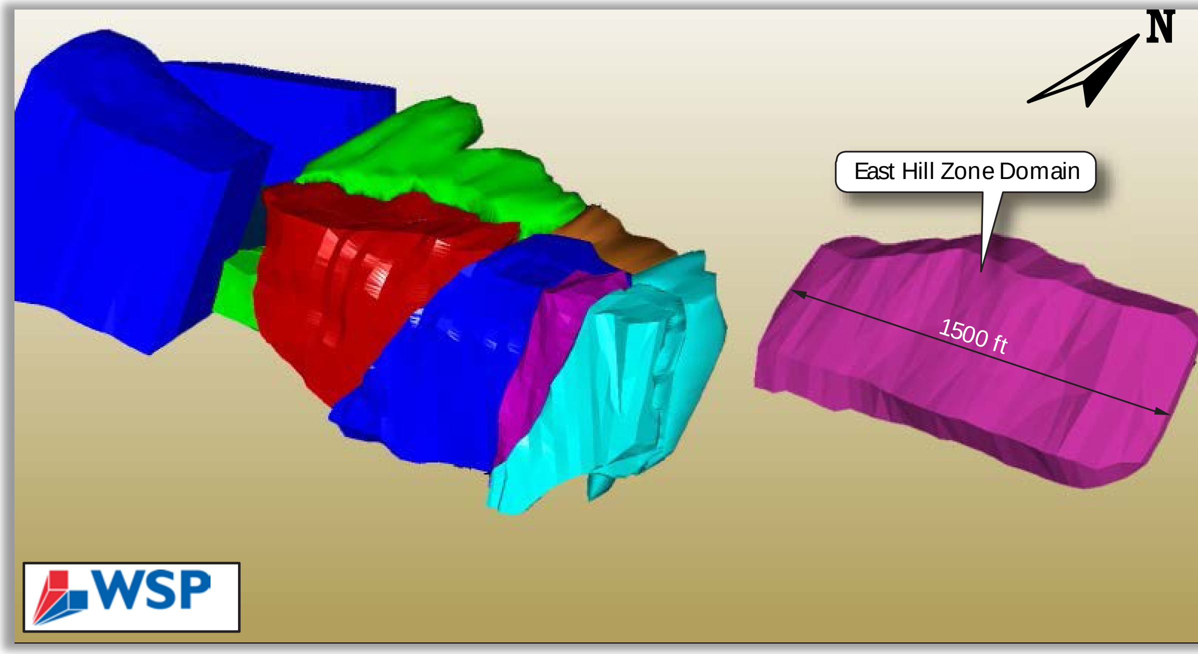


Figure 14.11 - Oblique View East Hill Zone



14.5 EXPLORATORY DATA ANALYSIS

14.5.1 ASSAYS

The portion of the deposit included in the mineral resource was sampled by a total of 23,828 gold assays and 24,261 silver assays. The assay intervals within each zone were captured using a Datamine™ macro into individual borehole files. These borehole files were reviewed to ensure all the proper assay intervals were captured. Table 14.4 summarizes the basic statistics for the assays in the various Talapoosa domains wireframes. Figure 14.12 to Figure 14.21 are the frequency histogram plots for gold in each of the mineral domains.

The non-assayed intervals were assigned void (-) value. Tetra Tech believes that non- assayed material should not be assigned a zero value, as this does not reflect the true value of the material.

Table 14.4 - Summary of Talapoosa Borehole Statistics

Zone	Field	N Samples	Minimum	Maximum	Mean	Standard Deviation
Bear Creek Footwall Vein	Length	1,540	0.50	10.00	5.57	1.80
	Au	1,535	0.0005	0.8150	0.0341	0.0488
	Ag	1,534	0.0015	20.2410	0.5339	1.1447
Bear Creek Hanging-Wall Vein	Length	3,041	1.00	10.00	5.65	1.91
	Au	2,987	0.0001	2.4092	0.0409	0.0827
	Ag	2,987	0.0025	23.1105	0.5331	1.0626
Main Vein	Length	1,671	1.00	16.00	5.25	1.26
	Au	1,662	0.0001	1.0060	0.0326	0.0533

Zone	Field	N Samples	Minimum	Maximum	Mean	Standard Deviation
	Ag	1,641	0.0022	41.7560	0.4205	1.4843
Dyke Adit Vein	Length	354	4.00	5.00	4.99	0.11
	Au	352	0.0010	1.0260	0.0577	0.1046
	Ag	354	0.0030	9.6070	0.8453	1.2274
East Hill Vein	Length	282	5.00	5.00	5.00	-
	Au	282	0.0010	0.3340	0.0262	0.0308
	Ag	282	0.0100	6.3400	0.4469	0.6268
Bear Creek Footwall Zone	Length	5,088	1.00	10.70	5.47	1.64
	Au	5,028	0.0003	5.3890	0.0138	0.0826
	Ag	4,967	0.0005	27.5500	0.2193	0.6605
Bear Creek Hanging-Wall Zone	Length	4,454	1.00	10.00	5.68	1.79
	Au	4,211	0.0001	2.3560	0.0114	0.0385
	Ag	4,212	0.0015	20.1620	0.1830	0.4520
Main Zone	Length	4,690	1.00	13.50	5.31	1.29
	Au	4,493	0.0001	0.6860	0.0074	0.0203
	Ag	4,515	0.0015	9.0400	0.1330	0.3356
Dyke Adit Zone	Length	2,454	1.00	15.00	5.03	0.74
	Au	1,787	0.0005	0.1690	0.0056	0.0103
	Ag	2,310	0.0015	4.6300	0.1097	0.2489
East Hill Zone	Length	1,511	5.00	5.00	5.00	-
	Au	1,491	0.0005	0.2175	0.0053	0.0118
	Ag	1,459	0.0015	2.3250	0.0700	0.1387

Figure 14.12 - Bear Creek Hanging-Wall Vein Gold Histogram Plot

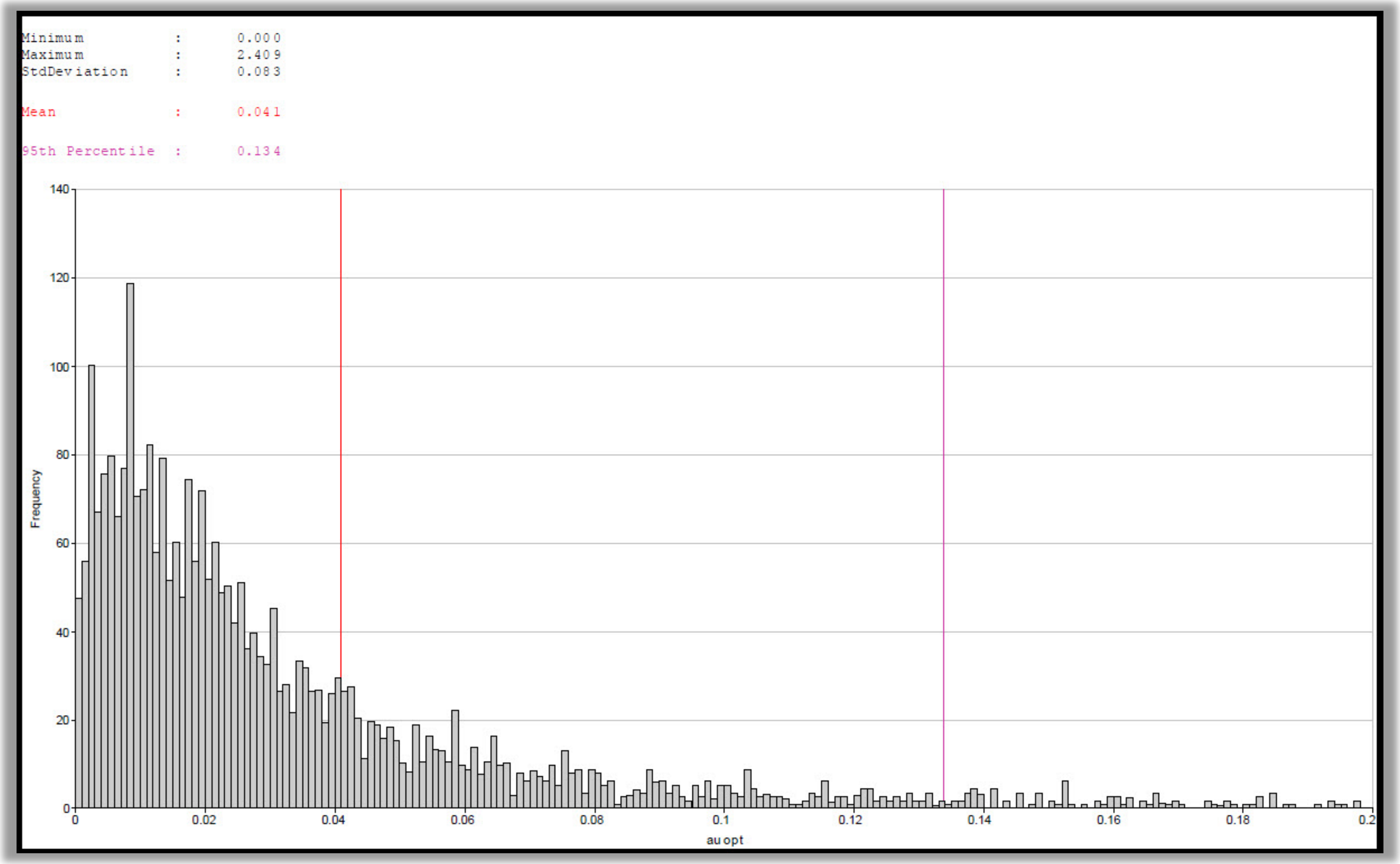


Figure 14.13 - Bear Creek Footwall Vein Gold Histogram Plot

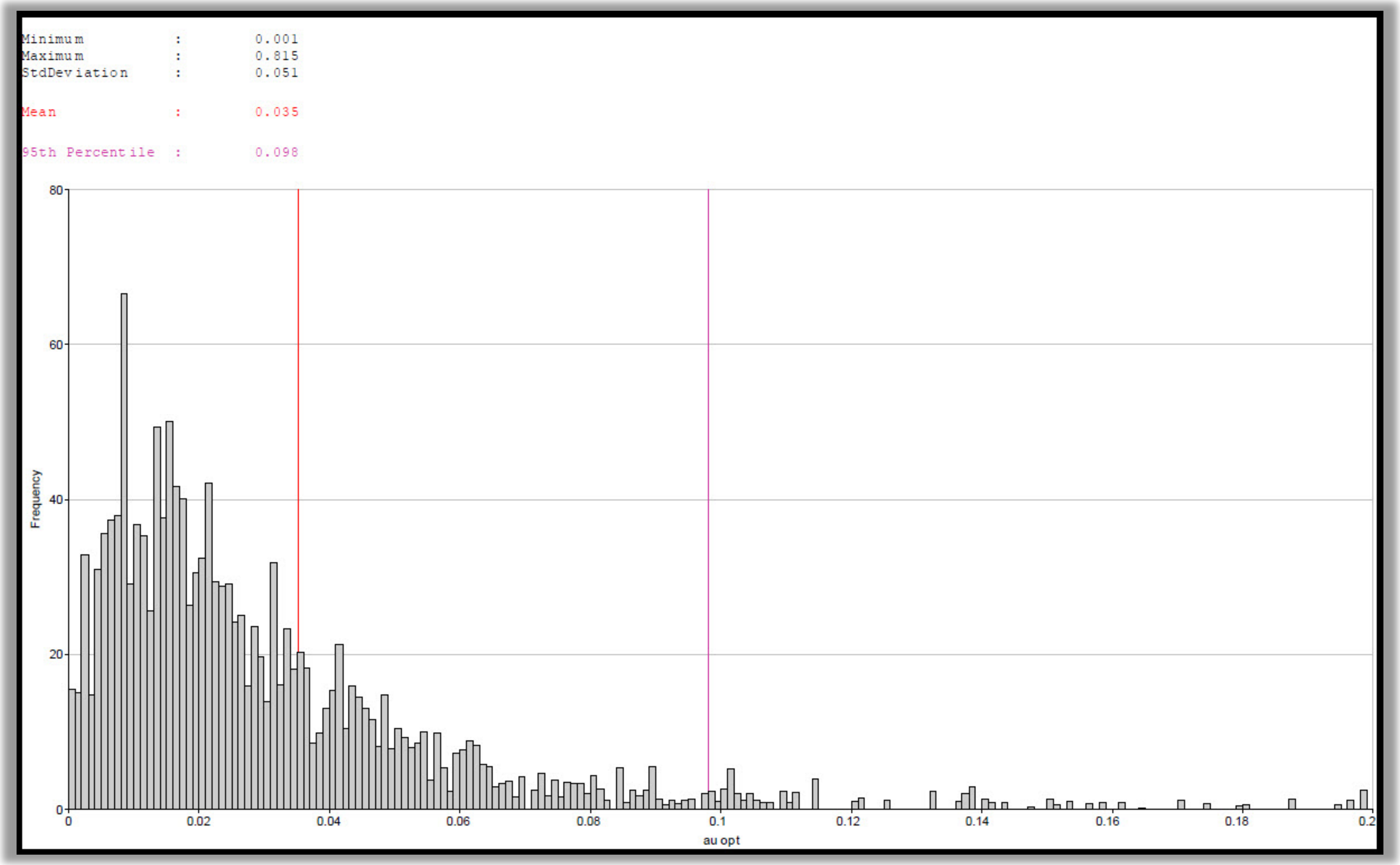


Figure 14.14 - Main Vein Gold Histogram Plot

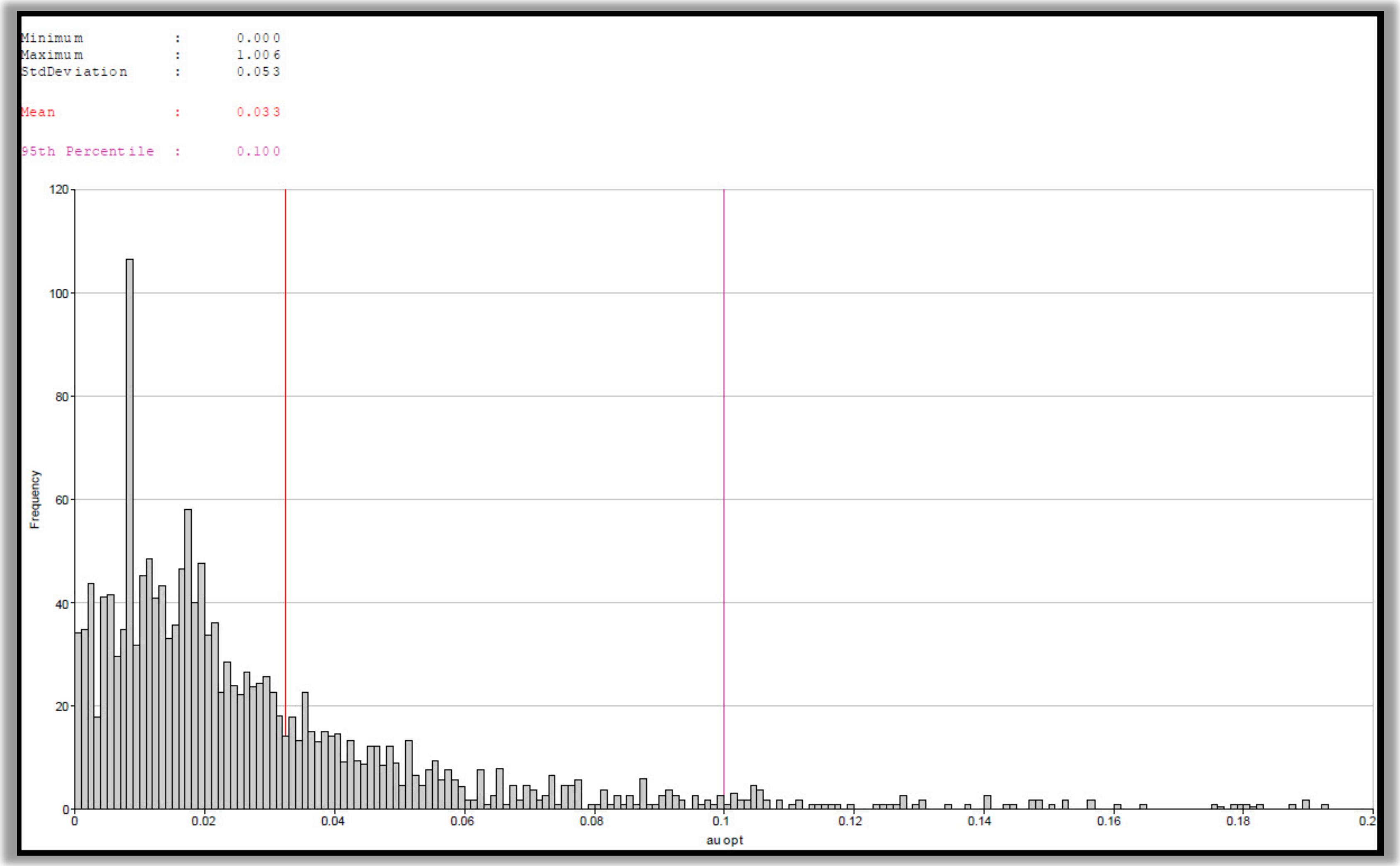


Figure 14.15 - Dyke Adit Vein Gold Histogram Plot

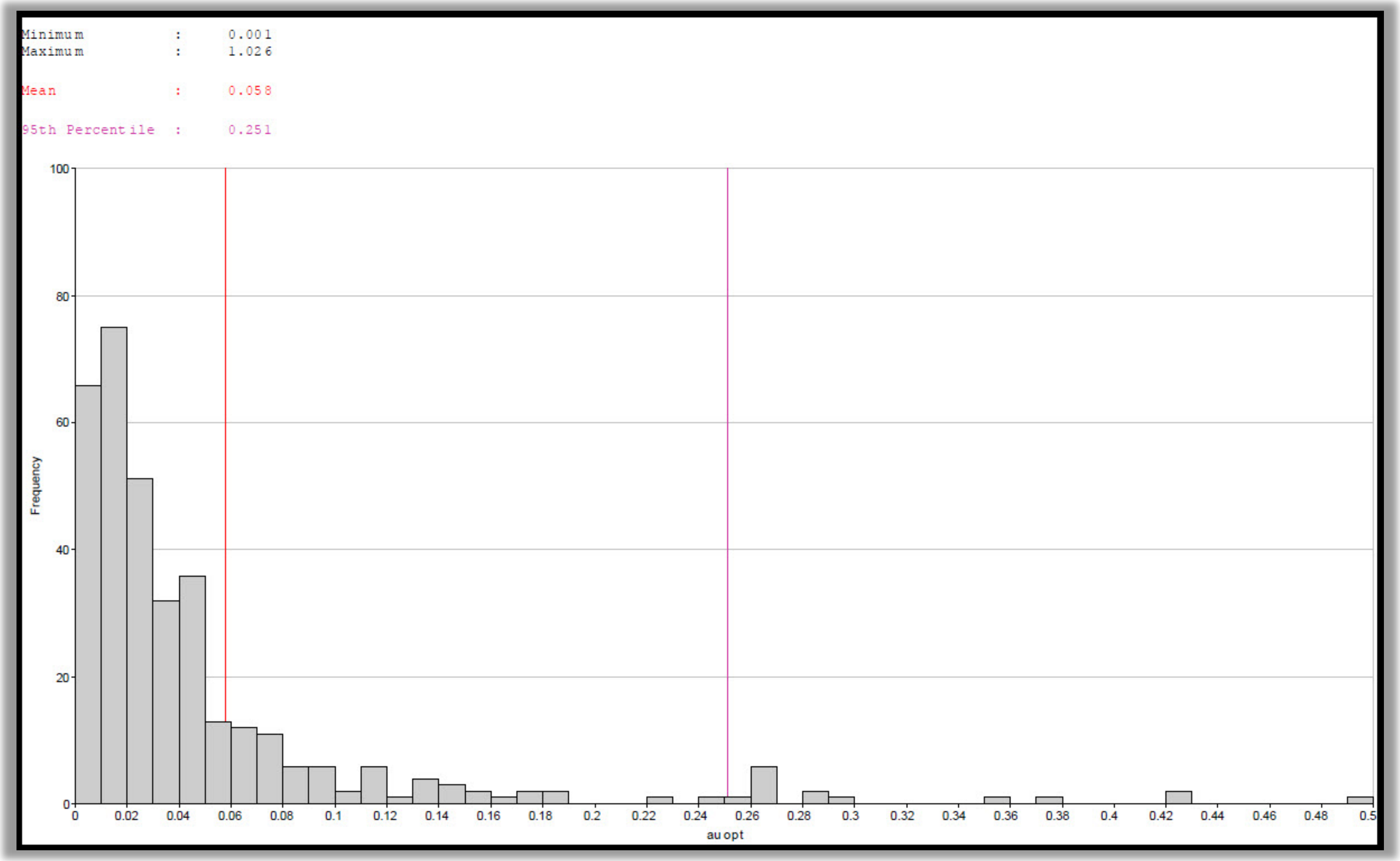


Figure 14.16 - East Hill Vein Gold Histogram Plot

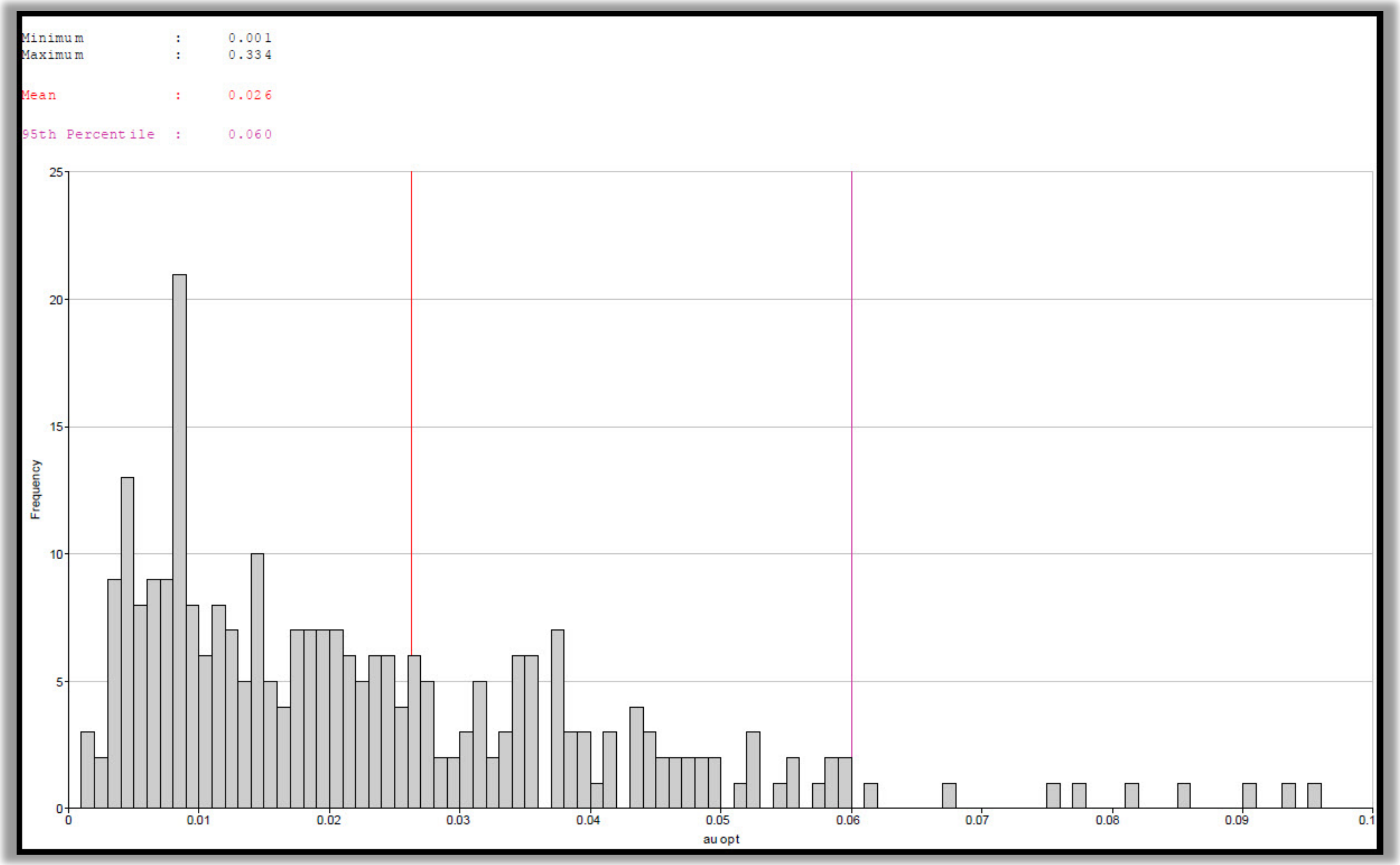


Figure 14.17 - Bear Creek Hanging Wall Zone Histogram Plot

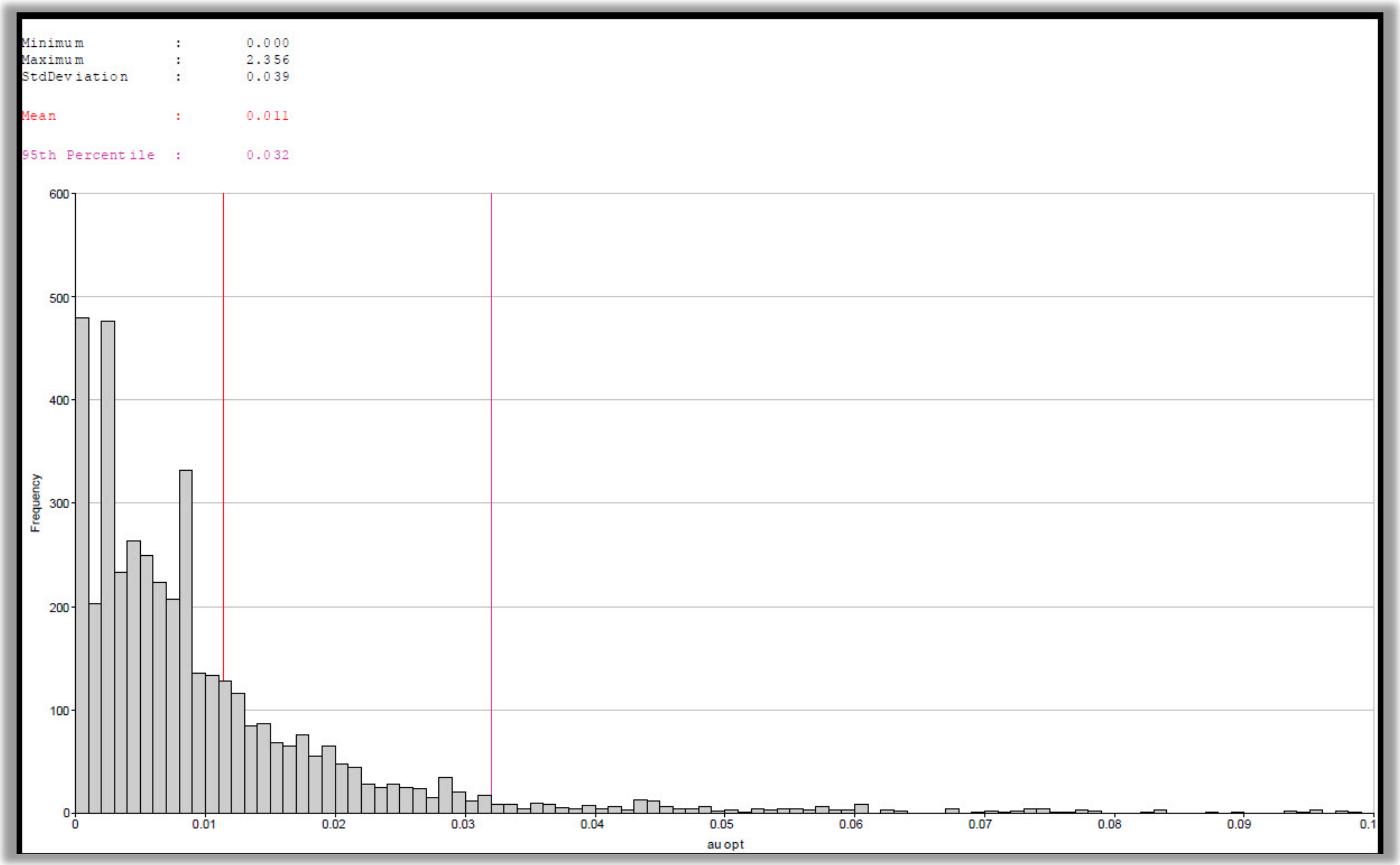


Figure 14.18 - Bear Creek Footwall Zone Gold Histogram Plot

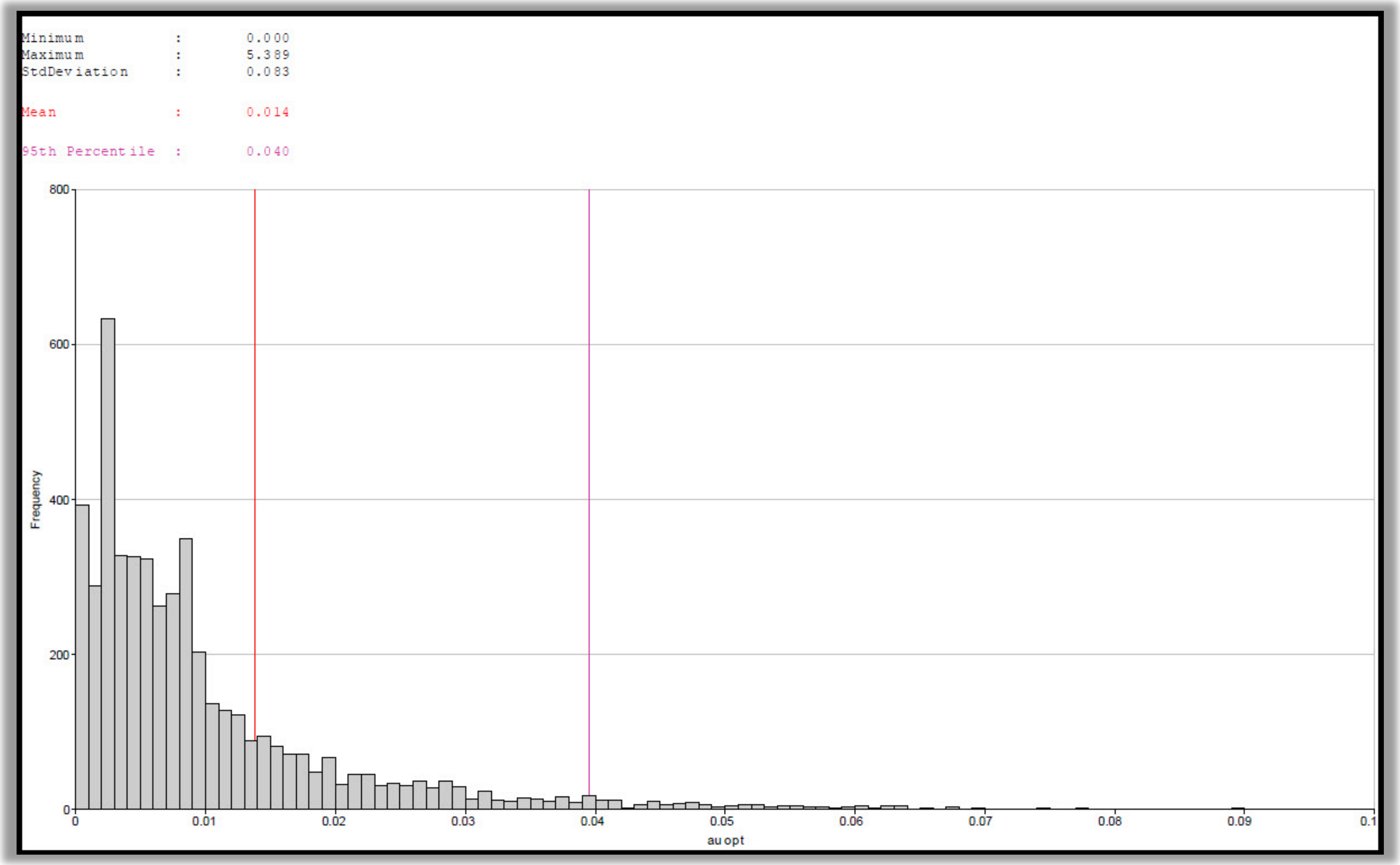


Figure 14.19 - Main Zone Gold Histogram Plot

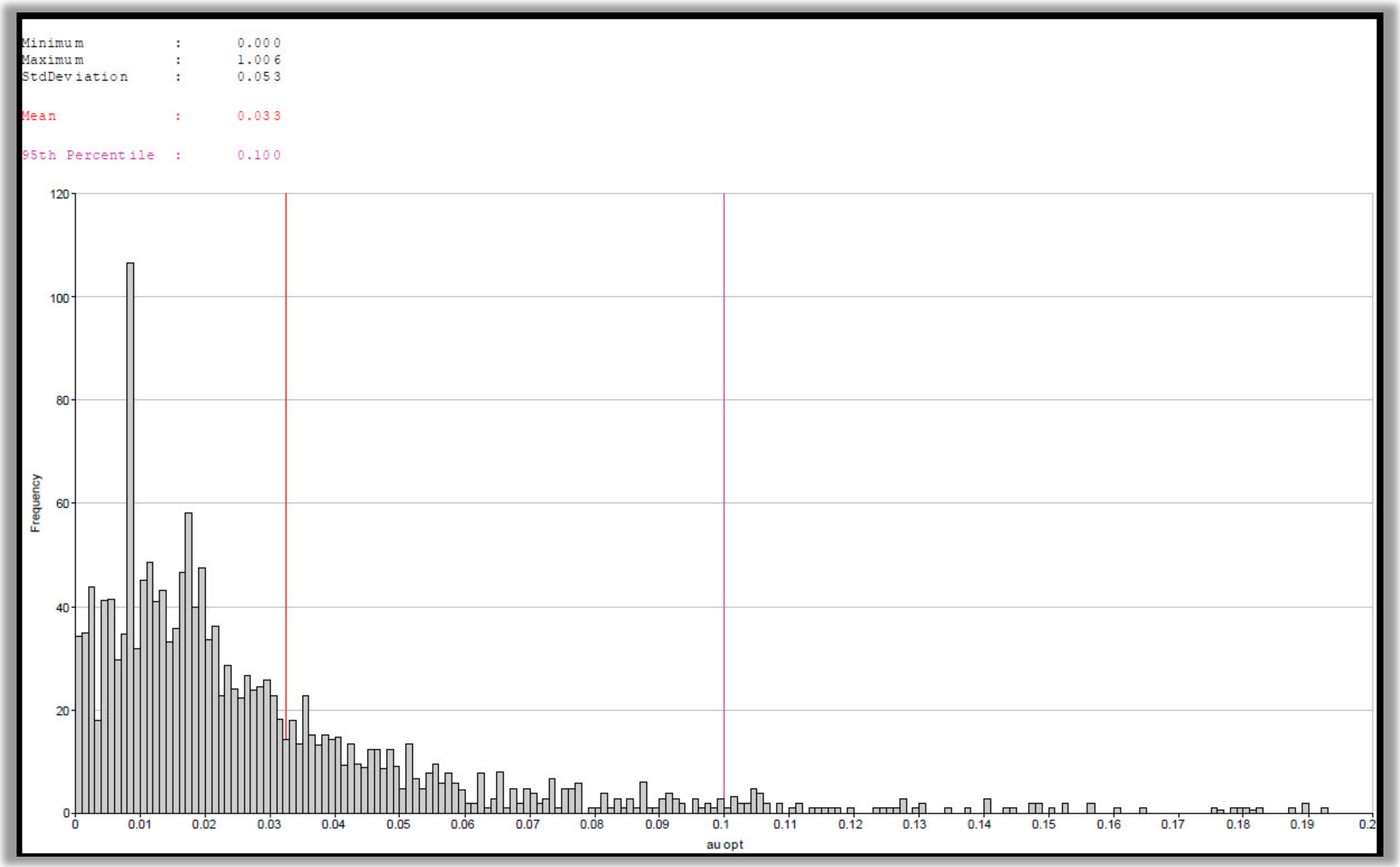


Figure 14.20 - Dyke Adit Zone Gold Histogram Plot

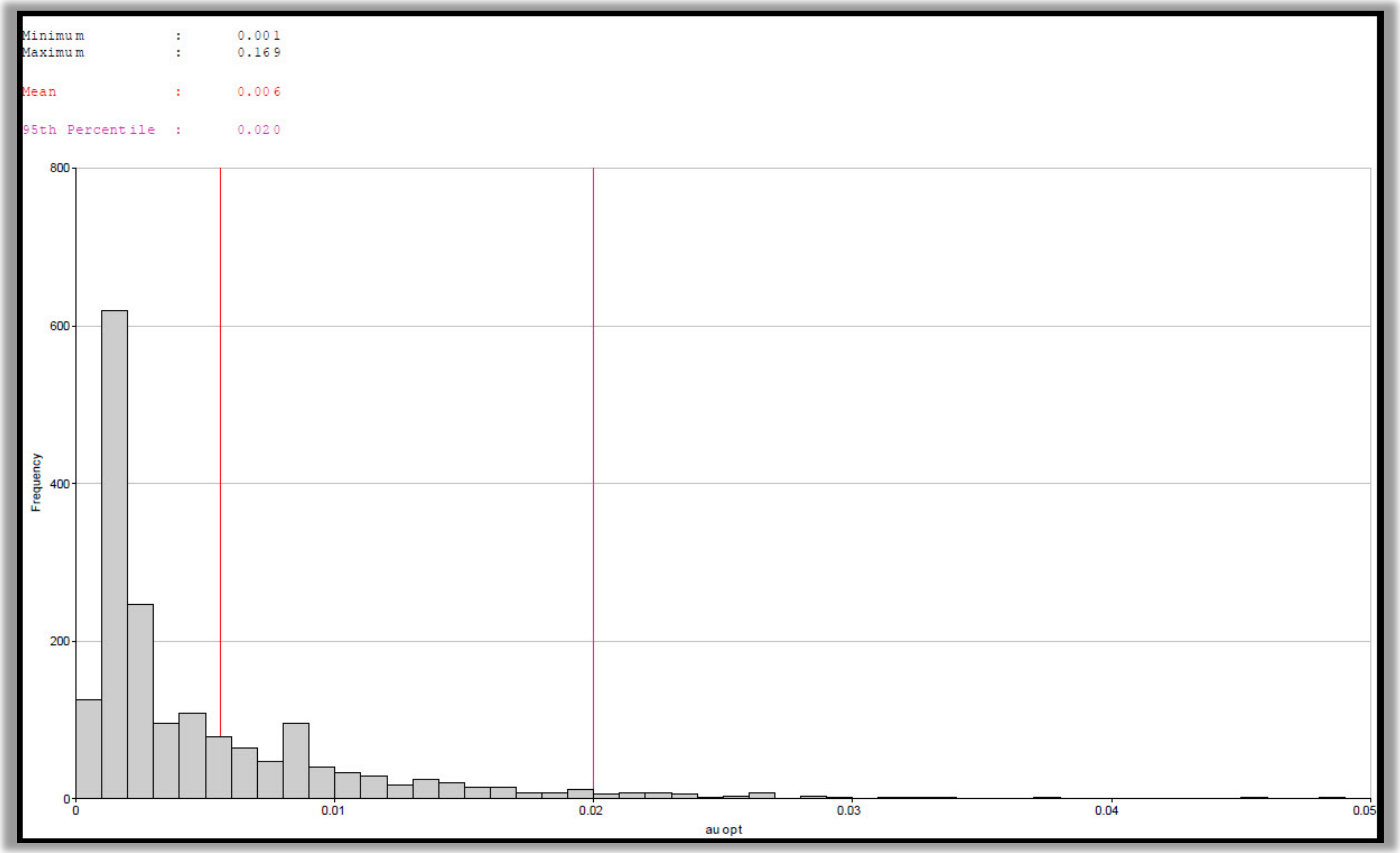
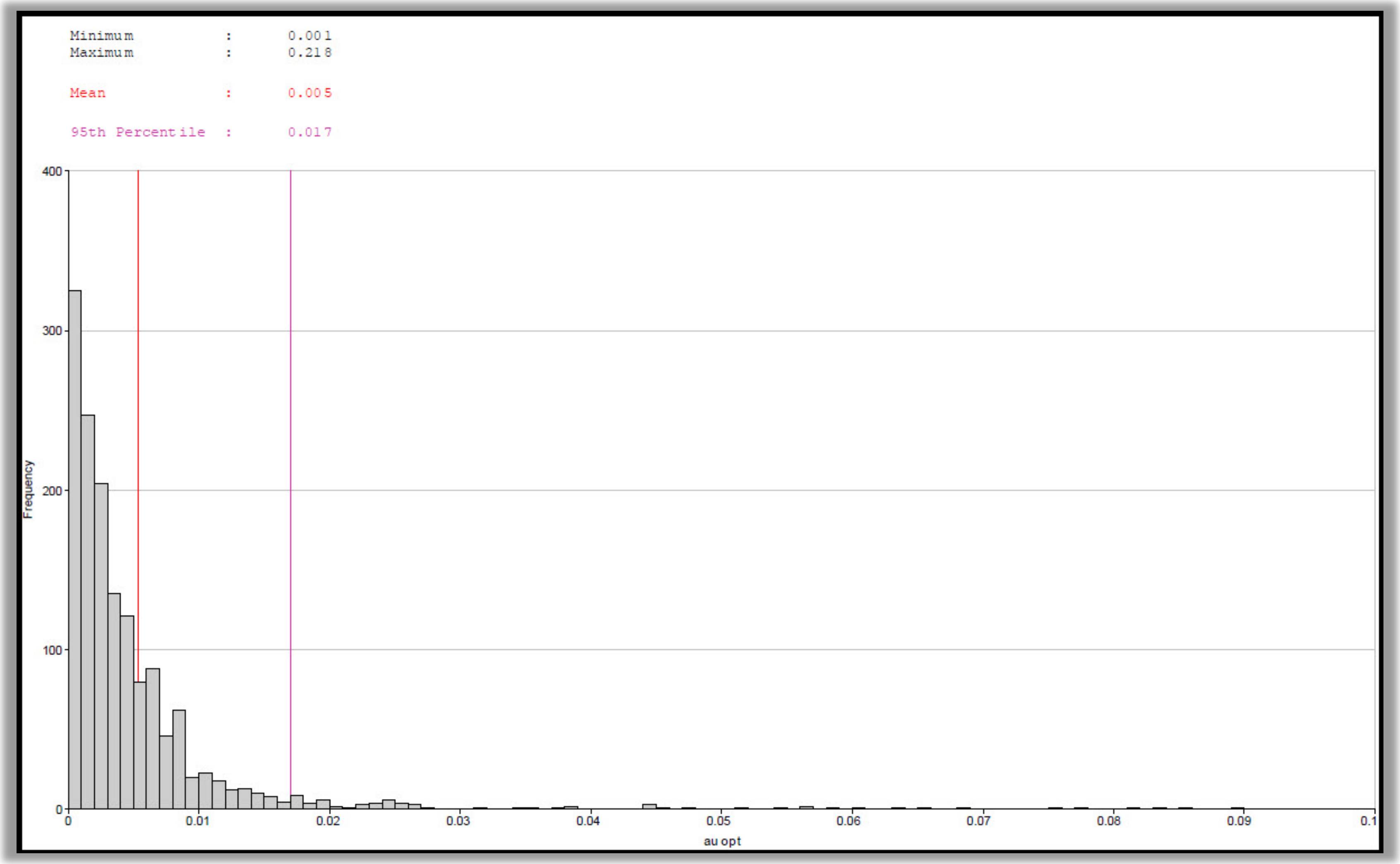


Figure 14.21 - East Hill Zone Gold Histogram Plot



14.5.2 GRADE CAPPING

Raw assay data for each of the wireframes was examined individually to assess the amount of metal that is at risk from high-grade assays. The Datamine™ Decile function was used to assist in the determination if grade capping was required on each of the elements in the dataset by using the Parrish analysis (Parrish 1997).

When using the Parrish analysis, the following criteria may warrant grade capping:

- The top decile of 90 to 100% contains more than 40% of the metal content, or
- The top decile of 90 to 100% has more than twice the metal content of the next decile at 80 to 90%, or
- The top percentile of 99 to 100% has more than 10% of the metal content, or
- The top percentile of 99 to 100% has more than twice the metal content of the next percentile of 98 to 99%.

Table 14.5 summarizes the results of the Parrish analysis. The results of the analysis indicate that capping of gold and silver maybe required with the dataset.

In addition to the Parrish analysis, the spatial distribution of the samples was reviewed to determine if the population of anomalous samples are in close proximity and may represent a subset within the data. The review of the data resulted in capping of gold at 0.686 oz/ton and silver at 9.60 oz/ton within the Talapoosa data set.

Table 14.5 - Grade Capping Summary

Zone	Field	N Samples	Minimum	Maximum	Mean	Standard Deviation	No. of Samples Capped	% of Dataset Capped	% Change of Mean After Capping
Footwall Vein	Length	1,540	0.50	10.00	5.57	1.80	-	-	-
	Au	1,535	0.0005	0.8150	0.0341	0.0488	-	-	-
	Aucap	1,535	0.0005	0.6860	0.0341	0.0477	1	0.1	0.2
	Ag	1,534	0.0015	20.2410	0.5339	1.1447	-	-	-
	Agcap	1,534	0.0015	9.6000	0.5179	0.9461	4	0.3	3.0
Hanging-Wall Vein	Length	3,041	1.00	10.00	5.65	1.91	-	-	-
	Au	2,987	0.0001	2.4092	0.0409	0.0827	-	-	-
	Aucap	2,987	0.0001	0.6860	0.0398	0.0613	5	0.2	2.8
	Ag	2,987	0.0025	23.1105	0.5331	1.0626	-	-	-
	Agcap	2,987	0.0025	9.6000	0.5254	0.9584	5	0.2	1.4
Main Zone Vein	Length	1,671	1.00	16.00	5.25	1.26	-	-	-
	Au	1,662	0.0001	1.0060	0.0326	0.0533	-	-	-
	Aucap	1,662	0.0001	0.6860	0.0322	0.0483	3	0.2	1.0
	Ag	1,641	0.0022	41.7560	0.4205	1.4843	-	-	-
	Agcap	1,641	0.0022	9.6000	0.3848	0.6993	3	0.2	8.5
Dyke Adit Vein	Length	354	4.00	5.00	4.99	0.11	-	-	-
	Au	352	0.0010	1.0260	0.0577	0.1046	-	-	-
	Aucap	352	0.0010	0.6860	0.0564	0.0940	2	0.6	2.4
	Ag	354	0.0030	9.6070	0.8453	1.2274	-	-	-
	Agcap	354	0.0030	9.6000	0.8453	1.2272	1	0.3	0.0
East Hill Vein	Length	282	5.00	5.00	5.00	-	-	-	-
	Au	282	0.0010	0.3340	0.0262	0.0308	-	-	-
	Aucap	282	0.0010	0.3340	0.0262	0.0308	0	0.0	0.0
	Ag	282	0.0100	6.3400	0.4469	0.6268	-	-	-
	Agcap	282	0.0100	6.3400	0.4469	0.6268	0	0.0	0.0

Zone	Field	N Samples	Minimum	Maximum	Mean	Standard Deviation	No. of Samples Capped	% of Dataset Capped	% Change of Mean After Capping
Footwall Zone	Length	5,088	1.00	10.70	5.47	1.64	-	-	-
	Au	5,028	0.0003	5.3890	0.0138	0.0826	-	-	-
	Aucap	5,028	0.0003	0.6860	0.0126	0.0320	6	0.1	8.5
	Ag	4,967	0.0005	27.5500	0.2193	0.6605	-	-	-
	Agcap	4,967	0.0005	9.6000	0.2127	0.4737	2	0.0	3.0
Hanging-Wall Zone	Length	4,454	1.00	10.00	5.68	1.79	-	-	-
	Au	4,211	0.0001	2.3560	0.0114	0.0385	-	-	-
	Aucap	4,211	0.0001	0.6860	0.0111	0.0209	1	0.0	3.0
	Ag	4,212	0.0015	20.1620	0.1830	0.4520	-	-	-
	Agcap	4,212	0.0015	9.6000	0.1802	0.3566	2	0.0	1.5
Main Zone	Length	4,690	1.00	13.50	5.31	1.29	-	-	-
	Au	4,493	0.0001	0.6860	0.0074	0.0203	-	-	-
	Aucap	4,493	0.0001	0.6860	0.0074	0.0203	0	0.0	0.0
	Ag	4,515	0.0015	9.0400	0.1330	0.3356	-	-	-
	Agcap	4,515	0.0015	9.0400	0.1330	0.3356	0	0.0	0.0
Dyke Adit	Length	2,454	1.00	15.00	5.03	0.74	-	-	-
	Au	1,787	0.0005	0.1690	0.0056	0.0103	-	-	-
	Aucap	1,787	0.0005	0.1690	0.0056	0.0103	0	0.0	0.0
	Ag	2,310	0.0015	4.6300	0.1097	0.2489	-	-	-
	Agcap	2,310	0.0015	4.6300	0.1097	0.2489	0	0.0	0.0
East Hill	Length	1,511	5.00	5.00	5.00	-	-	-	-
	Au	1,491	0.0005	0.2175	0.0053	0.0118	-	-	-
	Aucap	1,491	0.0005	0.2175	0.0053	0.0118	0	0.0	0.0
	Ag	1,459	0.0015	2.3250	0.0700	0.1387	-	-	-
	Agcap	1,459	0.0015	2.3250	0.0700	0.1387	0	0.0	0.0

14.5.3 COMPOSTING

Compositing of all assay data within the wireframes was completed at 5 ft intervals. The downhole intervals honoured the interpretation of the geological solids. The backstitching process was used in the compositing routine to ensure all captured sample material was included. The backstitching routine adjusts the composite lengths for each individual borehole in order to compensate for the last sample interval.

The 5 ft composites were selected as the optimal composite length to use in the estimation based on the large amount of RC drilling and in order to maintain the complex nature of the high grade vein system. Table 14.6 summarizes the statistics for the boreholes after compositing.

Table 14.6 - Drillhole Compositing Statistics

Zone	Field	N Samples	Minimum	Maximum	Mean	Standard Deviation
Footwall Vein	Length	1,716	4.33	5.50	5.00	0.06
	Au	1,711	0.0005	0.8150	0.0341	0.0483
	Aucap	1,711	0.0005	0.6860	0.0341	0.0471
	Ag	1,710	0.0015	20.2410	0.5339	1.1429
	Agcap	1,710	0.0015	9.6000	0.5179	0.9439
Hanging Wall Vein	Length	3,439	4.50	5.50	5.00	0.04
	Au	3389	0.0001	1.9430	0.0409	0.0779
	Aucap	3,389	0.0001	0.6860	0.0398	0.0606
	Ag	3,390	0.0025	23.1105	0.5328	1.0576
	Agcap	3,390	0.0025	9.6000	0.5251	0.9537
Main Zone Vein	Length	1,756	4.00	5.22	5.00	0.05
	Au	1,747	0.0001	1.0060	0.0326	0.0532
	Aucap	1,747	0.0001	0.6860	0.0322	0.0482
	Ag	1,727	0.0022	41.7560	0.4205	1.4802
	Agcap	1,727	0.0022	9.6000	0.3848	0.6914
Dyke Adit Vein	Length	353	4.89	5.25	5.00	0.04
	Au	351	0.0010	1.0260	0.0577	0.1046
	Aucap	351	0.0010	0.6860	0.0564	0.0940
	Ag	353	0.0030	9.6070	0.8453	1.2270
	Agcap	353	0.0030	9.6000	0.8453	1.2269
East Hill Vein	Length	282	5.00	5.00	5.00	-
	Au	282	0.0010	0.3340	0.0262	0.0308
	Aucap	282	0.0010	0.3340	0.0262	0.0308
	Ag	282	0.0100	6.3400	0.4469	0.6268
	Agcap	282	0.0100	6.3400	0.4469	0.6268

Zone	Field	N Samples	Minimum	Maximum	Mean	Standard Deviation
Footwall Vein	Length	5,566	2.50	5.33	5.00	0.04
	Au	5,516	0.0003	5.3890	0.0138	0.0825
	Aucap	5,516	0.0003	0.6860	0.0126	0.0319
	Ag	5,473	0.0005	27.5500	0.2192	0.6581
	Agcap	5,473	0.0005	9.6000	0.2127	0.4711
Hanging Wall Vein	Length	5,060	3.00	6.00	5.00	0.05
	Au	4,817	0.0001	2.3560	0.0114	0.0385
	Aucap	4,817	0.0001	0.6860	0.0111	0.0208
	Ag	4,818	0.0015	20.1620	0.1829	0.4512
	Agcap	4,818	0.0015	9.6000	0.1802	0.3556
Main Zone Vein	Length	4,985	4.00	5.67	5.00	0.04
	Au	4,798	0.0001	0.6860	0.0074	0.0203
	Aucap	4,798	0.0001	0.6860	0.0074	0.0203
	Ag	4,821	0.0015	9.0400	0.1330	0.3337
	Agcap	4,821	0.0015	9.0400	0.1330	0.3337
Dyke Adit	Length	2,467	4.25	5.19	5.00	0.02
	Au	1,795	0.0005	0.1690	0.0056	0.0102
	Aucap	1,795	0.0005	0.1690	0.0056	0.0102
	Ag	2,316	0.0015	4.6300	0.1098	0.2484
	Agcap	2,316	0.0015	4.6300	0.1098	0.2484
East Hill	Length	1,511	5.00	5.00	5.00	-
	Au	1,491	0.0005	0.2175	0.0053	0.0118
	Aucap	1,491	0.0005	0.2175	0.0053	0.0118
	Ag	1,459	0.0015	2.3250	0.0700	0.1387
	Agcap	1,459	0.0015	2.3250	0.0700	0.1387

14.6 SPATIAL ANALYSIS

Variography, using Datamine™ software, was completed for each element globally for all the composited data. Downhole variograms were used to determine nugget effect and then correlograms were modelled with two structures to determine spatial continuity in the zones.

Table 14.7 summarizes results of the variography, while Figure 14.22 to Figure 14.41 depicts the correlograms for each of the elements being estimated in each of the mineral domains.

Table 14.7 - Variogram Parameters

VDesc	V Refnum	V Angle1	V Angle2	V Angle3	V Axis1	V Axis2	V Axis3	Nugget	St1	St1 Par1	St1 Par2	St1 Par3	St1 Par4	St2	St2 Par1	St2 Par2	St2 Par3	St2 Par4
Au_Dyke Zone	1	30	0	60	3	2	1	0.3	1	32	51	58	0.038	1	75	174	101	0.662
Ag_Dyke Zone	2	-60	0	120	3	2	1	0.05	1	109	119	88	0.225	1	176	193	110	0.725
Au_Dyke Vein	3	-30	0	120	3	2	1	0.32	1	100	86	99	0.013	1	266	176	200	0.667
Ag-Dyke Vein	4	30	0	60	3	2	1	0.4	1	50	46	57	0.15	1	100	85	153	0.45
Au_East Hill Zone	5	0	0	60	3	2	1	0.05	1	50	46	100	0.085	1	210	250	235	0.865
Ag_East Hill Zone	6	60	0	60	3	2	1	0.1	1	67	50	113	0.111	1	215	131	178	0.789
Au_East Hill Vein	7	-60	0	120	3	2	1	0.4	1	47	47	99	0.104	1	101	148	301	0.496
Ag_East Hill Vein	8	120	-60	0	3	2	1	0.35	1	100	109	0	0.019	1	297	456	0	0.631
Au_BCFW Zone	9	-30	0	60	3	2	1	0.03	1	104	52	71	0.279	1	165	81	122	0.691
Ag_BCFW Zone	10	-30	0	60	3	2	1	0.03	1	177	68	47	0.103	1	408	150	119	0.867
Au_BCFWVein	11	-90	0	150	3	2	1	0.2	1	186	351	50	0.344	1	193	820	272	0.456
Ag_BCFW Vein	12	-60	0	60	3	2	1	0.1	1	60	73	146	0.063	1	100	150	446	0.837
Au_BCHW Zone	13	0	0	60	3	2	1	0.1	1	28	21	81	0.027	1	63	98	149	0.873
Ag_BCHW Zone	14	60	0	120	3	2	1	0.05	1	90	81	92	0.075	1	211	119	158	0.875
Au_BCHW Vein	15	-30	0	150	3	2	1	0.1	1	133	58	65	0.716	1	165	388	88	0.184
Ag_BCHW Vein	16	-90	0	60	3	2	1	0.15	1	90	21	157	0.329	1	124	170	250	0.521
Au_Main Zone	17	-90	0	120	3	2	1	0.05	1	67	164	24	0.084	1	217	254	92	0.866
Ag_Main Zone	18	30	0	60	3	2	1	0.03	1	23	114	68	0.52	1	110	236	98	0.45
Au_Main Vein	19	0	0	150	3	2	1	0.4	1	150	145	31	0.243	1	150	642	80	0.357
Ag_Main Vein	20	-60	0	60	3	2	1	0.1	1	156	159	80	0.192	1	400	298	120	0.708

Figure 14.22 - Bear Creek Hanging Wall Vein Gold Variogram

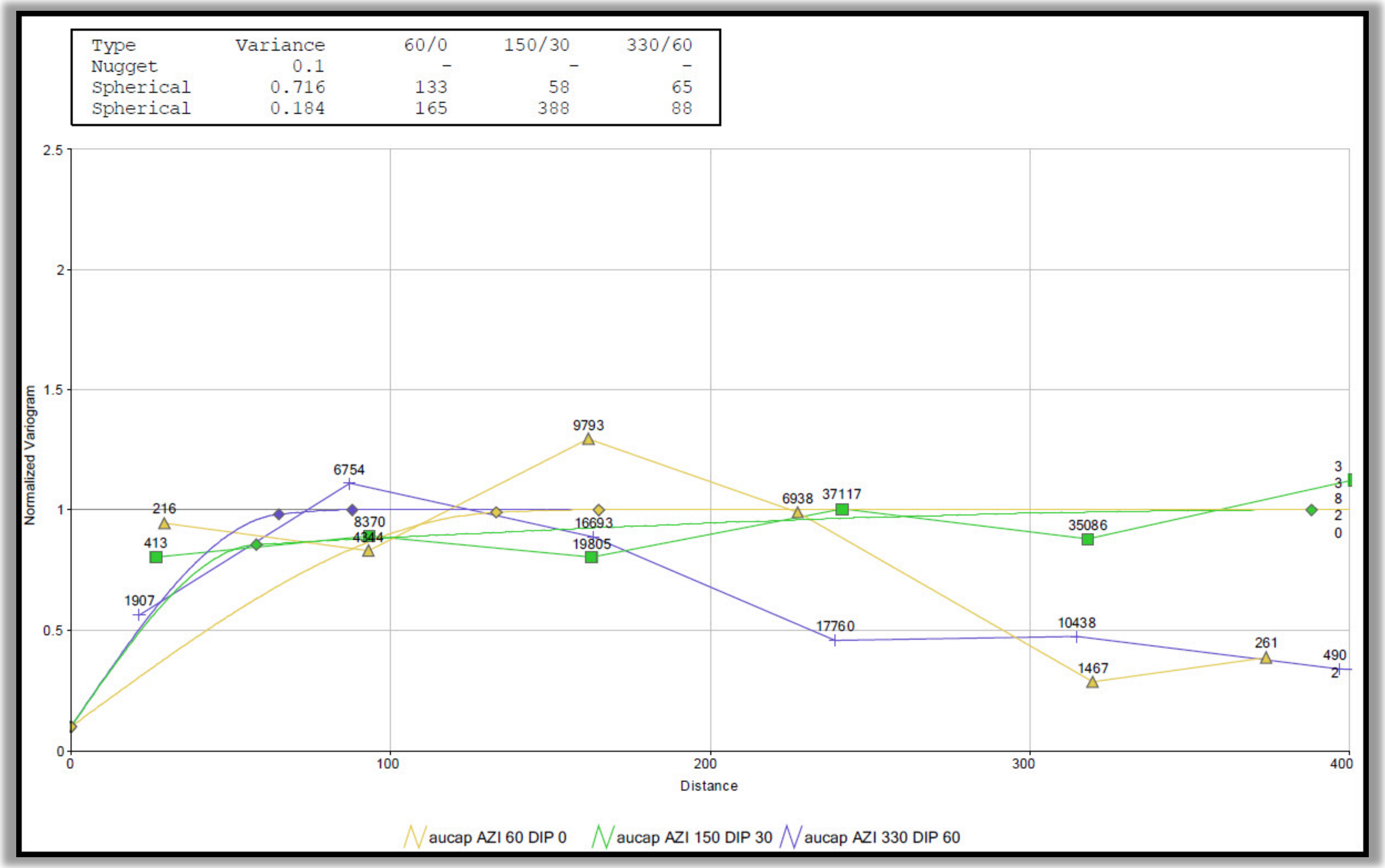


Figure 14.23 - Bear Creek Hanging Wall Vein Silver Variogram

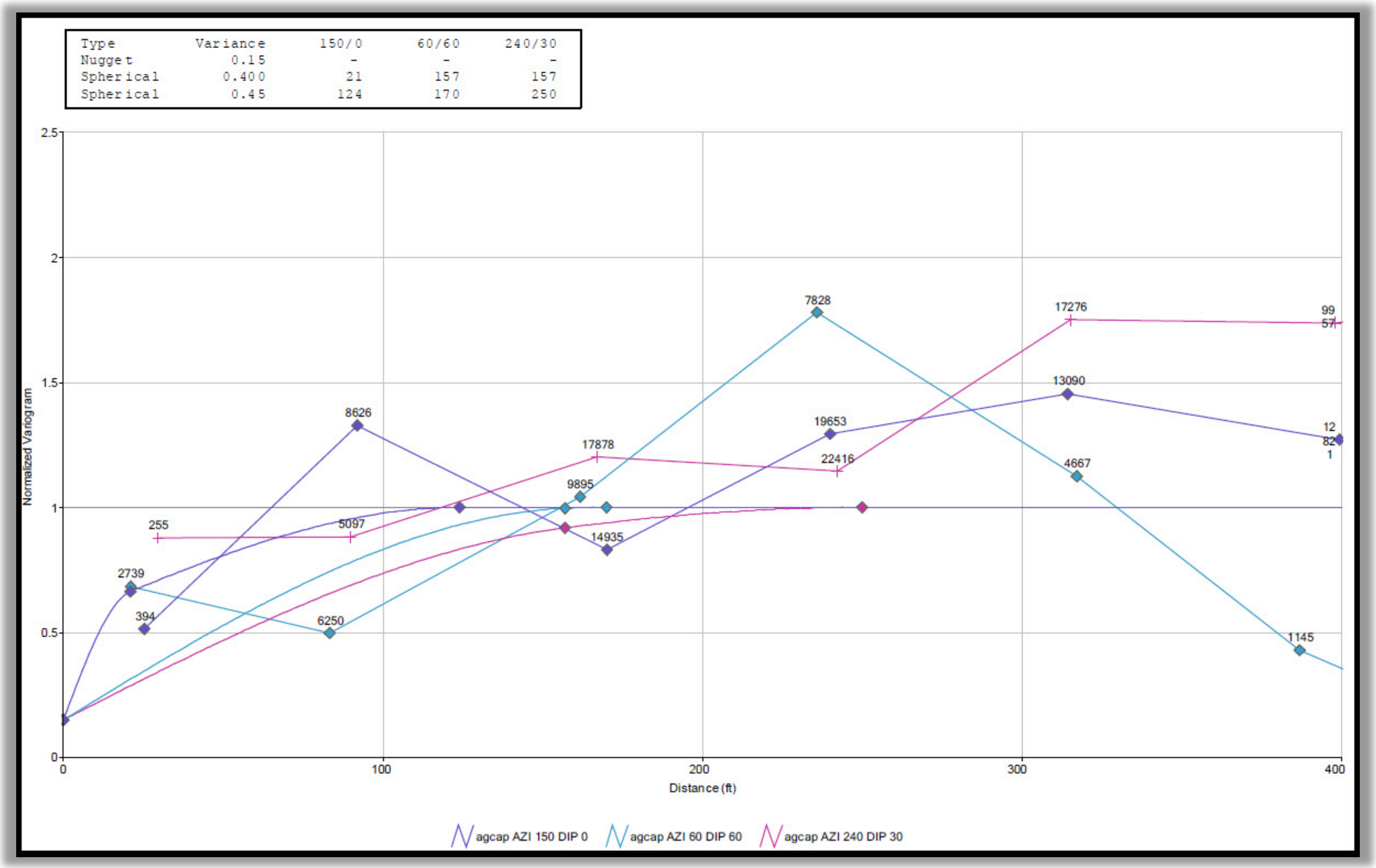


Figure 14.24 - Bear Creek Footwall Vein Gold Variogram

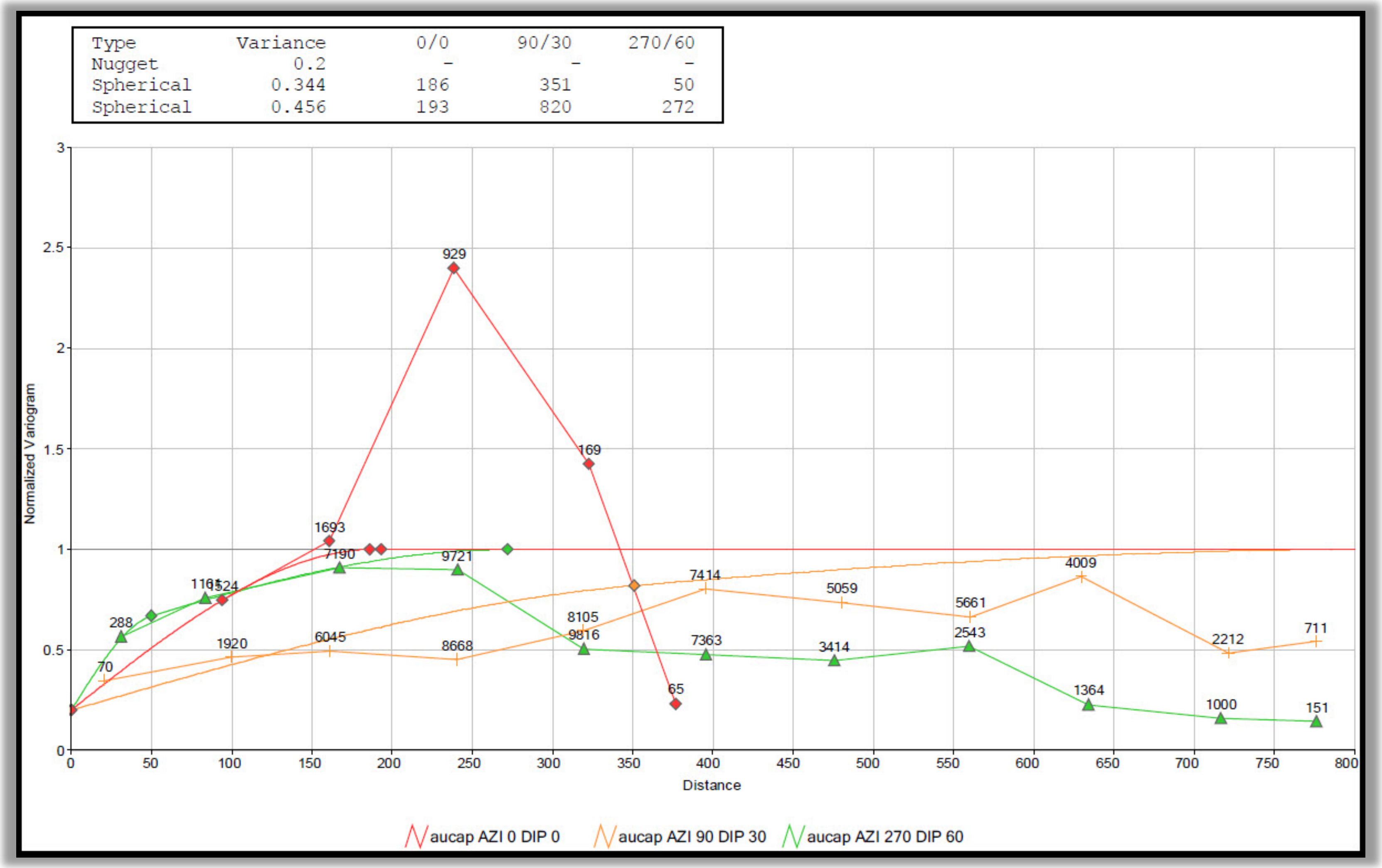


Figure 14.25 - Bear Creek Footwall Vein Silver Variogram

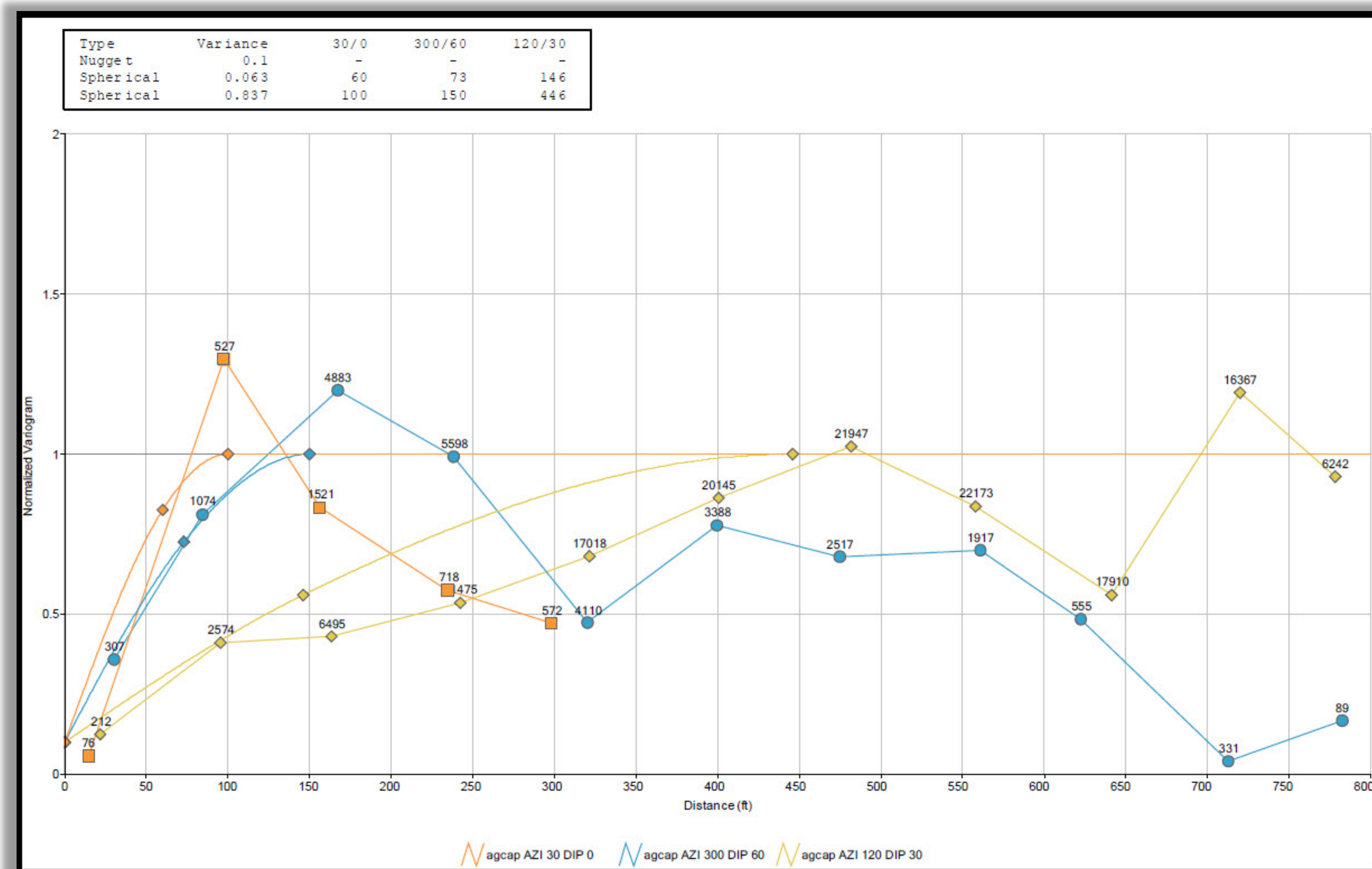


Figure 14.26 - Main Vein Gold Variogram

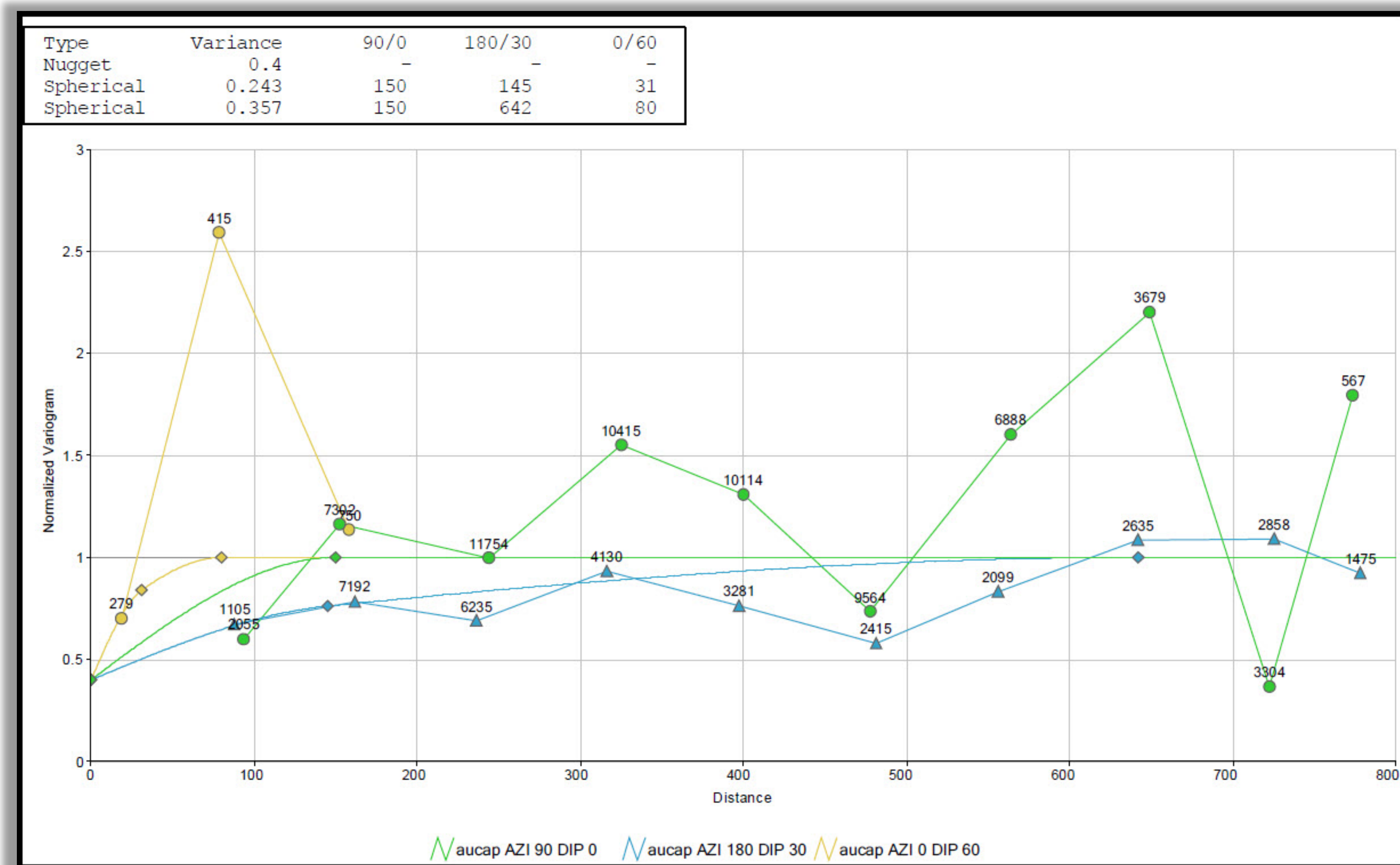


Figure 14.27 - Main Vein Silver Variogram

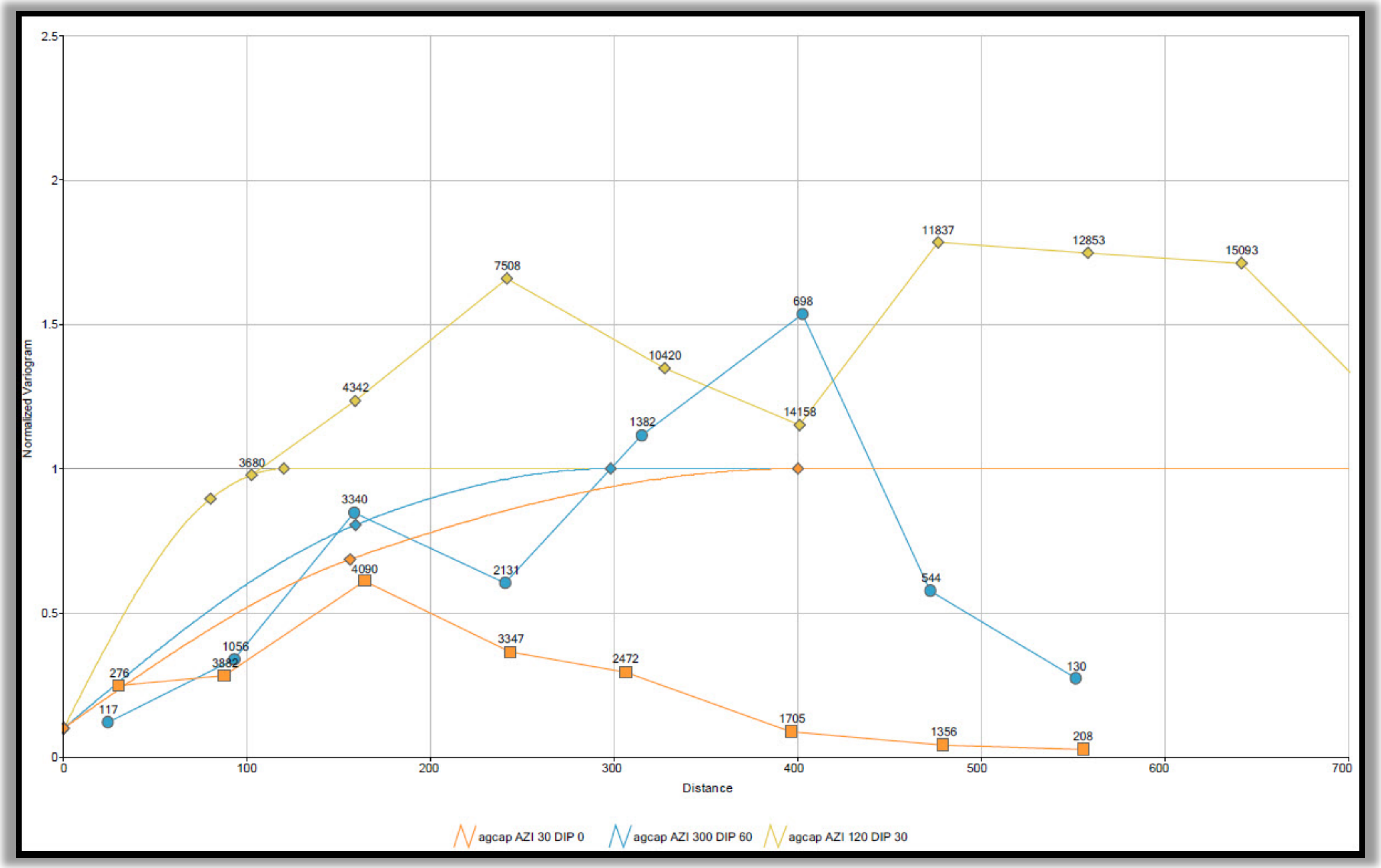


Figure 14.28 - Dyke Adit Vein Gold Variogram

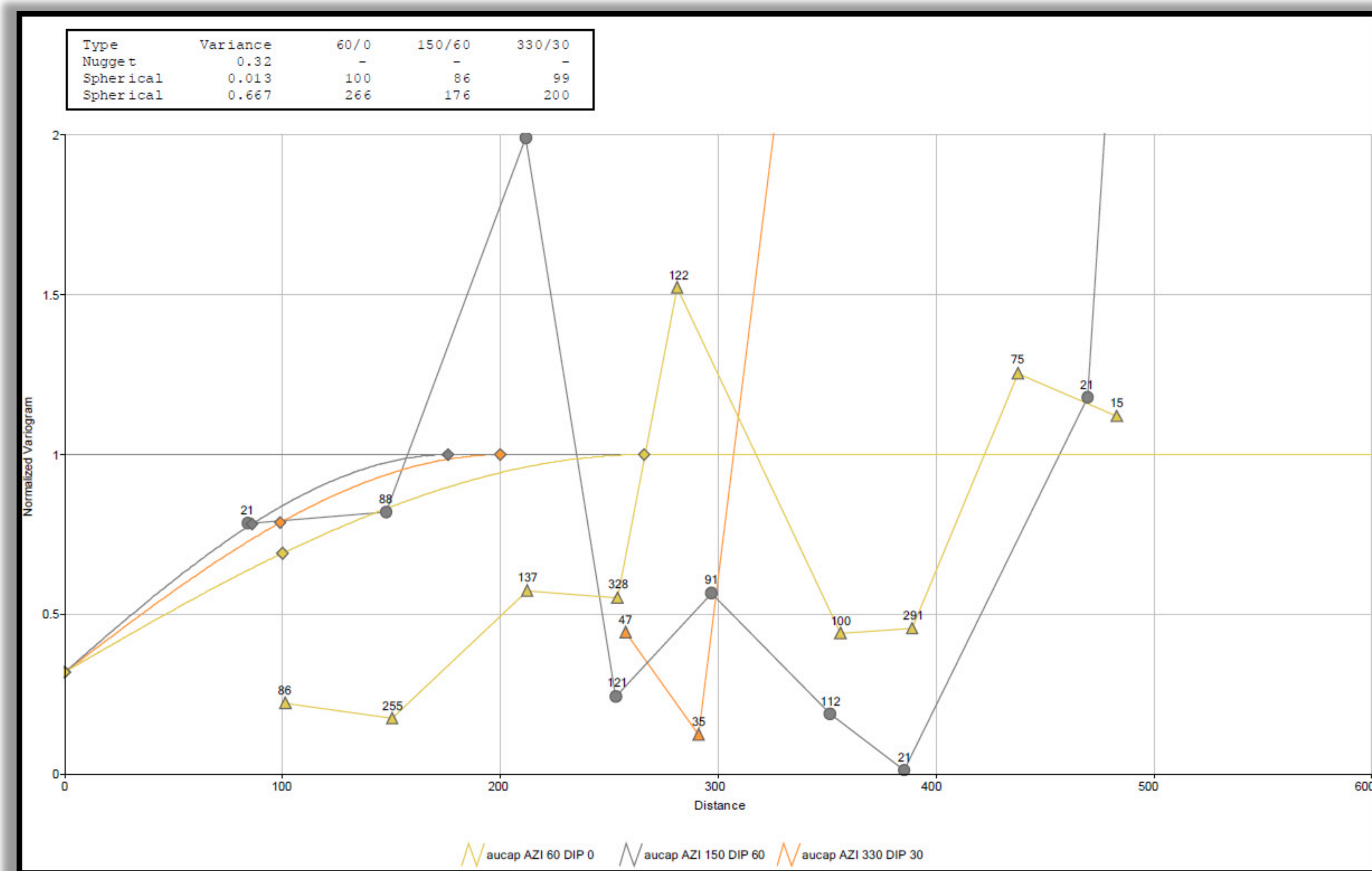


Figure 14.29 - Dyke Adit Vein Silver Variogram

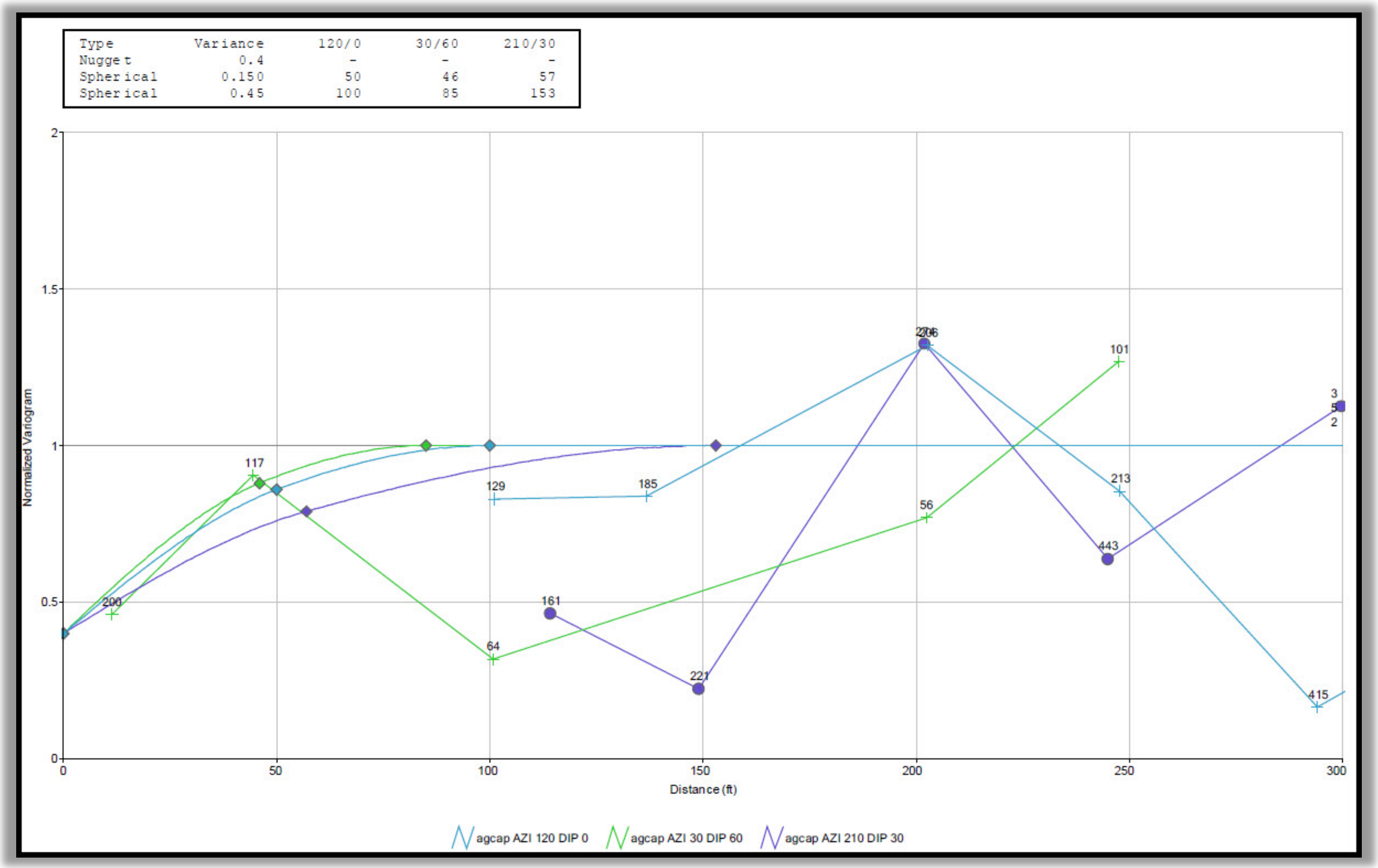


Figure 14.30 - East Hill Vein Gold Variogram

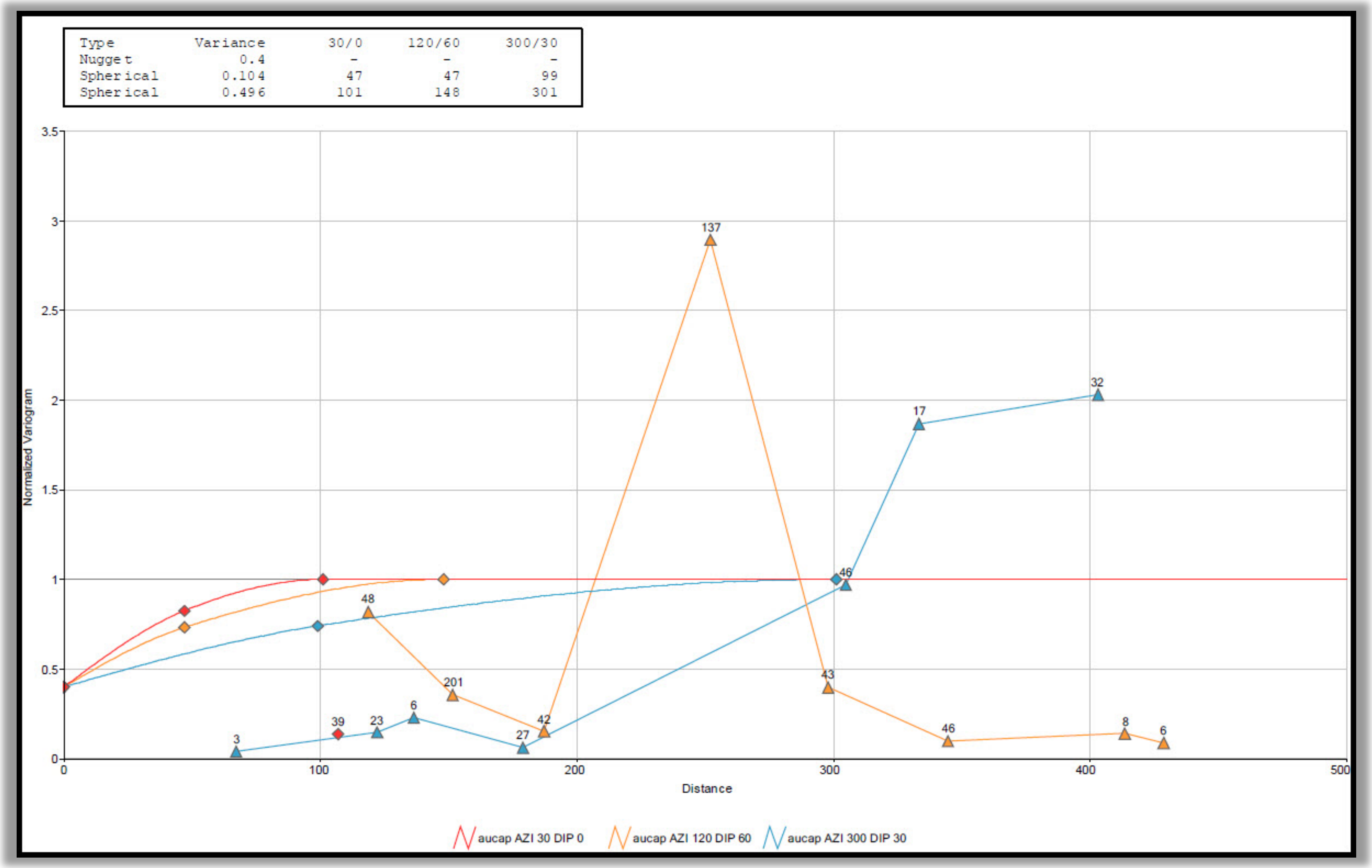


Figure 14.31 - East Hill Vein Silver Variogram

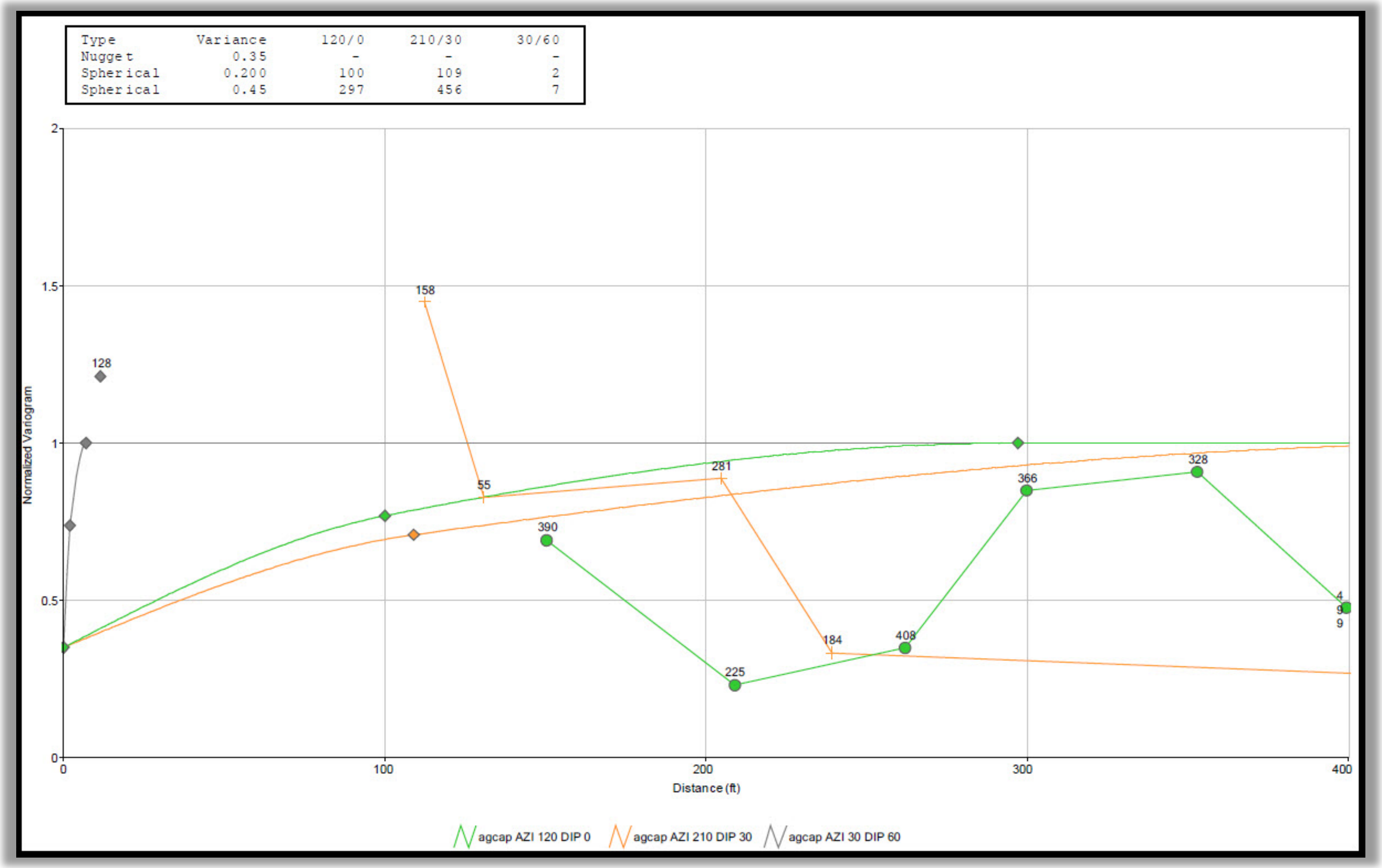


Figure 14.32 - Bear Creek Hanging Wall Zone Gold Variogram

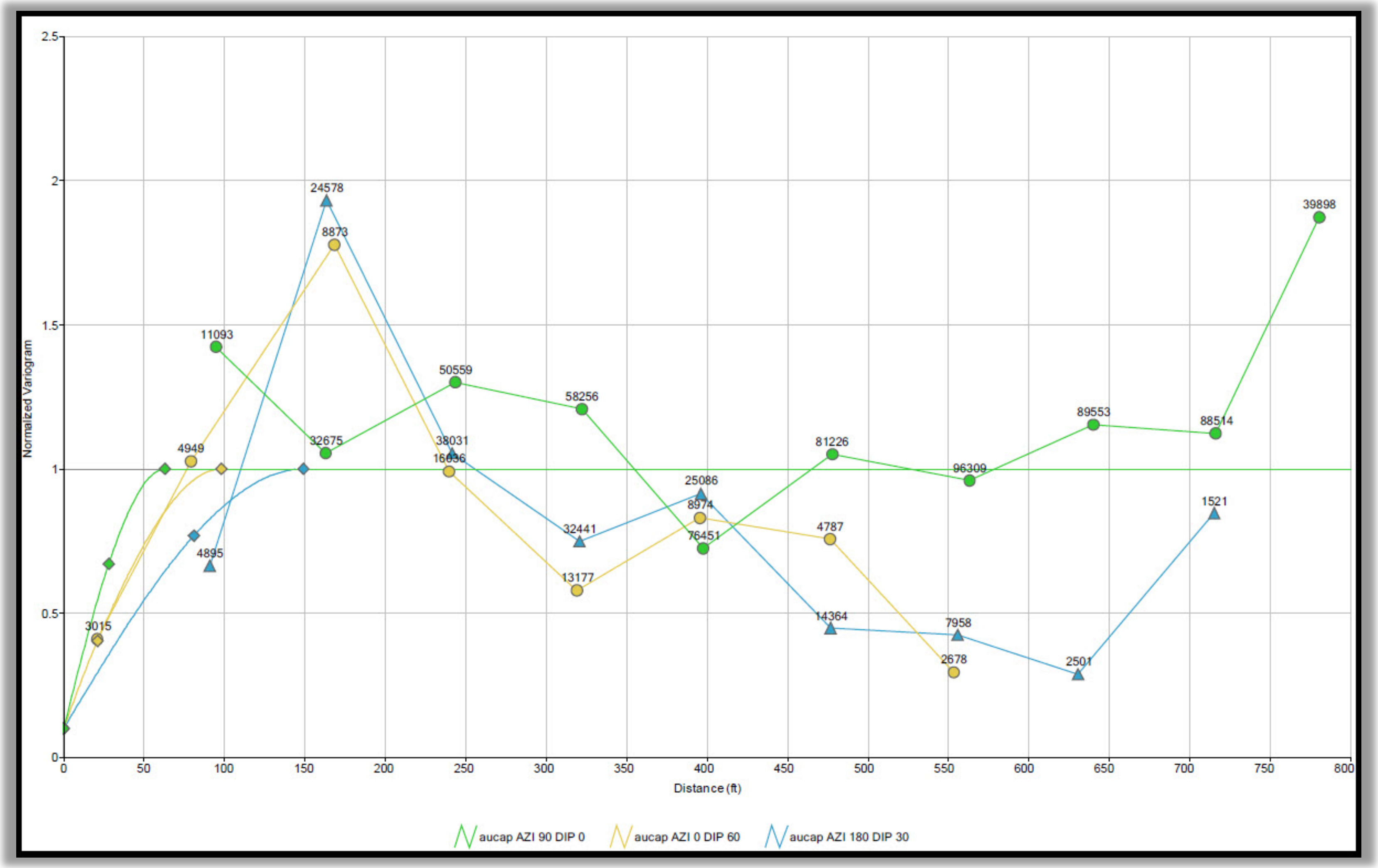


Figure 14.33 - Bear Creek Hanging Wall Zone Silver Variogram

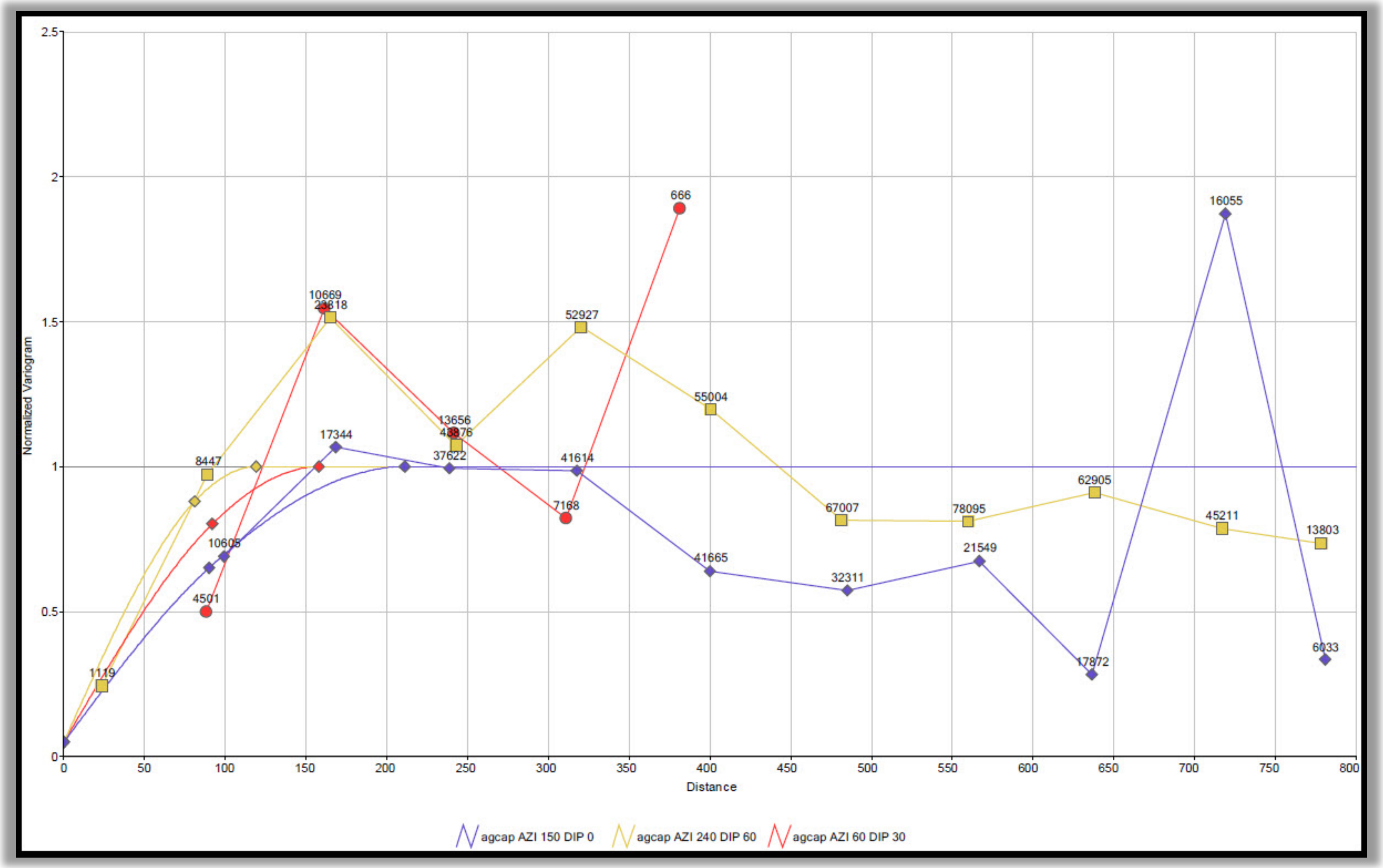


Figure 14.34 - Bear Creek Footwall Zone Gold Variogram

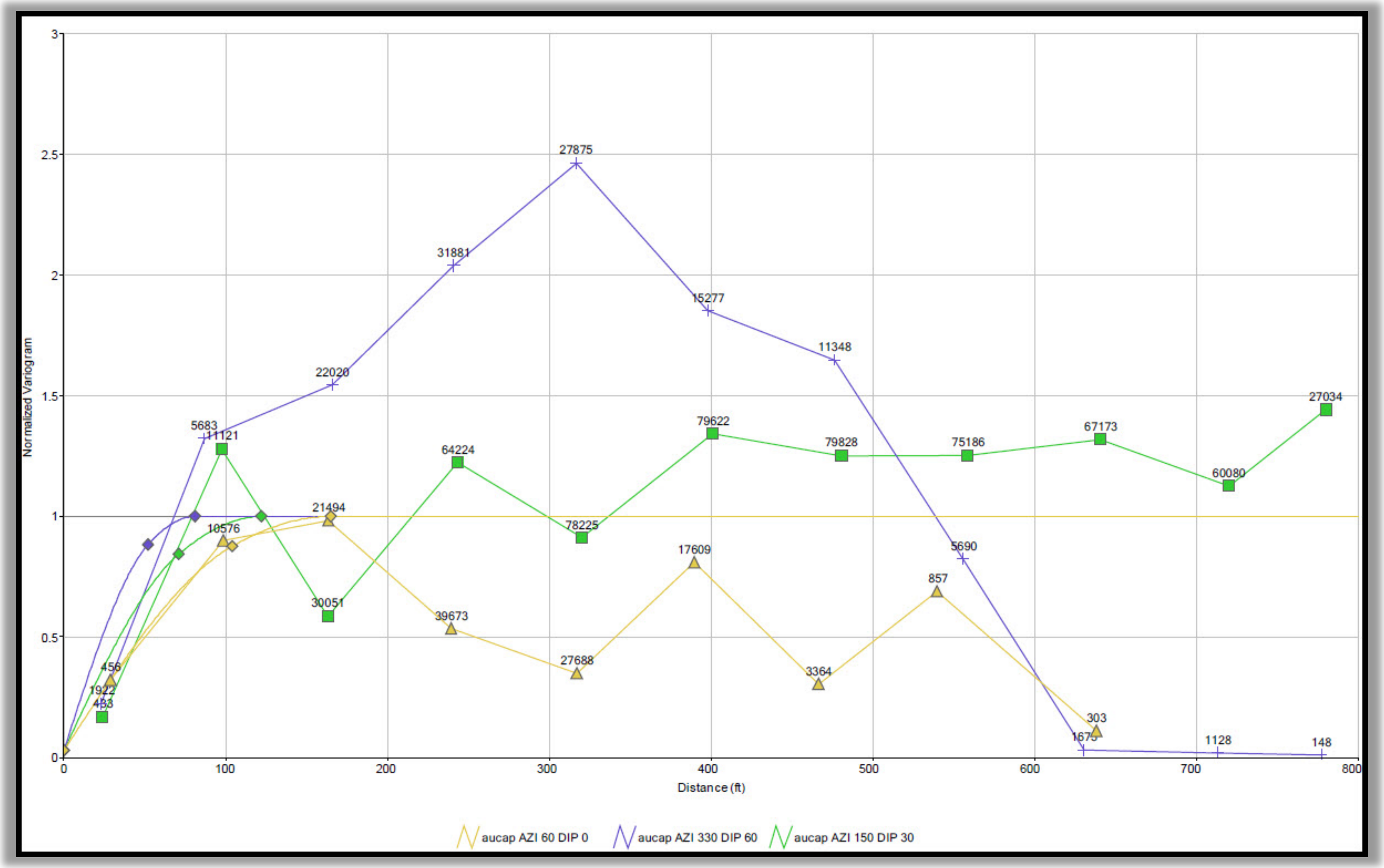


Figure 14.35 -Bear Creek Footwall Zone Silver Variogram

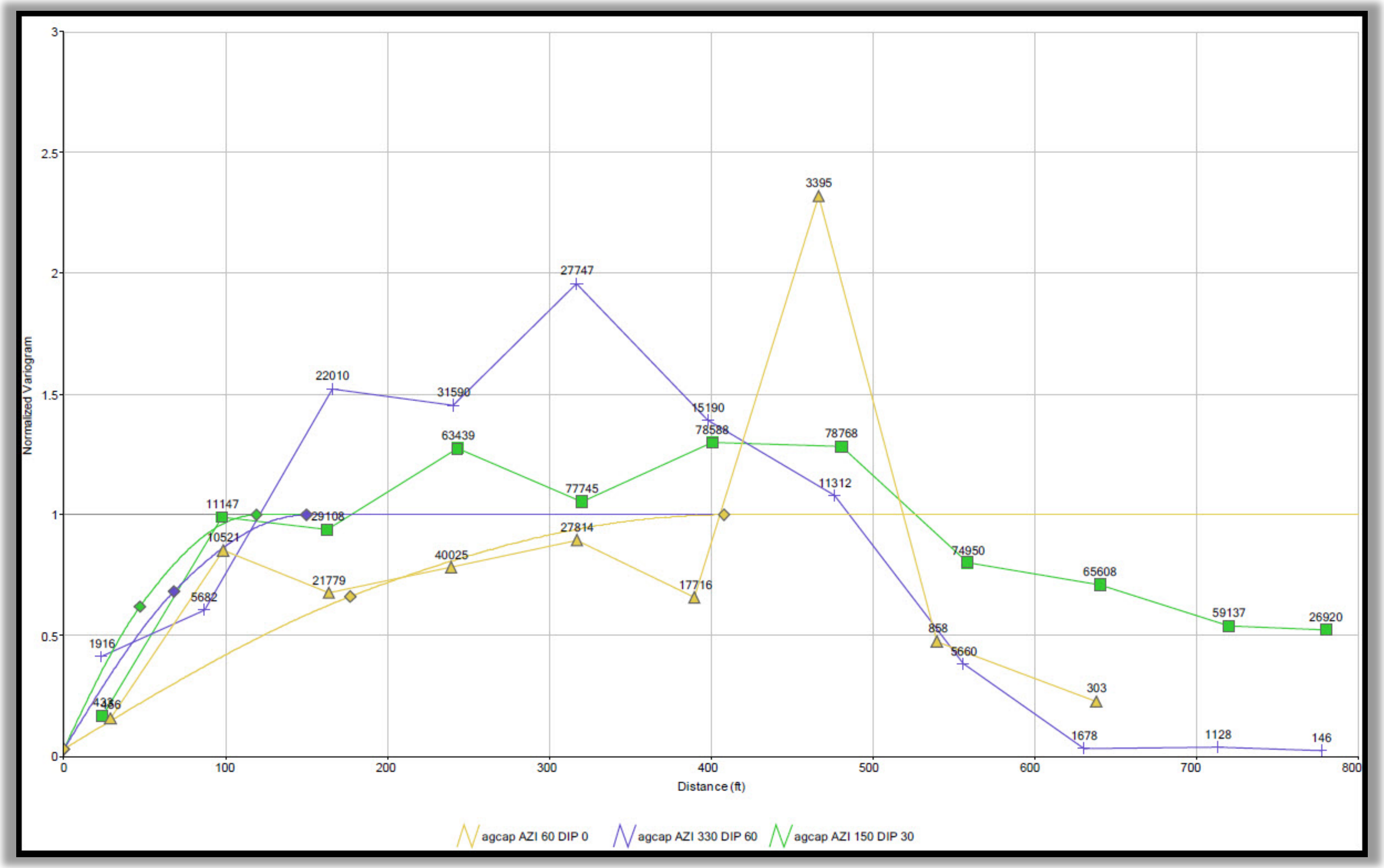


Figure 14.36 - Main Zone Gold Variogram

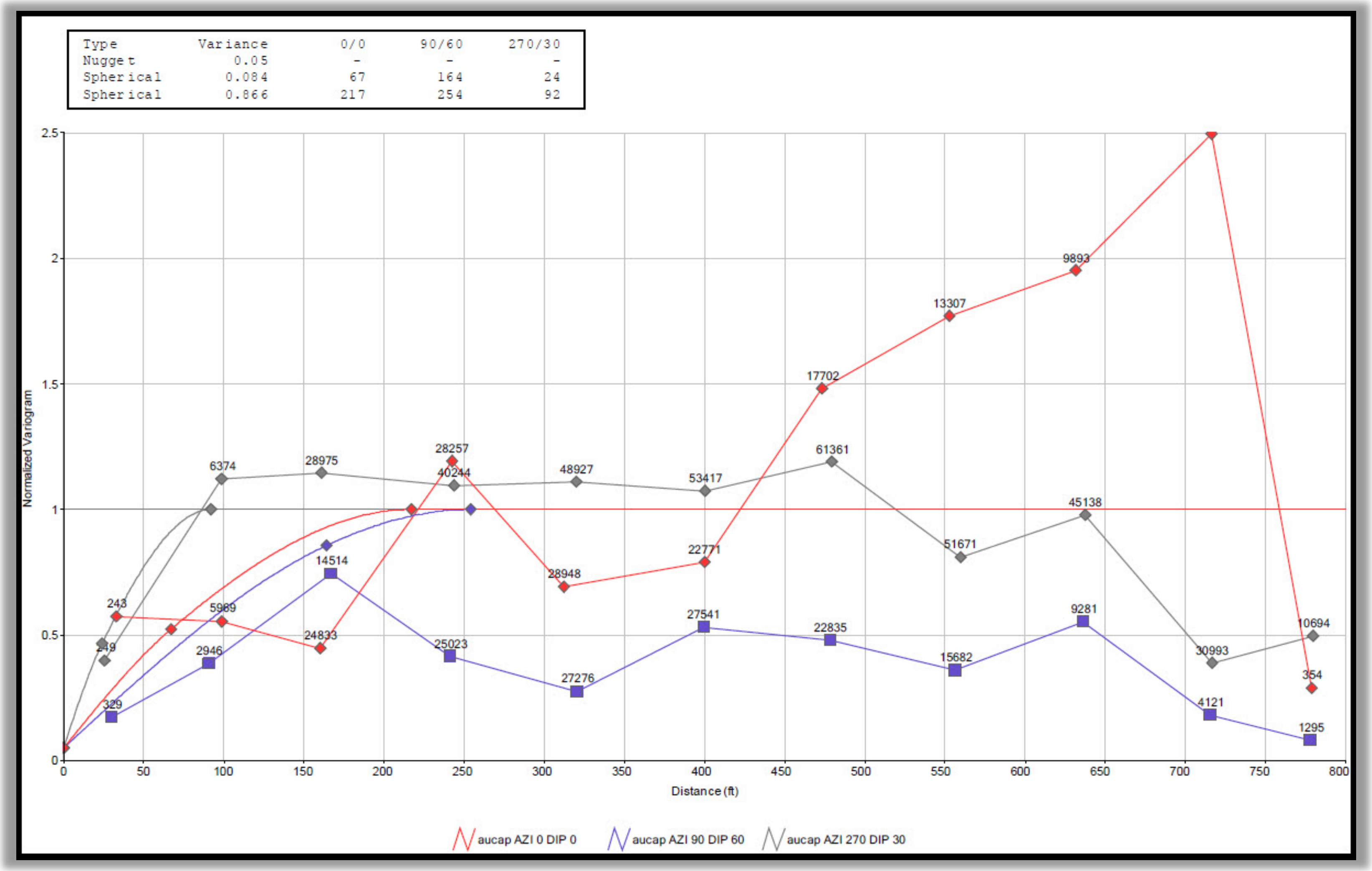


Figure 14.37 - Main Zone Silver Variogram

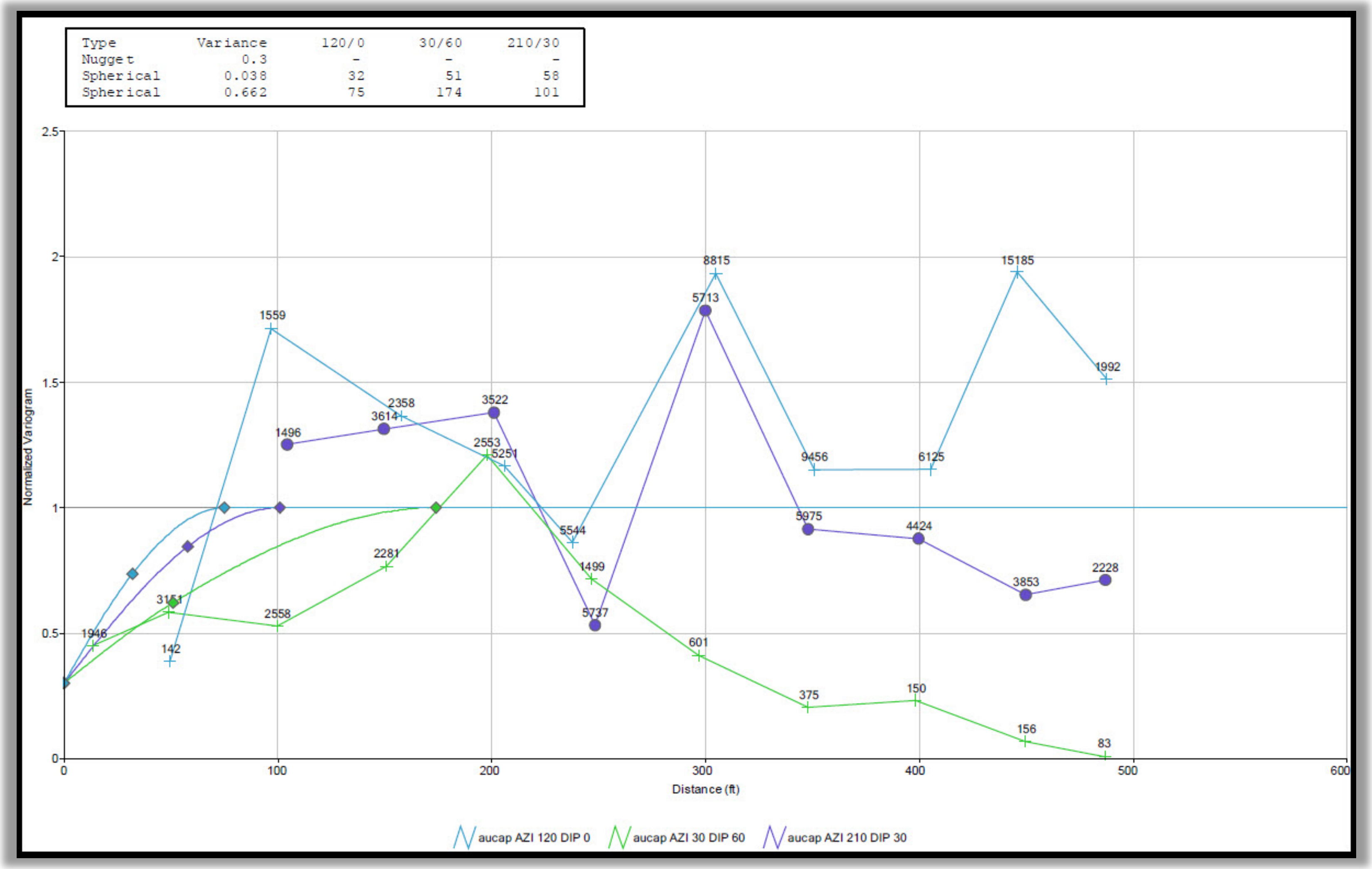


Figure 14.38 - Dyke Adit Zone Gold Variogram

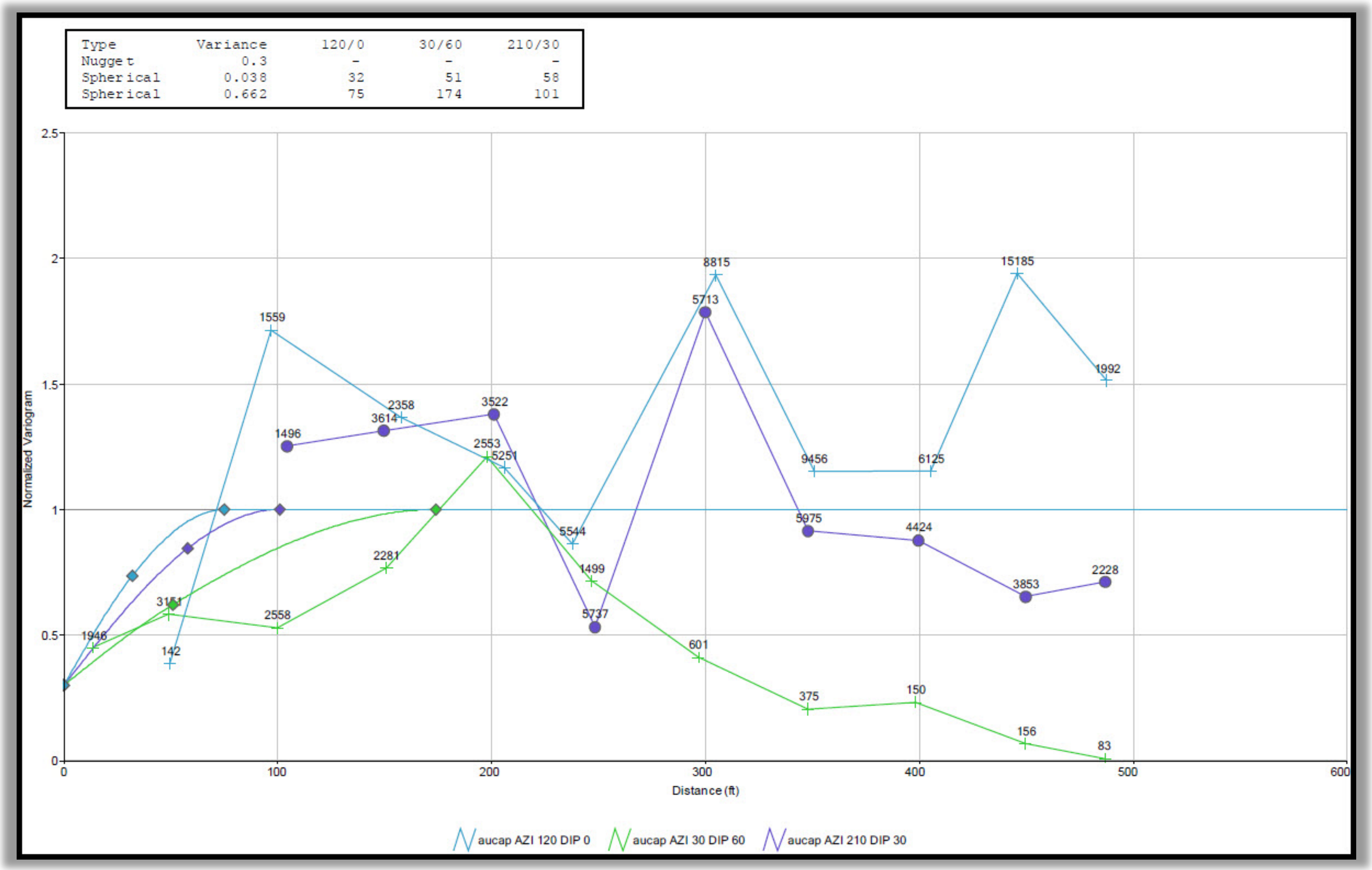


Figure 14.39 - Dyke Adit Zone Silver Variogram

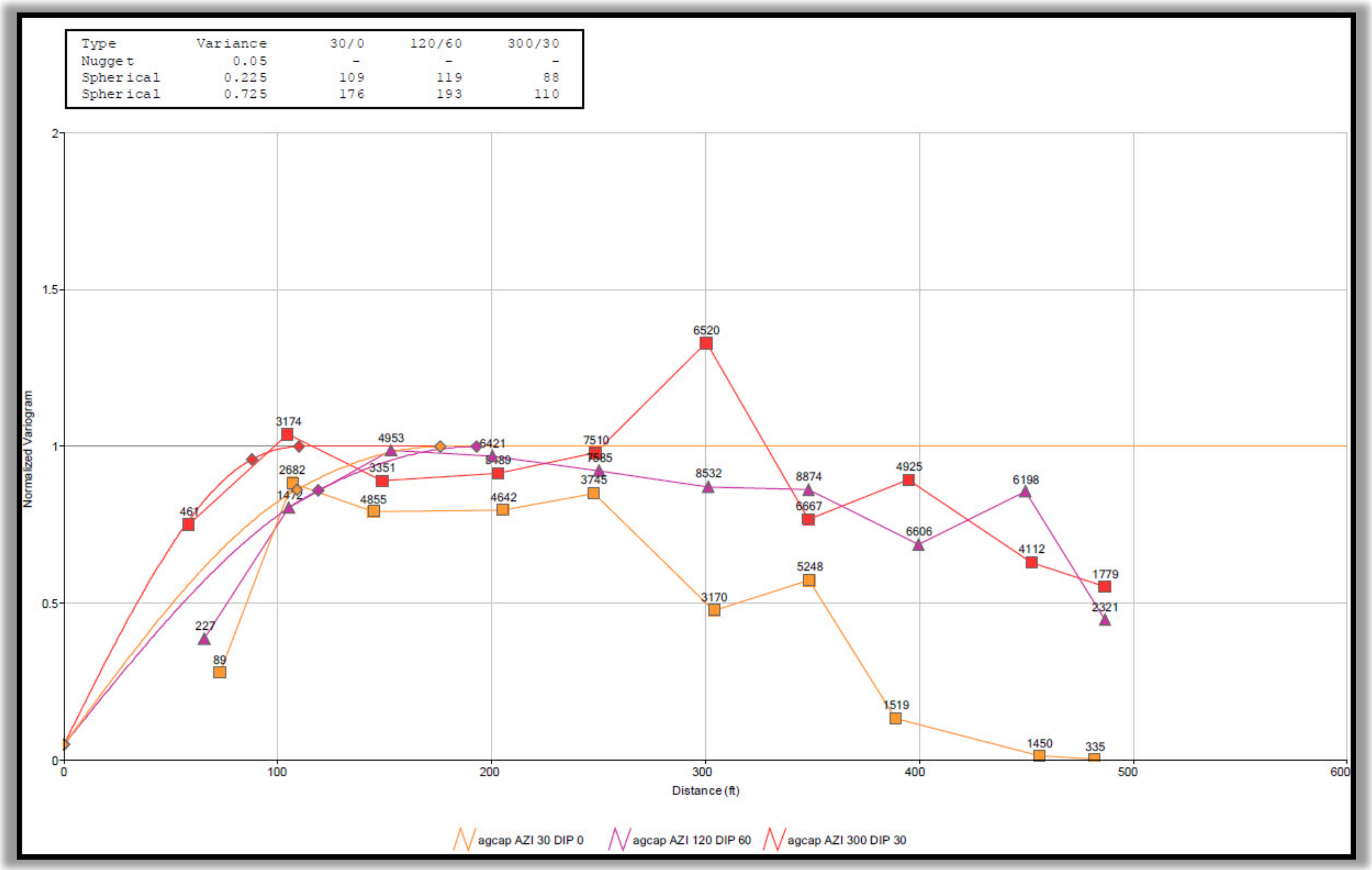


Figure 14.40 - East Hill Zone Gold Variogram

Type	Variance	90/0	0/60	180/30
Nugget	0.05	-	-	-
Spherical	0.085	50	46	100
Spherical	0.865	210	250	235

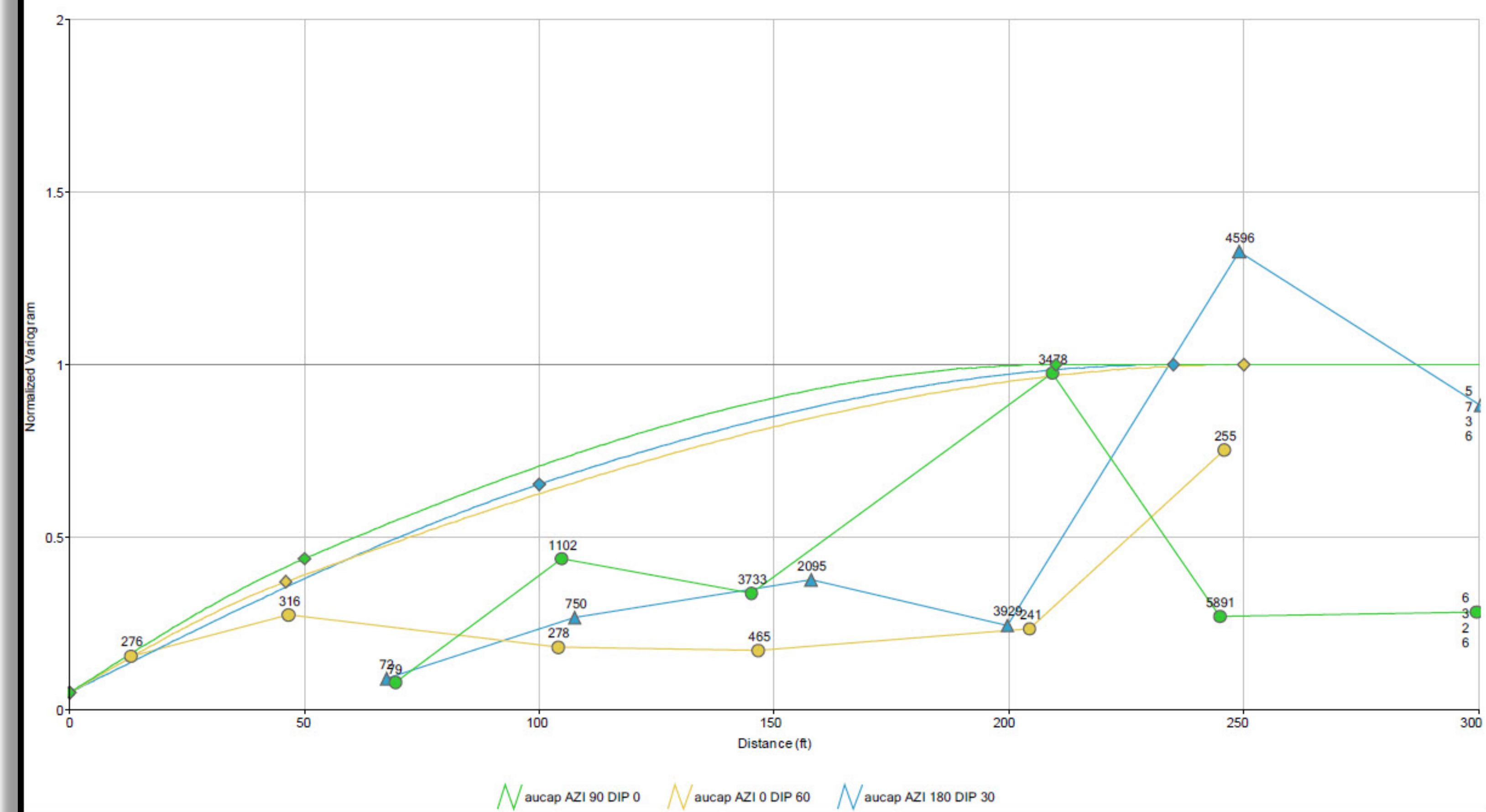
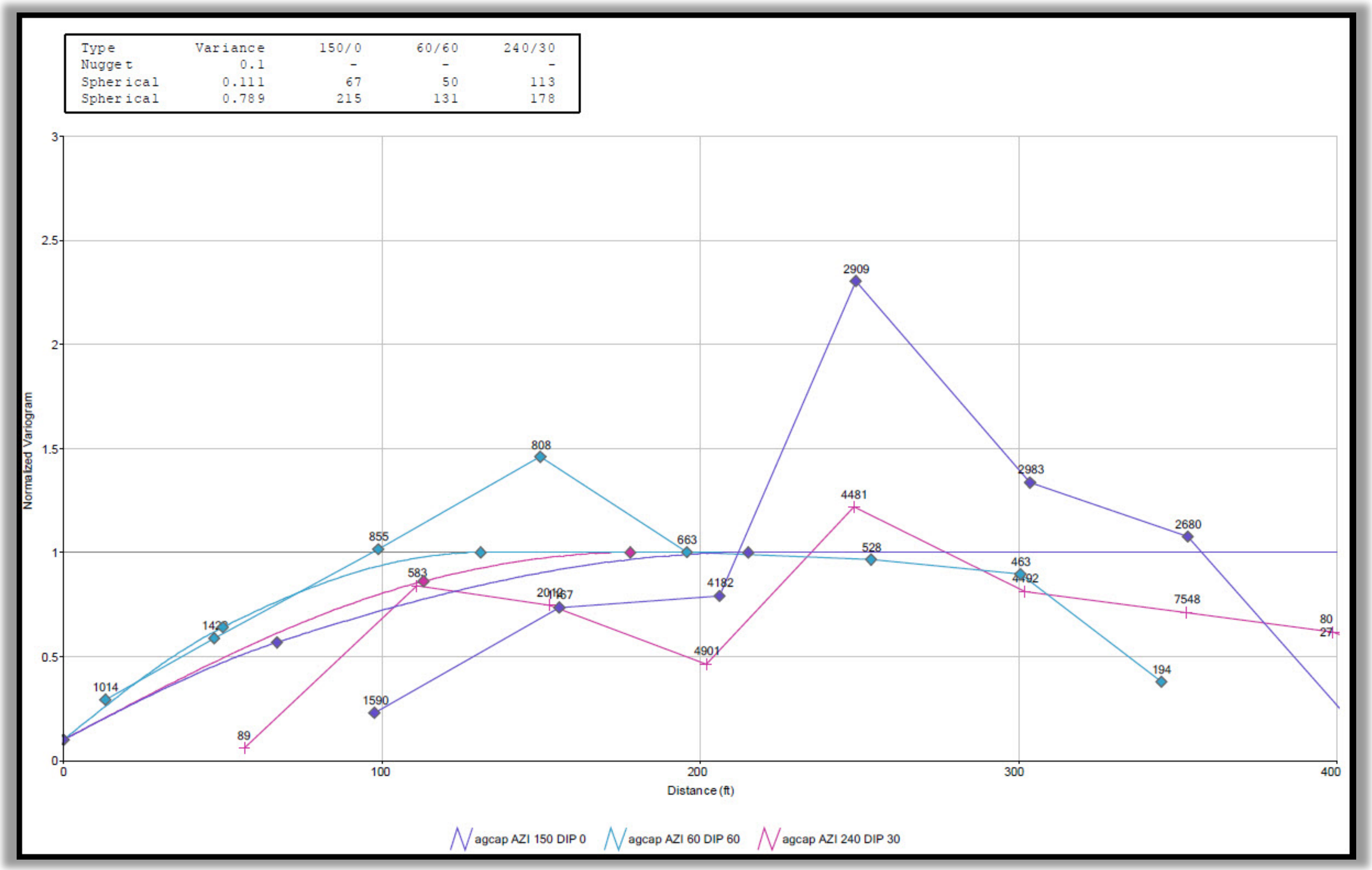


Figure 14.41 - East Hill Zone Silver Variogram



14.7 RESOURCE BLOCK MODEL

Individual block models were established in Datamine™ for the mineral wireframes using one parent model as the origin. The model was not rotated.

Drillhole spacing is variable with the majority of the surface drilling spaced at 25 m section and 25 to 100 m on sections. A block size of 30 ft by 30 ft by 30 ft was selected in order to accommodate the nature of the mineralization and be amenable for open mining potential.

Sub-celling of the block model on a 7.5 by 7.5 by 7.5 pattern in the XZ plane allows the parent block to be split in each direction to more accurately fill the volume of the wireframes, thus more accurately estimate the tonnes in the resource.

At the end of the modelling process, the high grade model was overlain on the low grade model.

Table 14.8 summarizes details of the parent block model, while Table 14.9 compares the volumes of the wireframes to the volume of the block models as a validation that prior to the estimation, the entire wireframe volumes are filled with blocks.

Table 14.8 - Parent Model Summary

Origin			Cell Size			Number of Cells		
X Origin	Y Origin	Z Origin	XINC	YINC	ZINC	NX	NY	NZ
302000	1711100	4200	30	30	30	220	150	70

Table 14.9 - Wireframe versus Model Volumes

Zone	Wireframe Volume (ft^3)	Model Volume (ft^3)	Difference (%)
Bear Creek Hanging-Wall Vein	104,010,965.5	104,014,286.9	0.00
Bear Creek Footwall Vein	64,305,299.2	64,306,852.5	0.00
Main Vein	79,122,762.1	79,113,984.4	0.01
East Hill Vein	23,591,416.9	23,591,509.2	0.00
Dyke Adit Vein	25,981,203.5	25,987,671.7	0.02
Bear Creek Hanging-Wall Zone	385,778,386.4	366,367,073.0	5.03
Bear Creek Footwall Zone	335,113,638.8	329,787,390.3	1.59
Main Zone	492,976,463.2	492,826,336.1	0.03
East Hill Zone	218,257,322.5	218,225,001.8	0.01
Dyke Adit Zone	685,016,156.6	685,012,452.6	0.00

14.7.1 DYNAMIC ANISOTROPY

Due to the erratic nature of the wireframes compared to the likely geology geometry and the distribution of the mineralization within the zones, a single search ellipse would not be practical and would result in the smearing of grades.

Dynamic anisotropy is an option in Datamine™ Studio 3 that allows the anisotropy rotation angles that define search volumes and variogram models to be defined individually for each cell in the model, thus allowing the search volume to be precisely oriented to follow the trend of the mineralization. Figure 14.42 is an example on how the orientation of the search ellipse will vary across the mineralized zone.

Figure 14.42 - Dynamic Anisotropy Direction

14.7.2 ESTIMATION AND SEARCH PARAMETERS

The interpolations of the zones were completed using the estimation methods: NN, ID^2 and OK. The estimations were designed for three passes. In each pass a minimum and maximum number of samples were required as well as a maximum number of samples from a borehole in order to satisfy the estimation criteria.

Estimation runs were completed in two steps. Step one involved the estimation was completed on the high grade vein domain. The second step was the estimation on the lower-grade domain. The results of the high-grade domain model were overlain on the results of the low-grade domain. This allowed the higher-grade domain to be preserved and eliminate the potential to grade smearing across strike.

Tables 14.10 and 14.11 summarize the interpolation criteria for the various mineral domains.

Table 14.10 - Estimation Parameters

Zone	Description	Est Ref #	Value_In	Value_Out	Search Ref #	Numsam_F	Svol_F	Imethod	Vrefnum
Dyke Adit Zone	auok	1	aucap	auok	17	NUMSAM	SVOL	3	1
	auid	2	aucap	auid	17			2	1
	aunn	3	aucap	aunn	17			1	1
	F	4	aucap	F	17			101	1
	LG	5	aucap	LG	17			101	1
	agok	6	agcap	agok	18			3	2
	agid	7	agcap	agid	18			2	1
	agnn	8	agcap	agnn	18			1	1
	CONF	9	Confiden	C	17			2	1
Dyke Adit Vein	auok	1	aucap	auok	7	NUMSAM	SVOL	3	3
	auid	2	aucap	auid	7			2	1
	aunn	3	aucap	aunn	7			1	1
	F	4	aucap	F	7			101	3
	LG	5	aucap	LG	7			101	3
	agok	6	agcap	agok	8			3	4
	agid	7	agcap	agid	8			2	1
	agnn	8	agcap	agnn	8			1	1
	CONF	9	Confiden	C	7			2	1
East Hill Zone	auok	1	aucap	auok	19	NUMSAM	SVOL	3	5
	auid	2	aucap	auid	19			2	1
	aunn	3	aucap	aunn	19			1	1
	F	4	aucap	F	19			101	5
	LG	5	aucap	LG	19			101	5
	agok	6	agcap	agok	20			3	6
	agid	7	agcap	agid	20			2	1
	agnn	8	agcap	agnn	20			1	1
	CONF	9	Confiden	C	19			2	1

Zone	Description	Est Ref #	Value_In	Value_Out	Search Ref #	Numsam_F	Svol_F	Imethod	Vrefnum
East Hill Vein	auok	1	aucap	auok	9	NUMSAM	SVOL	3	7
	auid	2	aucap	auid	9			2	1
	aunn	3	aucap	aunn	9			1	1
	F	4	aucap	F	9			101	7
	LG	5	aucap	LG	9			101	7
	agok	6	agcap	agok	10			3	8
	agid	7	agcap	agid	10			2	1
	agnn	8	agcap	agnn	10			1	1
	CONF	9	Confiden	C	9			2	1
Bear Creek Footwall Zone	auok	1	aucap	auok	13	NUMSAM	SVOL	3	9
	auid	2	aucap	auid	13			2	1
	aunn	3	aucap	aunn	13			1	1
	F	4	aucap	F	13			101	9
	LG	5	aucap	LG	13			101	9
	agok	6	agcap	agok	14			3	10
	agid	7	agcap	agid	14			2	1
	agnn	8	agcap	agnn	14			1	1
	CONF	9	Confiden	C	13			2	1
Bear Creek Footwall Vein	auok	1	aucap	auok	3	NUMSAM	SVOL	3	11
	auid	2	aucap	auid	3			2	1
	aunn	3	aucap	aunn	3			1	1
	F	4	aucap	F	3			101	11
	LG	5	aucap	LG	3			101	11
	agok	6	agcap	agok	4			3	12
	agid	7	agcap	agid	4			2	1
	agnn	8	agcap	agnn	4			1	1
	CONF	9	Confiden	C	3			2	1

Zone	Description	Est Ref #	Value_In	Value_Out	Search Ref #	Numsam_F	Svol_F	Imethod	Vrefnum
Bear Creek Hanging-Wall Zone	auok	1	aucap	auok	15	NUMSAM	SVOL	3	13
	auid	2	aucap	auid	15			2	1
	aunn	3	aucap	aunn	15			1	1
	F	4	aucap	F	15			101	13
	LG	5	aucap	LG	15			101	13
	agok	6	agcap	agok	16			3	14
	agid	7	agcap	agid	16			2	1
	agnn	8	agcap	agnn	16			1	1
	CONF	9	Confiden	C	15			2	1
Bear Creek Hanging-Wall Vein	auok	1	aucap	auok	5	NUMSAM	SVOL	3	15
	auid	2	aucap	auid	5			2	1
	aunn	3	aucap	aunn	5			1	1
	F	4	aucap	F	5			101	15
	LG	5	aucap	LG	5			101	15
	agok	6	agcap	agok	6			3	16
	agid	7	agcap	agid	6			2	1
	agnn	8	agcap	agnn	6			1	1
	CONF	9	Confiden	C	5			2	1
Main Zone	auok	1	aucap	auok	11	NUMSAM	SVOL	3	13
	auid	2	aucap	auid	11			2	1
	aunn	3	aucap	aunn	11			1	1
	F	4	aucap	F	11			101	13
	LG	5	aucap	LG	11			101	13
	agok	6	agcap	agok	12			3	14
	agid	7	agcap	agid	12			2	1
	agnn	8	agcap	agnn	12			1	1
	CONF	9	Confiden	C	11			2	1

Zone	Description	Est Ref #	Value_In	Value_Out	Search Ref #	Numsam_F	Svol_F	Imethod	Vrefnum
Main Vein	auok	1	aucap	auok	1	NUMSAM	SVOL	3	19
	auid	2	aucap	auid	1			2	1
	aunn	3	aucap	aunn	1			1	1
	F	4	aucap	F	1			101	19
	LG	5	aucap	LG	1			101	19
	agok	6	agcap	agok	2			3	20
	agid	7	agcap	agid	2			2	1
	agnn	8	agcap	agnn	2			1	1
	CONF	9	Confiden	C	1			2	1

Table 14.11 - Search Parameters

Element	Srefnum	Search Method	Search Distance - Along Strike (X)	Search Distance - Down Dip (Z)	Search Distance - Across Strike (Y)	Z Axis Rotation	Y Axis Rotation	X Axis Rotation	DA Angle - Z	DA Angle - Y	DA Angle - X
Main Vein_Au	1	ellipse	110	481	60	26	0	120	TRDIPDIR	-	TRDIP
Main Vein_Ag	2	ellipse	300	225	90	26	0	-60	TRDIPDIR	-	TRDIP
BCFW Vein_Au	3	ellipse	144	615	204	26	0	120	TRDIPDIR	-	TRDIP
BCFW Vein_Ag	4	ellipse	334	112	75	26	0	120	TRDIPDIR	-	TRDIP
BCHW Vein_Au	5	ellipse	124	291	66	26	0	110	TRDIPDIR	-	TRDIP
BCHW Vein_Ag	6	ellipse	187	127	93	26	0	110	TRDIPDIR	-	TRDIP
Dyke Adit Vein_ Au	7	ellipse	266	200	176	26	0	120	TRDIPDIR	-	TRDIP
Dyke Adit Vein_ Ag	8	ellipse	100	85	153	26	0	120	TRDIPDIR	-	TRDIP
East Hill Vein_Au	9	ellipse	225	111	75	26	0	120	TRDIPDIR	-	TRDIP
East Hill Vein_Ag	10	ellipse	222	456	5	26	0	120	TRDIPDIR	-	TRDIP
Main Zone_Au	11	ellipse	162	190	69	26	0	120	-	-	-
Main Zone_Ag	12	ellipse	82	177	74	26	0	120	-	-	-
BCFW Zone_Au	13	ellipse	123	91	60	26	0	120	-	-	-
BCFW Zone_Ag	14	ellipse	306	112	89	26	0	120	-	-	-
BCHW Zone_Au	15	ellipse	47	73	66	26	0	110	-	-	-
BCHW Zone_Ag	16	ellipse	158	118	89	26	0	110	-	-	-
Dyke Adit Zone_ Au	17	ellipse	130	75	56	26	0	120	-	-	-
Dyke Adit Zone_ Ag	18	ellipse	144	85	82	26	0	120	-	-	-
East Hill Zone_Au	19	ellipse	187	176	157	26	0	120	-	-	-
East Hill Zone_Ag	20	ellipse	161	133	98	26	0	120	-	-	-
	SVOLFAC1	**Min No. of Samples**	**Max No. of Samples**	**SVOLFAC2**	**Min No. of Samples**	**Max No. of Samples**	**SVOLFA C3**	**Min No. of Samples**	**Max No. of Samples**		
	1	16	35	2	11	35	3	6	35		
	Octant Method	**Min No. of Octant**	**Min/Octant**	**Max/Octant**	**Max Samples/ Borehole**						
	0	2	1	4	5						

14.8 RESOURCE CLASSIFICATION

Several factors are considered in the definition of a resource classification:

- NI 43-101 requirements;
- CIM guidelines;
- Author's experience with epithermal gold deposits;
- Spatial continuity based on variography of the assays within the drillholes;
- Borehole spacing and estimation runs required to estimate the grades in a block;
- Observed mineralization in surface;
- The confidence with the dataset base on the results of the validation;
- The number of samples and boreholes used in each of the block estimations.

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to WSP that may affect the estimate of mineral resources. Mineral reserves can only be estimated on the basis of an economic evaluation that is used in a preliminary feasibility study or a feasibility study of a mineral project; thus, no reserves have been estimated. As per NI 43-101, mineral resources, which are not mineral reserves, do not have to demonstrate economic viability.

14.9 MINERAL RESOURCE TABULATION

The resource reported as of March 24, 2015, has been tabulated in terms of a gold cut-off grade. Figure 14.43 to Figure 14.45 and Table 14.12 to Table 14.14 are the grade- tonnage curve and tables for Talapoosa for each of the resource categories. The resources are tabulated using various cut-off grades to demonstrate the robust nature of the resource.

Table 14.12 - Talapoosa Measured Grade-Tonnage Table

Au Cut-off	Tons	Au (oz/ton)	Ag (oz/ton)
0.009	19,796,320	0.033	0.457
0.010	18,977,490	0.034	0.468
0.011	18,299,250	0.035	0.478
0.012	17,724,880	0.035	0.486
0.013	17,170,870	0.036	0.494
0.014	16,648,010	0.037	0.502
0.015	16,148,890	0.037	0.509
0.016	15,672,670	0.038	0.516
0.017	15,176,050	0.039	0.523
0.018	14,697,140	0.039	0.529
0.019	14,211,340	0.040	0.534
0.020	13,738,730	0.041	0.541

Table 14.13 - Talapoosa Indicated Grade-Tonnage Table

Au Cut-off	Tons	Au (oz/ton)	Ag (oz/ton)
0.009	20,631,200	0.023	0.311
0.010	18,534,500	0.024	0.326
0.011	16,766,500	0.025	0.341
0.012	15,358,900	0.027	0.353
0.013	14,093,700	0.028	0.366
0.014	12,959,500	0.029	0.378
0.015	11,918,900	0.031	0.393
0.016	10,971,600	0.032	0.407
0.017	10,114,000	0.033	0.421
0.018	9,333,900	0.034	0.434
0.019	8,628,500	0.036	0.447
0.020	8,035,400	0.037	0.459

Table 14.14 - Talapoosa Inferred Grade-Tonnage Table

Au Cut-off	Tons	Au (oz/ton)	Ag (oz/ton)
0.009	20,129,000	0.016	0.190
0.010	16,964,000	0.018	0.196
0.011	14,512,000	0.019	0.195
0.012	12,489,000	0.020	0.194
0.013	11,198,000	0.021	0.194
0.014	9,723,000	0.022	0.190
0.015	7,879,000	0.024	0.179
0.016	7,085,000	0.025	0.176
0.017	6,096,000	0.026	0.170
0.018	5,451,000	0.027	0.165
0.019	4,813,000	0.028	0.161
0.020	4,229,000	0.029	0.159

Figure 14.43 - Talapoosa Measured Grade Tonnage Curve

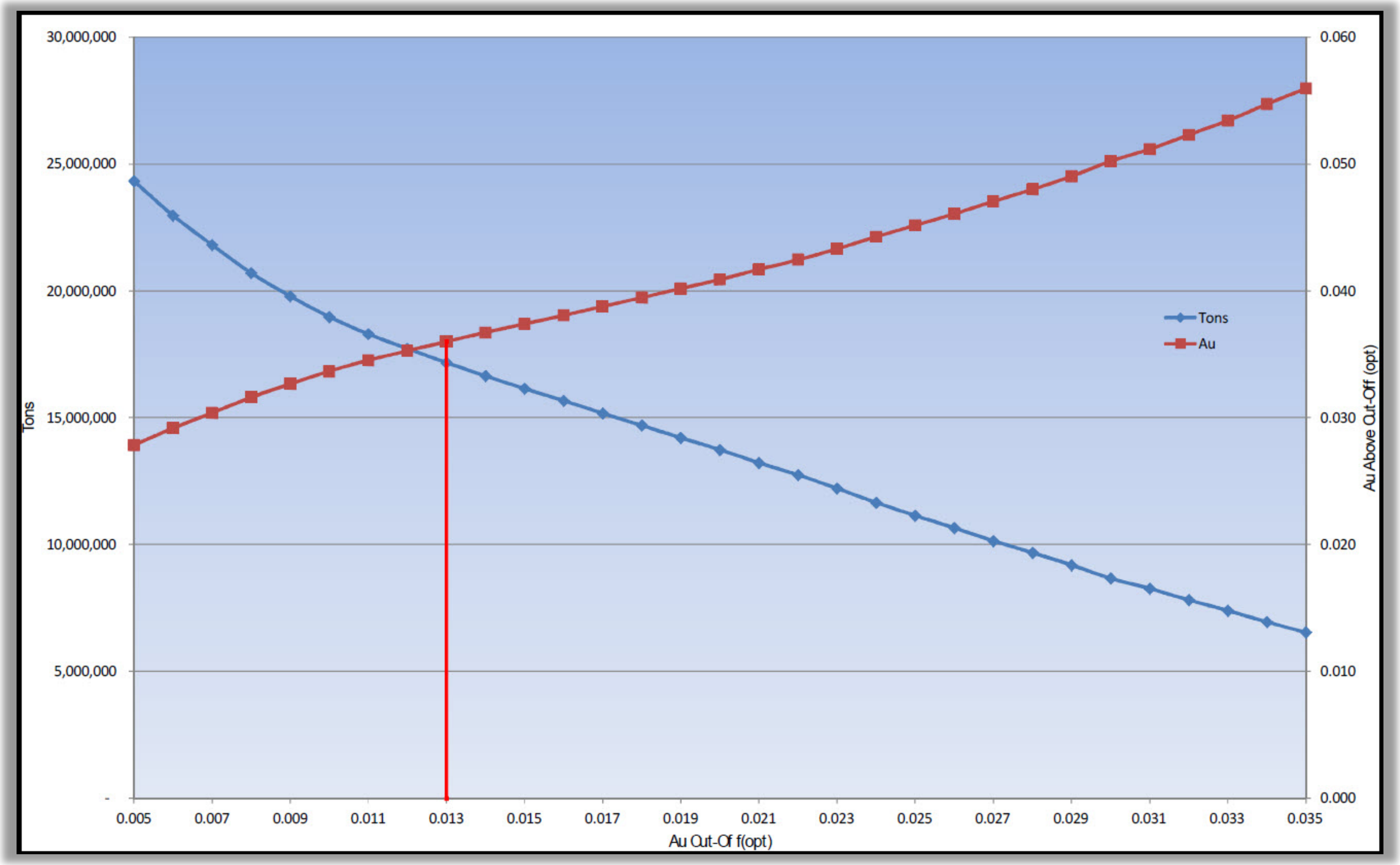


Figure 14.44 - Talapoosa Indicated Grade Tonnage Curve

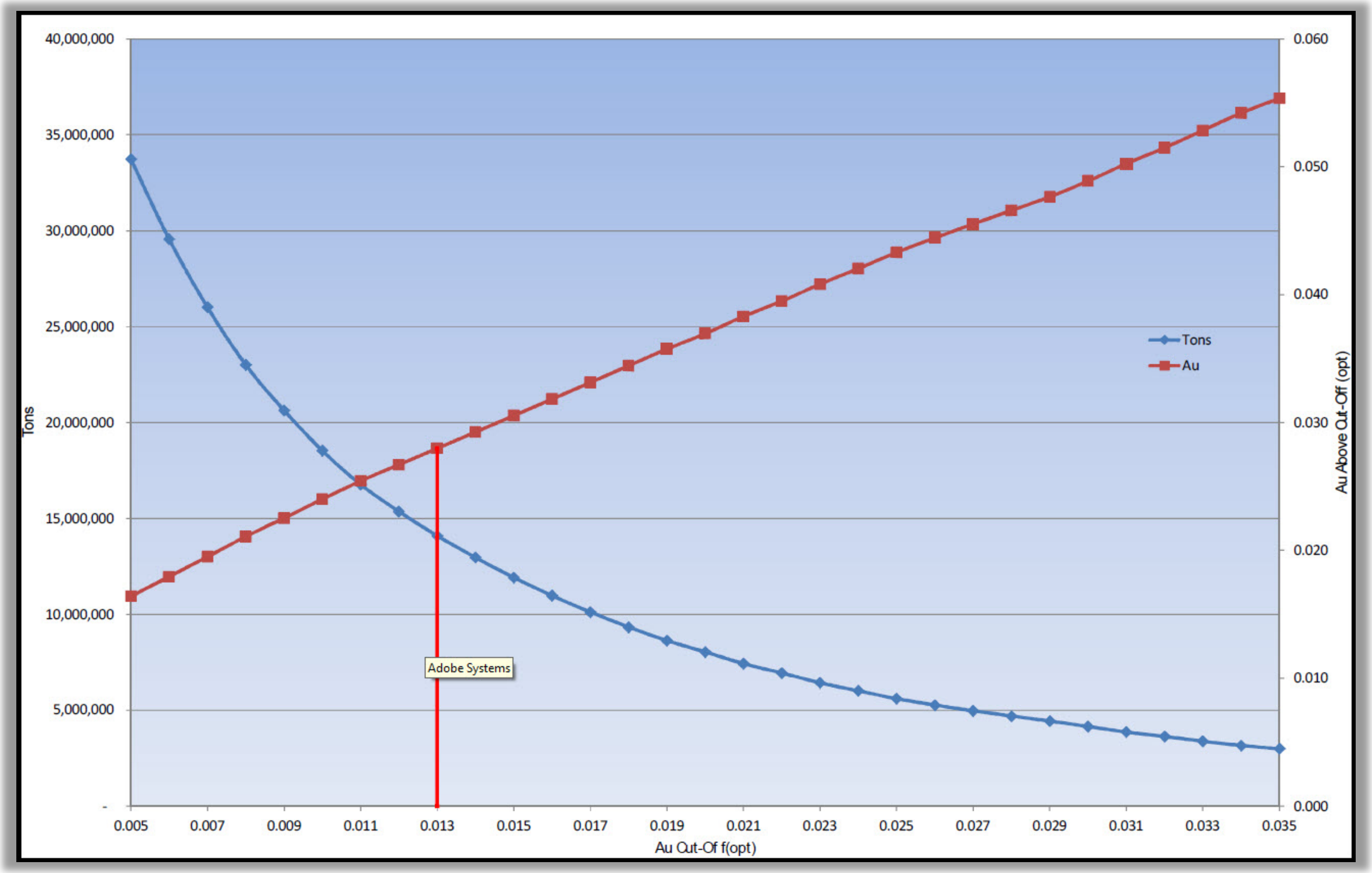
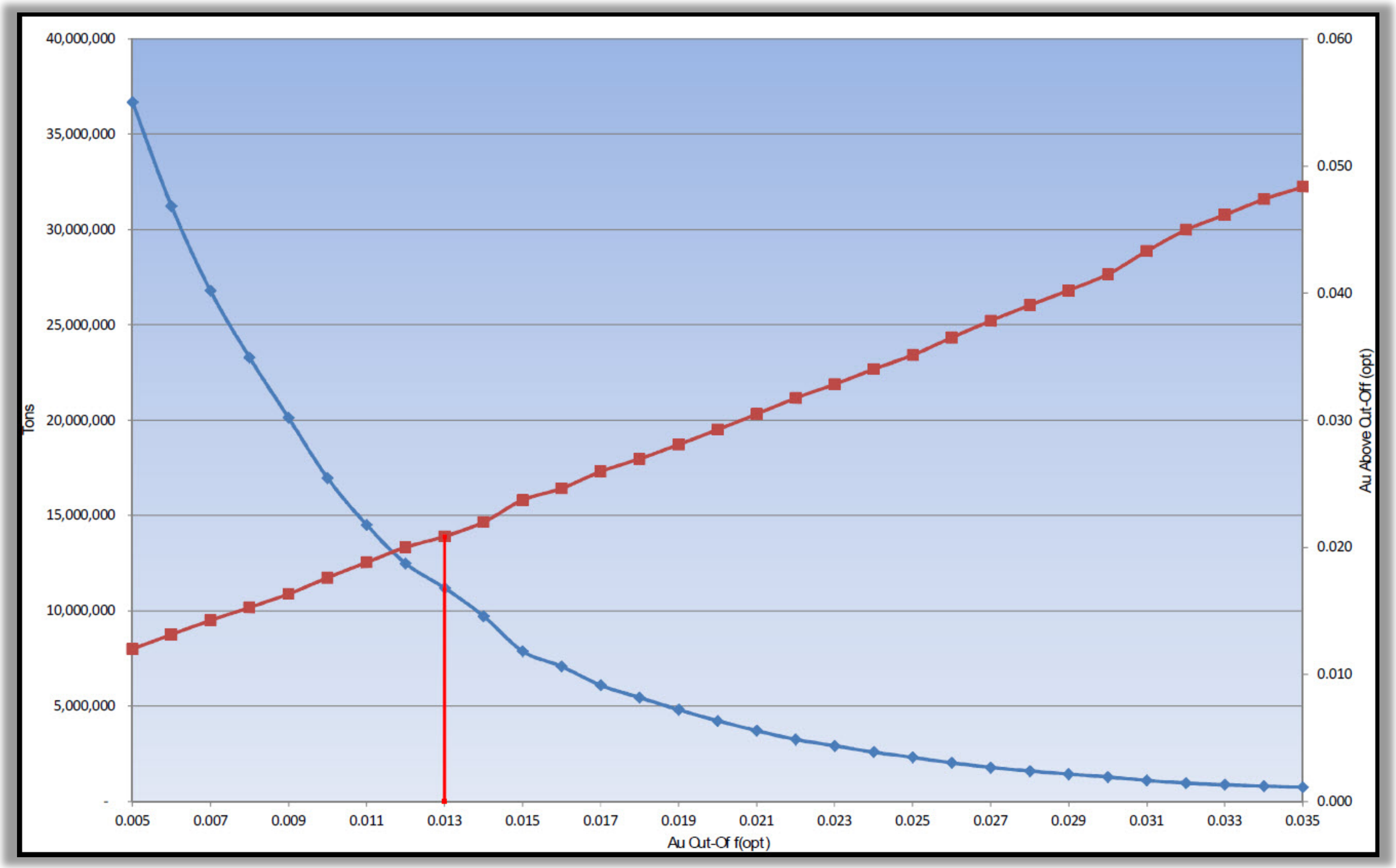


Figure 14.45 - Talapoosa Inferred Grade Tonnage Curve



Based on mines operating in the region and a gold price of $1,507/oz, a 0.013 oz/ton gold cut-off was used to tabulate the resource. Table 14.15 summarizes the resource estimate for each of the resource categories at Talapoosa. A cut-off grade based on the $1,507/oz of gold could be considered high based on current pricing. Using a higher cut-off based on a lower gold price, could decrease the resource tonnage and increase the grade, with little impact on the in-situ contained ounces. The grade tonnage table show that the resource model is robust enough to handle a higher cut-off grade.

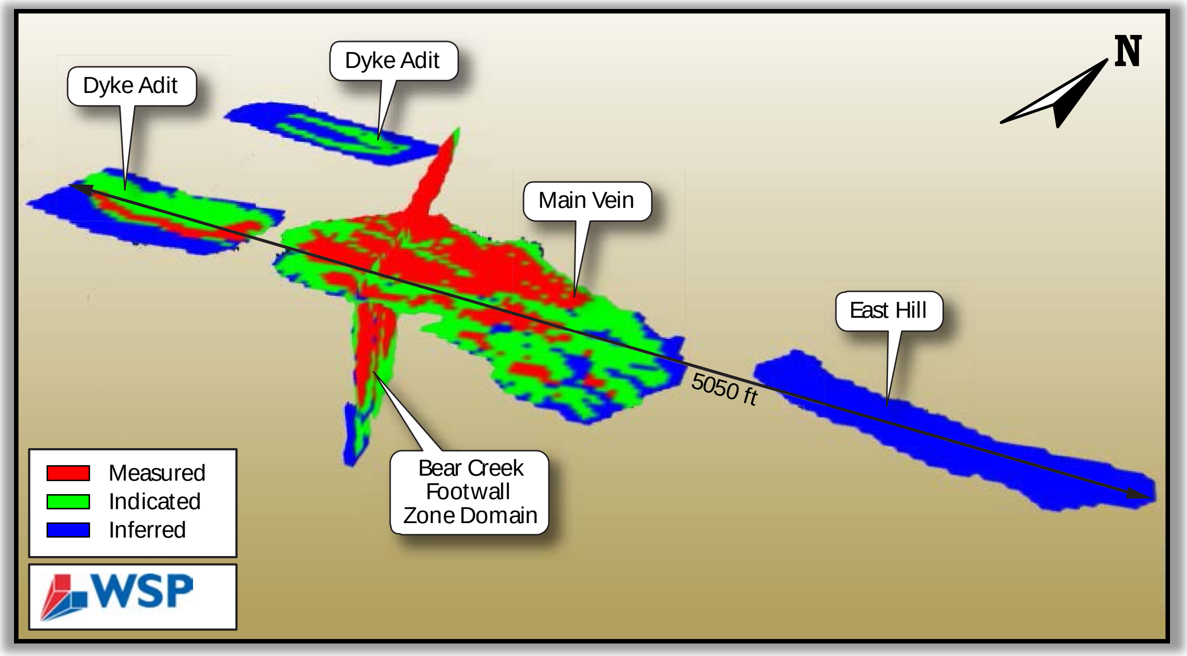
Table 14.15 - Talapoosa Mineral Resource Summary

	Cut-off (oz/ton)	Tons	Au (oz/ton)	Ag (oz/ton)	Au (oz)	Ag (oz)
Summary						
Oxide Measured	0.013	3,126,050	0.038	0.553	117,253	1,728,323
Sulphide Measured	0.013	14,044,820	0.036	0.481	501,215	6,760,763
Total Measured	**-**	**17,170,870**	**0.036**	**0.494**	**618,468**	**8,489,086**
Oxide Indicated	0.013	1,412,000	0.032	0.416	45,328	586,999
Sulphide Indicated	0.013	12,681,600	0.028	0.361	349,005	4,573,274
Total Indicated	**-**	**14,093,600**	**0.028**	**0.366**	**394,334**	**5,160,273**
Total Measured and Indicated	**-**	**31,264,470**	**0.032**	**0.437**	**1,012,802**	**13,649,358**
Oxide Inferred	0.013	1,762,000	0.027	0.065	47,745	115,115
Sulphide Inferred	0.013	9,436,000	0.020	0.218	185,787	2,057,651
Total Inferred		**11,198,000**	**0.021**	**0.194**	**233,532**	**2,172,766**
Oxide						
Main Zone	0.013	1,773,770	0.033	0.387	58,797	686,978
Bear Creek Footwall Zone	0.013	392,780	0.030	0.555	11,663	217,902
Bear Creek Hanging-Wall Zone	0.013	116,050	0.028	0.333	3,257	38,597
Dyke Adit	0.013	843,450	0.052	0.931	43,536	784,846
East Hill	0.013	-	0.000	0.000	-	-
Measured Subtotal	**0.013**	**3,126,050**	**0.038**	**0.553**	**117,253**	**1,728,323**
Main Zone	0.013	419,300	0.026	0.384	10,898	160,818
Bear Creek Footwall Zone	0.013	436,400	0.023	0.401	10,164	174,998
Bear Creek Hanging-Wall Zone	0.013	353,600	0.027	0.272	9,629	96,300
Dyke Adit	0.013	202,700	0.072	0.764	14,637	154,883
East Hill	0.013	-	0.000	0.000	-	-
Indicated Subtotal	**0.013**	**1,412,000**	**0.032**	**0.416**	**45,328**	**586,999**
Oxide Measured & Indicated Total	**-**	**4,538,050**	**0.036**	**0.510**	**162,581**	**2,315,321**
Main Zone	0.013	93,000	0.021	0.311	1,960	28,889
Bear Creek Footwall Zone	0.013	3,000	0.017	0.371	51	1,112
Bear Creek Hanging-Wall Zone	0.013	183,000	0.022	0.346	3,989	63,395
Dyke Adit	0.013	33,000	0.015	0.367	511	12,117
East Hill	0.013	1,450,000	0.028	0.007	41,234	9,602
Oxide Inferred Total	**0.013**	**1,762,000**	**0.027**	**0.065**	**47,745**	**115,115**

	Cut-off (oz/ton)	Tons	Au (oz/ton)	Ag (oz/ton)	Au (oz)	Ag (oz)
Sulphide						
Main Zone	0.013	3,235,140	0.027	0.330	87,219	1,066,333
Bear Creek Footwall Zone	0.013	5,147,790	0.033	0.496	169,891	2,555,726
Bear Creek Hanging-Wall Zone	0.013	5,258,210	0.042	0.555	223,327	2,915,997
Dyke Adit	0.013	403,680	0.051	0.552	20,778	222,707
East Hill	0.013	-	0.000	0.000	-	-
Measured Subtotal	**0.013**	**14,044,820**	**0.036**	**0.481**	**501,215**	**6,760,763**
Main Zone	0.013	2,154,100	0.025	0.320	54,808	689,749
Bear Creek Footwall Zone	0.013	4,976,700	0.025	0.339	122,447	1,685,319
Bear Creek Hanging-Wall Zone	0.013	4,711,000	0.030	0.370	139,614	1,744,948
Dyke Adit	0.013	839,800	0.038	0.540	32,136	453,258
East Hill	0.013	-	0.000	0.000	-	-
Indicated Total	**0.013**	**12,681,600**	**0.028**	**0.361**	**349,005**	**4,573,274**
Sulphide Measured & Indicated Total	**-**	**26,726,420**	**0.032**	**0.424**	**850,220**	**11,334,037**
Main Zone	0.013	392,000	0.023	0.242	8,948	95,018
Bear Creek Footwall Zone	0.013	149,000	0.029	0.221	4,353	32,988
Bear Creek Hanging-Wall Zone	0.013	5,513,000	0.020	0.231	107,950	1,271,444
Dyke Adit	0.013	2,000,000	0.016	0.276	31,503	552,808
East Hill	0.013	1,382,000	0.024	0.076	33,033	105,393
Sulphide Inferred Total	**0.013**	**9,436,000**	**0.020**	**0.218**	**185,787**	**2,057,651**

The distribution of the resource categories is displayed in Figure 14.46.

Figure 14.46 - Talapoosa Resource Category Distribution



14.10 VALIDATION

The Talapoosa model was validated by three methods:

1. Visual comparison of colour-coded block model grades with composite grades on section and plan.
2. Comparison of the global mean block grades for OK, ID^2, NN and composites.
3. Swath plots of the various zones in both plan and section views.

14.10.1 VISUAL VALIDATION

The visual comparisons of block model grades with composite grades for each of the zones show a reasonable correlation between the values. No significant discrepancies were apparent from the sections reviewed, yet grade smoothing is apparent in some locations due to the distance between drill samples being broader in some regions.

Figures 14.47 and 14.48 display the comparison between the block model and the original drillholes.

Figure 14.47 - Talapoosa Cross-Section

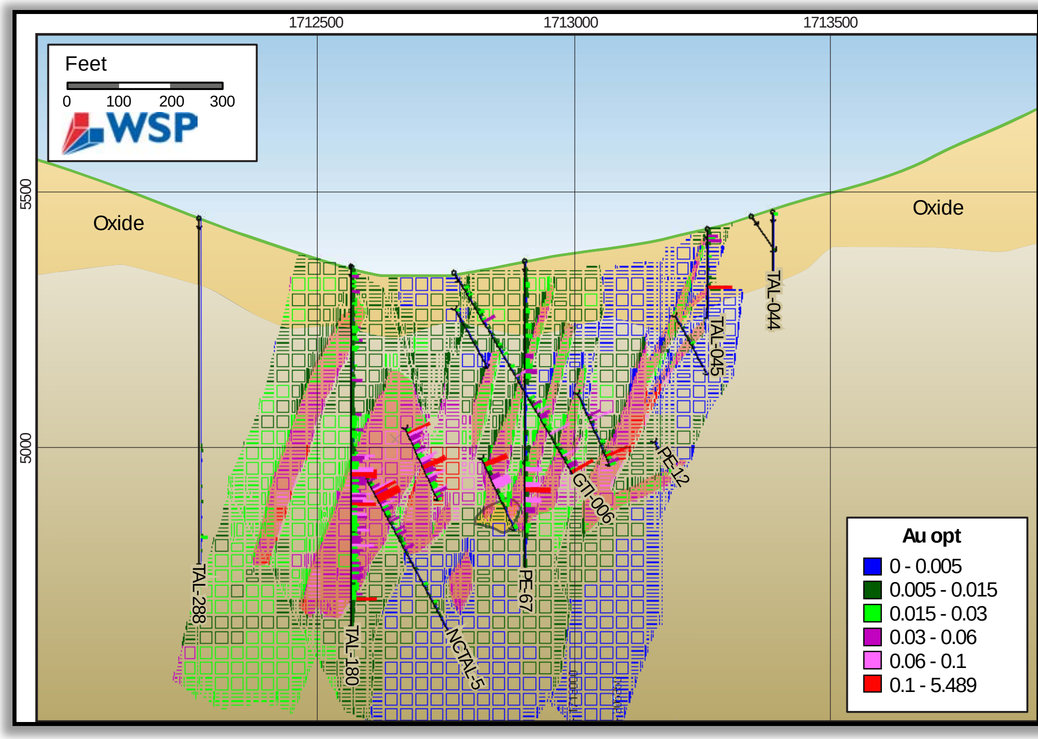
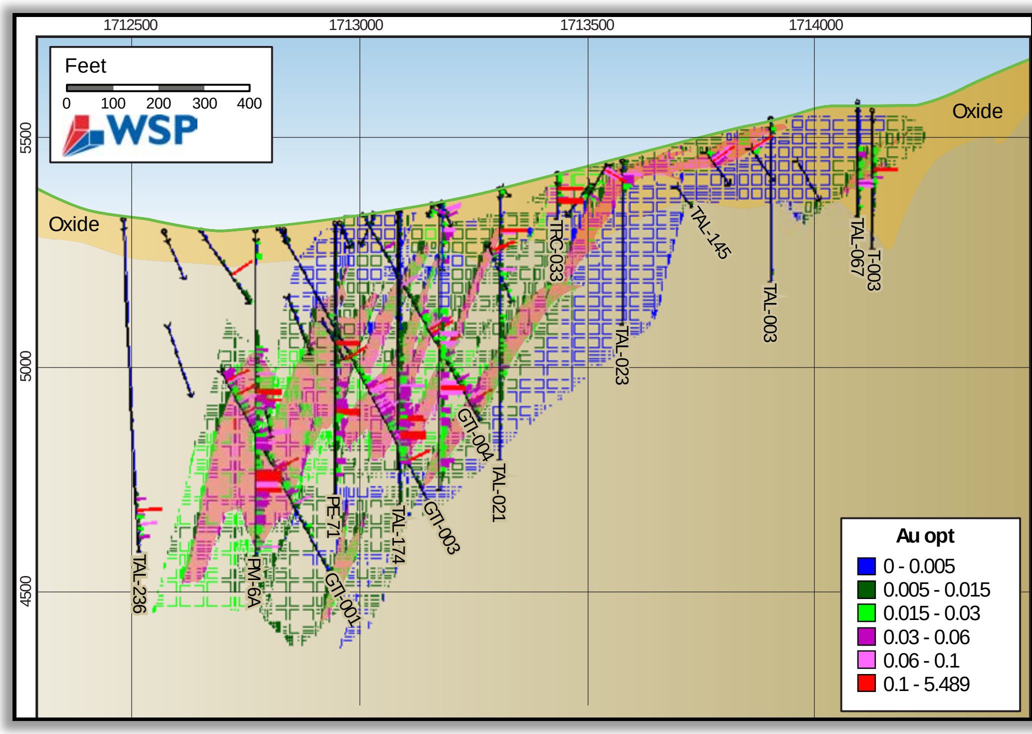


Figure 14.48 - Talapoosa Cross-Section 2



14.10.2 GLOBAL COMPARISON

The global block model statistics for the OK model were compared to the global ID^2 and NN model values as well as the composite capped drillhole data. Table 14.16 shows this comparison of the global estimates for the three estimation method calculations. In general, there is agreement between the OK model, the ID^2 model, and the NN model. Larger discrepancies are reflected as a result of lower drill density in some portions of the model. There is a degree of smoothing apparent when compared to the diamond drill statistics. Comparisons were made using all blocks at a 0 oz/ton gold cut-off.

Table 14.16 - Talapoosa Global Statistical Comparison

Field	Minimum	Maximum	Mean	Standard Deviation
auok	0.0003	0.448	0.019	0.020
auid	0.0004	0.377	0.018	0.020
aunn	0.0001	0.686	0.018	0.037
agok	0.0015	6.603	0.256	0.330
agid	0.0015	5.750	0.253	0.318
agnn	0.0005	9.600	0.250	0.591

14.10.3 SWATH PLOTS

Swath plots of eastings, northings, and elevations were generated for the Talapoosa resource. These plots are comparing the OK estimates with the NN and ID^2 estimates and are illustrated in Figures 14.49, to 14.51. There is a good correlation between the three estimation methods.

Figure 14.49 - Talapoosa Easting Plot

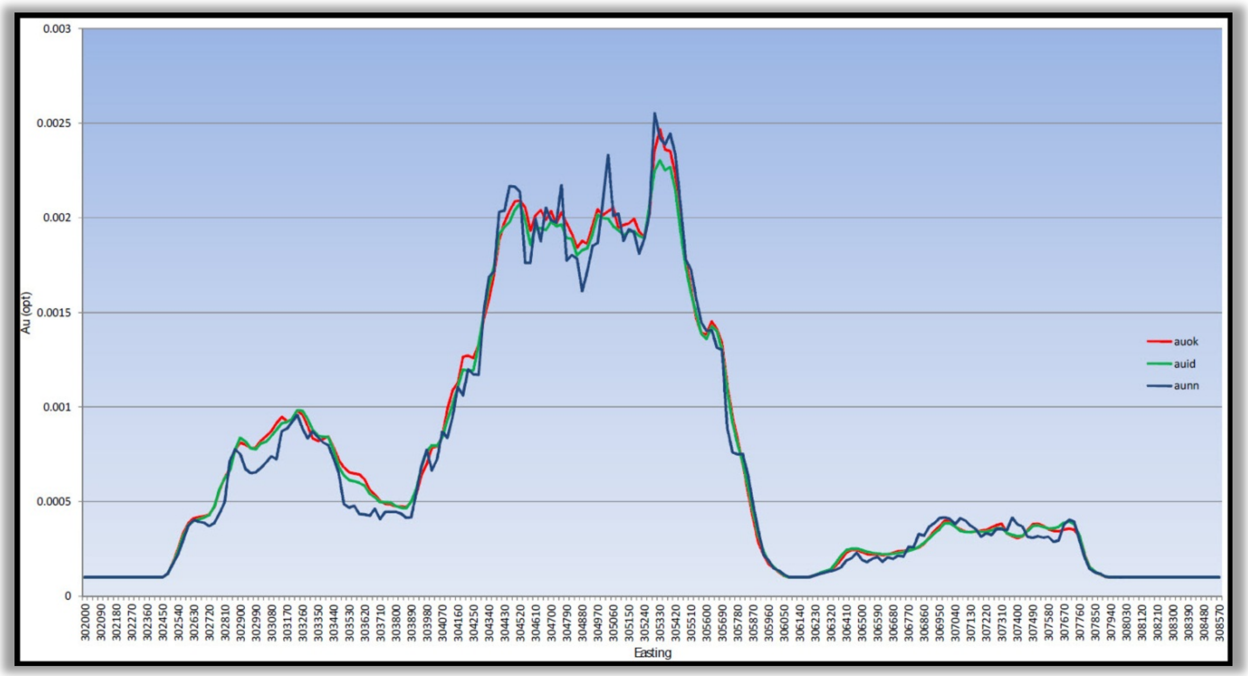


Figure 14.50 - Talapoosa Northing Plot

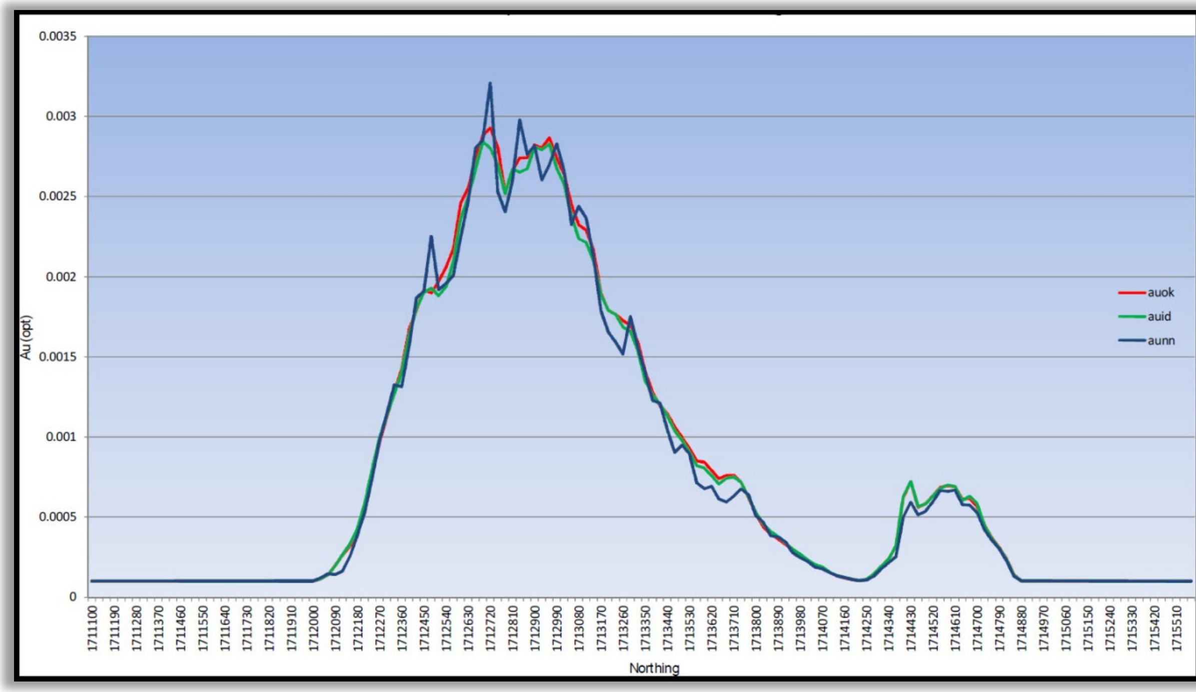
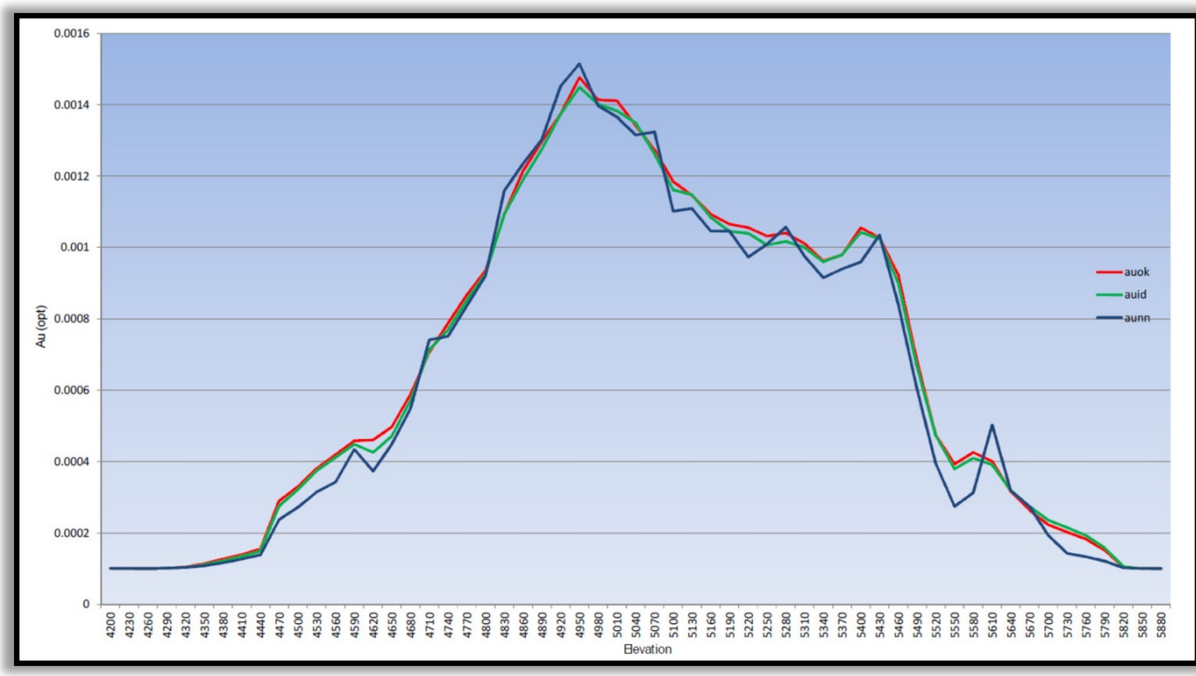


Figure 14.51 - Talapoosa Elevation Plot



14.11 PREVIOUS ESTIMATES

American Gold commissioned MDA to generate a resource estimate in 2010. This estimate was based on the interpretation of the geology at the time.

Table 14.17 compares the basic parameters of the previous 2010 estimate with the current 2013 mineral resource.

Table 14.17 - Modelling Parameter Comparison

	2010 MDA	2013 Tetra Tech Model
Number of Drillholes in Database	586 (not all holes used in the estimation)	545 used in the estimation process
Grade Capping	Vein and Breccia: 1.000 oz/ton gold and 10 oz/ton silver Disseminated: 0.250 oz/ton gold and 4.00 oz/ton silver Outside: 0.250 oz/ton gold and 2.00 oz/ton silver	Global 0.686 oz/ton gold and 9.60 oz/ton silver
Composite Length	10 ft downhole	5 ft average, back stitching allows for "tail" material to be spread evenly over the entire hole composite
Cut-off Grade	0.015 oz/ton gold equivalent	0.013 oz/ton gold
Specific Gravity	Quartz Vein: 2.70 Post Mineral: 2.40 Background: 2.60	Quartz Vein or Breccia: 0.078 t/ft3 (2.50) Altered Host Rock: 0.072 t/ft3 (2.32) Oxidized Host Rock: 0.067 t/ft3 (2.14)
Mineral Domains	2 (oxide and un-oxidized)	2 (High grade vein and Altered host rock
Number of Mineral Zones	1	5 Dyke Adit, East Hill, Bear Creek Hanging-Wall, Bear Creek Footwall and Main
Block Size	25 ft by 25 ft by 25 ft	30 ft by 30 ft by 30 ft (27000 ft3) with sub-celling
Estimation Method	OK with inverse distance cubed (ID3) and NN validation	OK with ID2 and NN validation

The primary difference between the 2010 resource model and the 2013 resource model is due to constraining the high grade material within the vein systems. This reduces the amount of grade smearing across the model and helps restrict the influence of the lower grade host rock material supressing the grades within the veins.

The Tetra Tech interpretation is volumetrically larger than the MDA 2010 model, yet Tetra Tech used lower specific gravity values based on a significantly larger specific gravity sample data set. The result is an increase in the reported tonnage and contained gold and silver by Tetra Tech.

Tetra Tech opted not to use a gold equivalent cut-off. Silver could be a recoverable by- product, yet at this time, the deposit is focused on the gold content.

Table 14.18 illustrates the differences in the 2010 resource estimate with the current NI 43-101 compliant resource from 2013.

Table 14.18 - Comparison of the 2010 and 2013 Resource Model

	Tons	Au (oz/ton)	Ag (oz/ton)	Au (oz)	Ag (oz)
2010 MDA Resource					
Measured Resource @ 0.015 oz/ton gold equivalent cut-off	1,065,000	0.032	0.499	34,000	531,000
Indicated Resource @ 0.015 oz/ton gold equivalent cut-off	21,986,000	0.027	0.350	598,000	7,695,000
Inferred Resource @ 0.015 oz/ton gold equivalent cut-off	12,594,000	0.026	0.338	326,000	4,257,000
2013 Tetra Tech Resource					
Measured Resource @ 0.013 oz/ton gold cut-off	17,170,870	0.036	0.494	618,000	8,489,000
Indicated Resource @ 0.013 oz/ton gold cut-off	14,093,700	0.028	0.366	394,000	5,160,000
Inferred Resource @ 0.013 oz/ton gold cut-off	11,198,000	0.021	0.194	234,000	2,173,000

15 ADJACENT PROPERTIES

There are no material properties adjacent to the Property.

16 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information that is material to this report.

17 INTERPRETATION AND CONCLUSIONS

17.1 GEOLOGY

The conclusions for the geology and resource of the Project are summarized below.

- The Property is currently controlled 100% by Timberline through its option to purchase agreement with Gunpoint.
- The Property is analogous to the low-sulphidation epithermal gold deposits typical to the western Basin and Range of Nevada.
- The Property is associated with sheared felsic to intermediate volcanics flows and tuff with intercalated sediments. Varying degrees of alteration are present including carbonate, silicification, sericitization and minor chloritization.
- Timberline has a strong understanding of the regional and local geology to support the interpretation of the mineralized zones on the Property.
- Mineralization is currently defined in five zones of various thicknesses over a strike length of the deposit.
- Drilling and sampling procedures, sample preparation and assay protocols conducted by the previous operator (Gunpoint) are generally conducted in agreement with best practices.
- Drilling and sampling procedures, sample preparation and assay protocols conducted by previous operator are generally conducted in agreement with best practices at the time, yet may not meet current standards.
- Verification of the drillhole collars, surveys, assays, core and drillhole logs indicates the previous operator's (Gunpoint) data is reliable.
- Based on the QA/QC program, the data is sufficiently reliable to support the resource estimate generated for the five zones on the Property.
- The mineral models have been constructed in conformance to industry standard practices.
- The geological understanding is sufficient to support the resource estimation.
- At a gold cut-off grade of 0.013 oz/ton gold, the combined Measured and Indicated Resource of in situ material is 17.2 Mt with an average grade of 0.036 oz/ton gold and 0.494 oz/ton silver. The Inferred Resource totals 11.2 Mt with an average grade of 0.021 oz/ton gold and 0.194 oz/ton silver.
- It is WSP's opinion that there is no material change to the resource model. No additional drilling or sampling has been completed on the deposit that would impact the tons or grades. The metallurgical test work confirms the recoveries from historical test work and the in-situ ounces remain material unchanged with a slight higher cut-off grade due to a slight lower gold price.
- The specific gravity value used to determine that tonnage was derived from a larger data set than used in previous estimates.

17.2 METALLURGY

The conclusions from the historical test work are summarized below.

- → The Main Zone samples tested showed amenability to column and agitated cyanidation, but generally were not amenable to flotation. This is possibly due to a lack of refractory sulphides and gold and silver locked in oxide minerals.
- → The Bear Creek Zone samples tested showed amenability to flotation, and were less amenable to column and agitated cyanidation, than the Main Zone samples. The presence of refractory sulphides in this zone is responsible for both the higher flotation recoveries and the lower leach recoveries. Some samples of the Bear Creek hanging-wall showed an amenability to agitated cyanidation, so the footwall and hanging-wall may behave differently.
- → The gold occurred mainly in gold/silver minerals such as argentian gold, acanthite, and electrum. The electrum was present within pyrite as a fine particle (i.e. less than 30 µm). The gold particle sizes varied in size from 200 µm down to a few microns in size.
- → Silver was present as acanthite native silver, electrum, and argentian gold.
- → When the silver recovery increased, the gold recovery increased.
- → The presence electrum appeared to cause slow gold and silver leach kinetics.
- → Agglomerating column leach feed with sodium cyanide and cement will likely increase leach kinetics and help to achieve a higher final precious metal recovery.
- → The use of HPGR for the size reduction generally leads to increased heap leach (column test) gold recoveries. It is unclear whether those increases result from a finer particle size distribution or the creation of micro-fractures in the feed which may help to increase the kinetics and final precious metal recoveries.
- → The samples tested are very sensitive to feed size, and will benefit from a fine crush before heap leaching. The highest precious metal recoveries were obtained by agitated cyanidation at a grind of P_{80} = -75 µm.
- → Gravity separation techniques employed as a pre-concentration step to flotation and to leaching did not help to increase precious metal recoveries. However, the technique should still be tested in future work to determine if it can be used to remove the electrum and possibly aid in increasing leach kinetics.
- → Bio-oxidation of the sulphide zone feeds did not have a significant impact on precious metal column leach recoveries. Bio-oxidation of flotation concentrate prior to leach might aid in increasing the leach recoveries for the precious metals. This has not yet been tested.
- → The addition of lead oxide (500 g/t) did significantly increase the silver leach recovery and slightly increased the gold leach recovery during agitated cyanidation leaching.

18 RECOMMENDATIONS

18.1 GEOLOGY AND METALLURGY

It is the author's opinion that additional exploration expenditures are warranted to improve the viability of the project. It is recommended that Timberline undertake a two- phased program that will concentrate on the evaluating the open pit potential of the mineralized deposit and complete additional metallurgical studies and step out drilling along strike of the known resource. The initiation of Phase 2 is contingent on the completion of Phase 1.

Further metallurgical test work is warranted for Talapoosa. New samples representative of the new mineable resource would need to be tested to determine if there are any new factors to be considered with respect to the mineralogy. The "East Hill" and "Dyke Adit" zones have been subjected to minimal test work since most of the previous test work focussed on the Main and Bear Creek Zones. As noted previously, the hanging-wall and footwall portions of the Bear Creek Zone have shown some different amenabilities to several processes, and this should be investigated further as well.

Trade off studies will also need to be run to determine what process scheme will be the most efficient and economic method to extract the precious metals. Gravity, HMS, flotation, agitated and column leach have all been tested previously, and different zones of the Project behave differently for each method. Different combinations will need to be tested to determine the best combination of processes will be optimal for the entire deposit. Different methods might need to be employed for each zone. Possible process combination to be tested should include:

- HPGR/heap leach (agglomeration with sodium cyanide and leach aid)
- Mill/gravity separation/agitated leach of gravity tails
- Mill/gravity separation/flotation of gravity tails/fine regrind float concentrate and agitated leach.
- HPGR/agglomerated heap leach oxidized zones (e.g. Main) and separate mill/flotation and agitated leach of regrind flotation concentrate for sulphide zones (e.g. Bear Creek)

Further test work will be required to optimize the flotation and leach work. More work is required for the leaching of flotation concentrates and possibly oxidation of the flotation concentrates prior to agitated leaching. A comminution study will be required to determine the hardness of the material in each zone, so that the proper crushing and grinding equipment can be selected and sized.

18.1.1 PHASE 1

Phase 1 will be to complete a Preliminary Economic Assessment (PEA) of the project based on an open pit mine with heap leach processing. . The PEA is estimated to cost $145,000. Table 18.1 summarizes the Phase 1 program proposed.

Table 18.1 - Proposed Phase 1 Historical Metallurgical Test Work Review and Evaluation

	Unit Rate ($)	No. of Units	Unit	Cost ($)
Preliminary Economic Assessment	145,000	1	unit	145,000

18.1.2 PHASE 2A – METALLURGICAL DRILLHOLE AND RESOURCE EXPANSION PROGRAM

The scope of Phase 2 of the program is dependent on the results from Phase 1. The decision of where to drill metallurgical sample holes will depend on the review in Phase 1.

The principal objectives of the program will be to:

- Diamond drill across the mineralized zones with emphasis on the sulphide and oxide horizons for the collection of material for metallurgical testing
- Expand the resource between the East Hill zone and the Main Zone
- Expand the resource between the Dyke Adit Zone and the Main Zone
- Convert Dyke Adit and East Hill Zones from Inferred to Indicated Resource
- Extend the Dyke Adit resource to the northwest
- Extend the Bear Creek zones to the southeast.

The program is estimated to cost $924,000. Table 18.2 summarizes the Phase 2A program proposed.

Table 18.2 - Proposed Phase 2A Diamond Drill Program

	Unit Rate ($)	No. of Units	Unit	Cost ($)
Diamond Drilling	175	5,000	m	875,000
Transportation and Accommodation	2,000	7	months	14,000
Operations Support	5,000	7	months	35,000
Program Costs (all in)	-	-	-	924,000

18.1.3 PHASE 2B – METALLURGICAL TEST PROGRAM

The metallurgical test program will utilize the metallurgical sample gathered in Phase 2A. The program will test the metallurgical process combinations discussed in Section 18.1 and other work evolving from Phase 1. Test work should include further optimization of heap leach processing of the oxide materials, and an extensive ore variability testing program.

The program is estimated to cost $600,000. Table 18.3 summarizes the Phase 2B program proposed.

Table 18.3 - Proposed Phase 2B Exploration Program

	Unit Rate ($)	No. of Units	Unit	Cost ($)
Metallurgical Testing	600,000	1	unit	600,000

18.1.4 OTHER RECOMMENDATIONS

The following recommendations are based on observations made during the site visits or during the resource estimation process. These recommendations are suggestions to policy and procedures to be conducted by Timberline.

- Continue the collection of specific gravity samples for the various rock types and mineralization styles. The accurate representation of specific for the various rock types will provide a better estimation of the tonnages for both the mineralized and un-mineralized material.
- Continue to assay for gold using the screen metallic procedure. The results continue to provide results that are 10 to 20% higher compared to the fire assays.

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20 CERTIFICATES OF QUALIFIED PERSONS

TODD MCCRACKEN, P.GEO.

I, Todd McCracken, P.Geo. of Sudbury, Ontario, do hereby certify:

- I am a Geologist with WSP Canada Inc. with a business address at 2565 Kingsway, Unit 2, Sudbury, Ontario, P3B 3V3.
- This certificate applies to the technical report entitled Technical Report and Resource Estimate on the Talapoosa Project, Nevada, dated March 24, 2015 (the "Technical Report").
- I am a graduate of the University of Waterloo (B.Sc. Honours, 1992). I am a member in good standing of the Association of Professional Geoscientists of Ontario, License #0631. My relevant experience includes 23 years of experience in exploration, operations and resource estimations, including several years working in epithermal gold deposits. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was September 23 to 25, 2012, inclusive.
- I am responsible for Sections 1.1 to 1.4, 1.6, 1.7, 2.0 to 12.0, 14.0 to 16.0, 17.1, 18.0, 19.1 and 20.0 of the Technical Report.
- I am independent of Timberline Resources Inc. Gunpoint Exploration Ltd. and the Talapoosa Property as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that is the subject of the Technical Report, having been a QP on a previous technical report for Gunpoint Exploration
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 24th day of March, 2015 at Sudbury, Ontario.

"Original document signed and stamped by Todd McCracken, P. Geo."

Todd McCracken, P. Geo.
Manager - Geology
WSP Canada Inc.

JACK MCPARTLAND, M.S. METALLURGICAL ENGINEERING

I, Jack McPartland do hereby certify:

- I am a Metallurgist / V.P. Operations with McClelland Laboratories, Inc., at 1016 Greg Street, Sparks, NV, 89431.
- This certificate applies to the technical report entitled Technical Report and Resource Estimate on the Talapoosa Project, Nevada, dated March 24, 2015 (the “Technical Report”).
- I am a graduate of the University of Nevada (BS Chemical Engineering – 1986 and MS Metallurgical Engineering – 1989). I am a member in good standing of the Mining and Metallurgical Society of America, with certification as a Qualified Professional (QP) Member with special expertise in Metallurgy/Processing (Member No. 01350QP). My relevant experience includes 28 years of experience in metallurgical testing design, interpretation and study management, including extensive work on gold and silver deposits. I am a “Qualified Person” for the purposes of National Instrument 43-101 (“the Instrument”).
- I have never visited the Property.
- I am responsible for Sections 1.5, 13, 17.2, portions of 18.1 and 19.2 of the Technical Report.
- I am independent of Timberline Resources Inc. Gunpoint Exploration Ltd. and the Talapoosa Property as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that includes supervision of some of the completed metallurgical testing programs for the Property as well as preparation of an earlier review of the Property metallurgy.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 24th day of March, 2015 at Sparks, Nevada.

“Original document signed by Jack McPartland”

Jack McPartland,
M.S. Metallurgical Engineering
Metallurgist / V.P. Operations
McClelland Laboratories, Inc.