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PRELIMINARY ECONOMIC ASSESSMENT ON THE TALAPOOSA PROJECT

NEVADA

AUGUST 2015

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Timberline Resources

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Prepared by:

Todd McCracken, P. Geo.
Joanne Robinson, P. Eng.
Richard Jolk, P.E.
Jack McPartland, Minerals Engineer
Michael Henderson, P. Eng.
Richard DeLong, P.G.

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WSP Canada Inc.
2565 Kingsway, Unit 2
Sudbury, ON   P3B 2G1

Phone: +1 705-674-0119
Fax: +1 705-674-0125
www.wspgroup.com

SIGNATURES

PREPARED BY

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

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

Original signed and stamped by Joanne Robinson, P.Eng.

Joanne Robinson, P. Eng.
Principal Mine Engineer, WSP Canada Inc.

Original signed and stamped by Richard Jolk, Minerals Engineer.

Richard Jolk, P.E.
Minerals Engineer, Founder and Principal, MPDI Inc.

Original signed and stamped by Jack McPartland, Metallurgical Engineer

Jack McPartland, Metallurgical Engineer
Metallurgist / V.P. Operations, McClelland Laboratories Inc.

Original signed and stamped by Michael Henderson, P.E.

Michael Henderson, P.E.
Practice Area Leader - Mining, DOWL LLC.

Original signed and stamped by Richard DeLong, P.G.

Richard DeLong, P.G.
President, Enviroscientists, Inc.

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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 m^2)	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	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
ounce	oz
ounce per short ton	oz/st
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 .... km^2
square metre .... m^2
three-dimensional 3D
tonne (1,000 kg) (metric ton) .... t
tonnes per day .... tpd or 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

AAL .... American Assay Laboratories
AAS .... Atomic Absorption Spectroscopy
American Gold .... American Gold Capital US Inc.
ASARCO .... American Smelting and Refining Company
Athena .... Athena Gold Inc.
Ag .... Silver
ANFO .... Ammonia Nitrate Fuel Oil
Au .... Gold
BAPC .... Bureau of Air Pollution Control
Bateman .... Bateman Metallurgical Laboratories Inc.
Bear Creek .... Bear Creek Mining Company
BLM .... Bureau of Land Management
BMRR .... Bureau of Mining Regulation and Reclamation
Bondar Clegg .... Bondar Clegg & Company Ltd.
CAPEX .... Capital Cost Estimate
Chesapeake .... Chesapeake Gold Corp.
CIL .... Carbon-in-Leach
CIM .... Canadian Institute of Mining, Metallurgy and Petroleum
CIP .... Carbon in Pulp
CVAA .... Cold Vapour Atomic Absorption
Dawson .... Dawson Metallurgical Laboratories Inc.
DML .... Fast Rolls at Dawson
DOWL .... DOWL LLC
EA .... Environmental Assessment
EIS .... Environmental Impact Statement
ES .... Enviroscientists, Inc.
FE .... Final Effluent
FLAA .... Flameless Atomic Absorption
FW .... Footwall
G&A .... General and Administrative
GPS .... Global Positioning System
Gunpoint .... Gunpoint Exploration Ltd.
Hazen .... Hazen Research Inc.
HLC .... Heinen-Lindstrom Consultants
Homestake .... Homestake Mining Company

HPGR .... High-Pressure Grinding Rolls
HPGR-DP .... High-Pressure Grinding Rolls – Double Pass
HPGR-SP .... High-Pressure Grinding Rolls – Single Pass
Hunter Mining .... Hunter Mining Laboratory
HW .... Hanging Wall
ICP .... Inductively Coupled Plasma
ICP-AES .... Inductively Coupled Plasma Atomic Emission Spectroscopy
ID^2 .... Inverse Distance Squared
ID^3 .... Inverse Distance Cubed
IEC .... International Electrotechnical Commission
ILS .... Intermediate Leach Solution
IP .... Induced Polarization
IRR .... Internal Rate of Return
ISO .... International Organization for Standardization
Kennecott .... Kennecott Copper Company
LBC .... Lower Bear Creek
LG .... Lerchs-Grossmann (algorithm)
LOM .... Life of Mine
Longchamps .... Fred de Longchamps & Sons
MACT .... Maximum Achievable Control Technology
McClelland .... McClelland Laboratories, Inc.
MDA .... Mine Development Associates
Minproc .... Minproc Engineers Inc.
Miramar .... Miramar Mining Corp.
MOPTC .... Mercury Operating Permit to Construct
MPDI .... Mineral Property Development, Inc.
NaCN .... Sodium Cyanide
NAD .... North American Datum
NDEP .... Nevada Division of Environmental Protection
NDWR .... Nevada Division of Water Resources
NEPA .... National Environmental Policy Act
Newcrest .... Newcrest Resources Inc.
NI 43-101 .... National Instrument 43-101
NN .... Nearest Neighbor
NPV .... Net Present Value
NRP .... Nevada Reclamation Permit
NSR .... Net Smelter Return
OK .... Ordinary Kriging
OPEX .... Operating Cost Estimate
OPTC .... Operating Permit to Construct
Oretest .... Oretest Pty Ltd.
PEA .... Preliminary Economic Assessment
PbO .... Lead Oxide
Pegasus .... Pegasus Gold Corp.
Placer Dome .... Placer Dome U.S. Inc.
Plan .... Plan of Operations
POD .... Plan of Development
the Project .... Talapoosa Project
the Property .... Talapoosa Property
PMET .... Pittsburgh Mineral and Environmental Technology Inc.
PV .... Present Value
QA/QC .... Quality Assurance / Quality Control
QP .... Qualified Person

RC ....Reverse Circulation
RCE .... Reclamation Cost Estimate
REMCO .... Rock Engineered Machinery Co. Inc.
RF .... Revenue Factor
ROW .... Right-of-Way
SEM-EDX ....Scanning Electron Microscopy – Energy Dispersive X-Ray Spectroscopy
SAG ....Semi-Autogenous Grinding
SIBX ....Sodium Isobutyl Xanthate
Sierra Denali Minerals ....Sierra Denali Minerals Inc.
SRM ....Standard Reference Material
Summit Valley ....Summit Valley Equipment & Engineering Inc.
SUP .... Special Use Permit
TMI ....Talapoosa Mining Inc.
Union Assay .... Union Assay Office
UBC ....Upper Bear Creek
USGS ....US Geological Survey
UTM ....Universal Transverse Mercator
VLF ....Very Low Frequency
VSI ....Vertical Shaft Impact Crusher
Westec ....Welsh Engineering Science and Technology
WPCP .... Water Pollution Control Permit
WSP... WSP Canada Inc.

1 SUMMARY

The Talapoosa Project (the Project or the Property) is located in the Talapoosa mining district approximately 30 miles east of Reno in northwestern Nevada.

WSP Canada Inc. (WSP) in conjunction with Mineral Property Development, Inc. (MPDI), McClelland Laboratories, Inc. (McClelland), DOWL LLC. (DOWL), and Enviroscientists, Inc. (ES), has prepared this technical report on the Project at the request of Timberline Resources Corporation. (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 is April 27, 2015 and the effective date of the resource estimate is current as of March 24, 2015.

1.1 LOCATION AND PROPERTY DESCRIPTION

The Property is located in northwestern Nevada 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.

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 Capital US Inc. is a corporation incorporated under the laws of Nevada, USA and a wholly-owned subsidiary of Gunpoint Exploration Ltd. (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. On March 12, 2015, Timberline acquired the option to purchase 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.

The Talapoosa Formation is a sequence of vesicular basalt, felsic ash-flow tuffs, which host hydrothermal eruption breccias associated with epithermal mineralization along the Appaloosa structure.

The Stagecoach Hills Volcanics host all of the known mineralization in the district and overlay the Pyramid Sequence. The Stagecoach Hills Volcanics consist of dacitic tuff, tuff breccia, flows, lava dome carapace debris, and post-volcanic dacite porphyry sills or dykes.

Coal Valley Formation is a mixture of sand, silt and clay derived from pyroclastic volcanic rocks and unconformably overlays the Stagecoach Hills Volcanics.

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

The East Hill and Dyke Adit zones occur east and west of the Bear Creek and Main zones areas, respectively. Mineralization within the Main zone, Dyke Adit and East Hill shows similarities in appearance and texture to that of the Bear Creek FW zone. Within approximately 100 ft. of the ground surface each zone is oxidized and generally contains no or very minor sulphide minerals.

Based on the distinctions within the zones and the near-surface oxidation of each, mineralized materials at Talapoosa are categorized in three types including: (1) oxidized, (2) HW type, and (3) FW type. The HW and FW-types represent unoxidized mineralization.

1.3 DRILLING

In 2011, Gunpoint completed seven PQ diamond drillholes totaling 5,302.5 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.

Drill 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

A substantial amount of metallurgical test work has been performed over the years on various materials from the Talapoosa deposit. Results from these tests are varied but indicate that the resource materials contained within the PEA pit shell are amenable to conventional cyanide heap leach extraction processes.

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

Available metallurgical test work results suggest that conventional heap leaching at a relatively fine crush size is the best approach for processing the materials from Talapoosa. Results from the recent and historic programs provided the basis from which the following estimated heap leach recoveries were used in the development of the PEA. These recovery estimates of three mineralization types were made assuming an agglomerated nominal 1.7 mm (10 mesh) crushed product, generated using high pressure grinding rolls (HPGR) (Table 1.1)

Table 1.1 - Leach Recoveries

	Au Recovery	Ag Recovery
Oxidized (HW + FW Type)	77%	47%
Hanging Wall (HW) Type (unoxidized)	65%	60%
Footwall (FW) Type (unoxidized)	59%	45%

Results from the 2015 test program generally support the earlier heap leach test work. In column tests, both HW and FW-type mineralization within the Bear Creek zones were shown to respond moderately well to simulated heap leach treatment at relatively fine (6.3 mm or finer) feed sizes. Gold recoveries tended to be lower for the Bear Creek FW-type unoxidized material than for the Bear Creek HW-type unoxidized material. Optimization of agglomerating conditions will be required to ensure that the finely crushed material will maintain acceptable permeability during commercial heap leach operations.

Mineralogical examination has shown that gold is frequently present in solid solution with silver as electrum. Metallurgical test work at Talapoosa indicates 50% extraction within 60 days, and 75% extraction within 120 days. These leach cycle times were considered during PEA process development.

All material types tested are sensitive to feed size with respect to gold and silver recovery. In the case of the unoxidized Bear Creek zone HW-type and FW-type material, a portion of the contained gold and silver was also shown to be locked in sulphide mineral grains, or locked in silica. Relatively fine crushing was shown to significantly increase gold and silver recovery.

Testing included evaluation of various feed size reduction equipment. Enhanced gold and silver particle liberation through the use of HPGR resulted in increased recoveries. However, it is unclear whether improvements to recovery were achieved due to the generation of finer particles or from micro-fracturing within particles induced by the HPGR process.

In limited testing, the recovery of gold and silver by flotation was generally high for the unoxidized (sulphidic) Bear Creek FW and HW-type material, but was lower for most of the oxidized Main zone material. Minimal test work or investigation has been conducted to further evaluate processing of flotation concentrates for recovery of contained gold and silver.

1.5 RESOURCE ESTIMATION

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 neighbor (NN) used for validation.

Table 1.2 is a summary of the resource estimation at Talapoosa.

Table 1.2 - Talapoosa Geological Resource Summary

Summary	Cut-Off (oz/st)	Tons (st)	Au (oz/st)	Ag (oz/st)	Tonnes (t)	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 PEA CONCLUSIONS

The PEA Base Case considers a low capital cost open pit mine and heap leach facility, with an annual processing rate of 3.8 million tons of PEA in-pit resource material placed on the leach pad.

The PEA demonstrates that the Project has economic merit. The Mineral Resources are of sufficient quantity and quality such that additional investigation at more advanced levels of engineering study (pre-feasibility or feasibility study) is warranted. Economic results are positive for the production scenario considered in the PEA, based upon the stated assumptions.

Table 1.3 summarizes the open pit resources determined by this PEA.

Table 1.3 Summary of Open Pit Resource

Category	Tonnage (k st)	Au Grade (oz/st)	Ag Grade (oz/st)
Total Material Mined	102,444		
Waste Rock Mined	61,023		
Heap Leach Feed by Zone, by Resource Category			
Bear Creek FW – Measured & Indicated			
Oxide	1,976	0.014	0.278
FW Type	11,502	0.023	0.373
Bear Creek FW - Inferred			
Oxide	14	0.009	0.249
FW Type	26	0.014	0.358
Bear Creek HW – Measured & Indicated			
Oxide	1,343	0.014	0.202
HW Type	9,587	0.028	0.441
Bear Creek HW - Inferred			
Oxide	528	0.012	0.192
HW Type	1,889	0.016	0.256
Main Zone – Measured & Indicated			
Oxide	4,228	0.018	0.289
HW Type	6,159	0.019	0.257
Main Zone – Inferred			
Oxide	72	0.015	0.257
HW Type	110	0.017	0.283
Dyke Adit – Measured & Indicated			
Oxide	1,972	0.025	0.491
HW Type	636	0.028	0.344

Category	Tonnage (k st)	Au Grade (oz/st)	Ag Grade (oz/st)
Dyke Adit – Inferred			
Oxide	151	0.007	0.042
HW Type	42	0.008	0.060
East Hill – Inferred			
Oxide	1,159	0.018	0.048
HW Type	27	0.009	0.060

Notes:

- Within Pit Shell #49, RF0.83.
- The following cut-off grades have been used in the evaluation:
  - Oxidized: 0.006 oz/t Au, 0.720 oz/t Ag;
  - HW Type: 0.007 oz/t Au, 0.564 oz/t Ag;
  - FW Type: 0.008 oz/t Au, 0.752 oz/t Ag.
- Mining Loss & Dilution at 96% and 4% respectively.
- $1150/oz Au and $16/oz Ag metal prices.

Table 1.4 summarizes the key project results.

Table 1.4 - Key Project Results

Category	Units	Value
Mining		
Mine Life	years	10.8
Total Material Mined	M st	102.4
Waste Material	M st	61.0
Heap Leach Feed	M st	41.4
Average Grade	oz/st	0.022 Au 0.339 Ag
Strip Ratio		1.47
Processing		
Annual Production	M st/y	3.8
Average Recovery	%	66 Au 52 Ag
Recovered Gold	k oz	593
Recovered Silver	k oz	7,365
Economic		
Gold Price	$/oz	1150
Silver Price	$/oz	16
LOM Total Operating Costs (Mining + Processing + GA + Reclamation)	M $	439.6

Category	Units	Value
LOM Total Cash Costs (Operating + Refining Charges , Royalties, Net Proceeds Tax)	M $	470.1
Total Capital Costs and Contingencies	M $	51.9
Pre-Tax NPV @ 5% Discount Rate	M $	184
Pre-Tax IRR	%	48
Pre-tax Payback Period (from Start of Production)	years	0.9
After-Tax NPV @ 5% Discount Rate	M $	136
After-Tax IRR	%	39
After-tax Payback Period (from Start of Production)	years	3.1

Additional highlights from the PEA are as follows:

- About 90% of the open pit production tonnage is classified as Measured or Indicated Mineral Resources, with the remaining 10% classified as Inferred.
- Heap leach feed consists of 28% oxidized material, 45% hanging wall type material and 28% footwall type material.
- Initial capital cost of $51.2 million that includes $5.7 million in contingency costs.
- Pre-tax payback occurs within the first year of production.
- Life of Mine (LOM) operating cost (net of silver credits) of $543 per ounce of gold (includes silver as a credit. Costs included are: operating costs, which include mining, processing, general and administrative, and reclamation. Costs not included are the following costs: Royalties and refining charges, and Nevada net proceeds tax, capital costs and corporate income tax).
- "All-in" cost (net of silver credits) of $682 per ounce of gold (includes silver as a credit. Costs included are: operating costs, which include mining, processing general and administrative, and reclamation; Royalties and refining charges; Nevada net proceeds tax; and Capital costs (initial and sustaining). Costs not included are corporate income tax).
- The project economics are most sensitive to variations in gold price and gold recovery.

1.7 OPEN PIT MINING

WSP reviewed the Project at the level of a PEA. WSP cautions that the mine plan in this study included Inferred Mineral Resources.

Under NI 43-101 Part 2, Section 2.3(3) and Companion Policy 43-101CP, Part 2 Section 2.3(3), the use of inferred mineral resources is allowed in a PEA in order to inform investors of the potential of the Property.

The proposed operation considered in this PEA includes two discrete open pits. The LOM plan delivers 41.4M tons of mineral resource to the heap leach facility, at overall average grades of 0.022 oz/st Au and 0.339 oz/st Ag, with a strip ratio of 1.47. The mine life for the 3.8 M st/y scenario is 10.8 years.

The LOM plan generates 61.0 M tons of waste rock. The waste rock will be stored in waste rock storage areas located in proximity to the open pits in order to minimize waste haulage cycle times.

The PEA considers a contract miner scenario for the mining operations. The mine will be operated on a 24 hours per day, 365 days per year basis.

1.8 RECOVERY METHODS

DOWL, in conjunction with MPDI, developed the process scenario considered in the PEA. Heap leaching of the PEA resources in conjunction with Merrill Crowe processing of the pregnant solution was preferentially selected as the appropriate metallurgical extraction and recovery scenario given the grade of the material and the relatively high ratio of recovered gold to silver.

Mineralized material of economic value is crushed to a nominal size of 10 mesh. The final stage of crushing is by high pressure grinding roll (HPGR). HPGR crushing is effective at increasing fracture surfaces within a crushed product.

The process scenario calls for the crushed material to be drum agglomerated and conveyed to a radial stacker by which it is placed in 20 ft. lifts on the heap.

Drip emitters will be used to distribute leach solution to the top of the lift at a rate of 0.004 gpm ft^2. The heap and solution pond is designed to allow for 60 days of primary leaching and 60 days of intermediate solution leaching (preg building).

Pregnant solution from the heap is pumped to a Merrill Crowe processing facility wherein the gold and silver are recovered. The filter cake is treated to separate any mercury that may have entered the leach circuit. Once treated, the filter cake is transferred to smelting furnaces. Fluxes are added and the material is smelted producing doré.

1.9 PROJECT INFRASTRUCTURE

No permanent project infrastructure exists on the site. Planned infrastructure includes power, water supply, access roads, plant and admin area and heap leach pad and ponds.

Power and water will be brought in from the southern portion of the property. A well will provide water for mining operations and plant and heap leach pad operations. It is estimated that the Project will need approximately 500 gpm water makeup.

Access roads will be constructed based on the largest piece of equipment that would use the roadway during operations or construction.

The plant and admin areas will include the Merrill Crowe processing plant, truck shop facilities, crushers and admin offices.

The heap leach pad will consist of a lined pad area and the associated ponds. The pregnant and intermediate ponds will be double lined with leak detection and the storm pond will be single lined. The ponds will be sized to contain the drain down of the heap and the 25yr-24hr-precipitation event without overtopping.

1.10 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project has current permits in place for mining and exploration activities. A mining Plan of Operations (Plan) and Nevada Reclamation Permit (NRP) was approved for open pit and heap leaching operations in 1996. The NRP is active for the life of the Project.

As part of the existing NRP and approved Plan, an updated reclamation cost estimate (RCE) must be completed and the work coordinated with the BLM and Bureau of Mining Regulation and Reclamation (BMRR). A decision must be issued by the BLM before operations can commence. This type of decision is not a Federal Action and is therefore not subject to further National Environmental Policy Act (NEPA) review.

An Environmental Impact Statement (EIS) was also completed for the proposed mining activities in 1996 and a positive Record of Decision granted in 1997 in support of the Plan approval. No mining activities were initiated and no associated reclamation liability currently exists.

Exploration activities are currently authorized through an approved BLM Plan and a NRP held by American Gold Capital US, Inc. The reclamation bond for the activities has been posted with the BLM by American Gold Capital US, Inc. and remains active.

With authorizations that are currently in place for mining activities, any new required permits will be obtained in accordance with the construction, operation and reclamation activities described in the approved Plan and NRP to maximize use of existing data and permit time frames. To operate the Project as outlined in the approved Plan and NRP, a water pollution control permit (WPCP), air quality operating permit, and a mercury-operating permit to construct will be required. In addition, water rights will need to be obtained from Nevada Division of Water Resources (NDWR) as well as a Lyon County special use permit (SUP); and other ministerial permits, plans, and notifications.

Power line, substation, and water line facilities outside of the mine Plan on BLM-administered land will require rights-of-way (ROW) authorizations. The remaining portion of the water line outside of the Plan area on non-public land will require conformance with applicable Lyon County requirements.

Some permit applications, such as the WPCP, will require the compilation or acquisition of new data and creation of planning documents such as engineering designs and geochemical characterization prior to submittal. NV Energy will perform the power line engineering and ROW permitting. Each of these items adds a level of uncertainty to the permitting timeframes.

1.11 CAPITAL AND OPERATING COSTS

1.11.1 CAPITAL COSTS

The initial capital costs are estimated at $51.2 million, including $7.8 million for indirect costs such as Engineering, Procurement and Construction Management (EPCM) fees and construction indirect costs; $5.7 million in contingencies; and a $2 million allowance for a reclamation bond. Direct capital costs include the costs for mine preparation, processing infrastructure, and site infrastructure. Sustaining capital is estimated at $2.7 million, which includes contingencies but excludes the expected return of the reclamation bond at the end of the mine life, and is expected to be low due to the mine contractor scenario.

1.11.2 OPERATING COSTS

Operating costs for the entire LOM period is estimated to $439.6 million. Cash Costs are estimated to be $470.1 million for the LOM period and include operating costs (including reclamation), royalties, refining charges, and Nevada Net Proceeds Tax.

1.12 RECOMMENDATIONS

Additional exploration and delineation expenditures are warranted to improve the viability of the Project. It is recommended that Timberline undertake a two-phased program that will develop, during Phase 1, a better understanding of the metallurgy of the various mineralized material types in and around the open pit shell and complete in-fill drilling at East Hill and Dyke Adit to convert material from Inferred to Indicated resource categories.

Phase 1A would focus on expanding and upgrading the resources as well as collection of material suitable for additional metallurgical testing of the mineralized material horizons. Phase 1B would focus on the metallurgical test program, which would utilize the samples collected in Phase 1A in addition to samples already available. The estimated cost of Phase 1 would be US$1.924 million.

Phase 2 would focus on development of a Pre-Feasibility Study based substantially on results of Phase 1. The estimated cost of Phase 2 would be US$1.665 million.

2 INTRODUCTION

Timberline entered an Option Agreement (Agreement) on March 12, 2015 to acquire 100% of the Talapoosa Project ("the Project") from Gunpoint Exploration Ltd. ("Gunpoint"). WSP Canada Inc. ("WSP") was commissioned by Timberline to complete a Preliminary Economic Assessment ("PEA") and update the technical report on the Project.

WSP Canada Inc. (WSP) in conjunction with Mineral Property Development, Inc. (MPDI), McClelland Laboratories, Inc. (McClelland), DOWL LLC. (DOWL), and Enviroscientists, Inc. (ES), has prepared this technical report in accordance with NI 43-101 Standards of Disclosure for Mineral Projects.

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 NYSE MKT and TSX Venture Exchange under the symbols TLR and TBR, respectively.

The effective date of this report is April 27, 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., of WSP visited the site from September 23 to 25, 2012 inclusive.
- Joanne Robinson, P. Eng., of WSP has not visited the site.
- Richard Jolk, P.E., of MPDI visited the site in January and March, 2015.
- Jack McPartland of McClelland Laboratories has not visited the site.
- Michael Henderson, P.E., of DOWL has not visited the site.
- Richard DeLong, of Enviroscientists 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.

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 Q1 2015 US dollars unless otherwise indicated.

All data sourced for this report are identified in Section 27 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.
- Joanne Robinson, P. Eng., relied upon: Ben Haberman, CPA, MBA, Tax Director at DeCoria, Maichel & Teague, who provided the tax portions of the economic analysis described in Section 22 of this PEA.
- Richard Jolk, P.E., relied upon Mr. Garth Colwell, Chief Estimator at Ledcor, Nevada, who, in conjunction with his team, provided capital and operating cost estimates for contract mining of the Talapoosa deposit as developed in this PEA; and relied upon Mr. Bruce Christiansen of Weir Minerals who, in conjunction with his team, provided capital and operating cost estimates for the crushing, agglomeration and conveying / stacking systems.

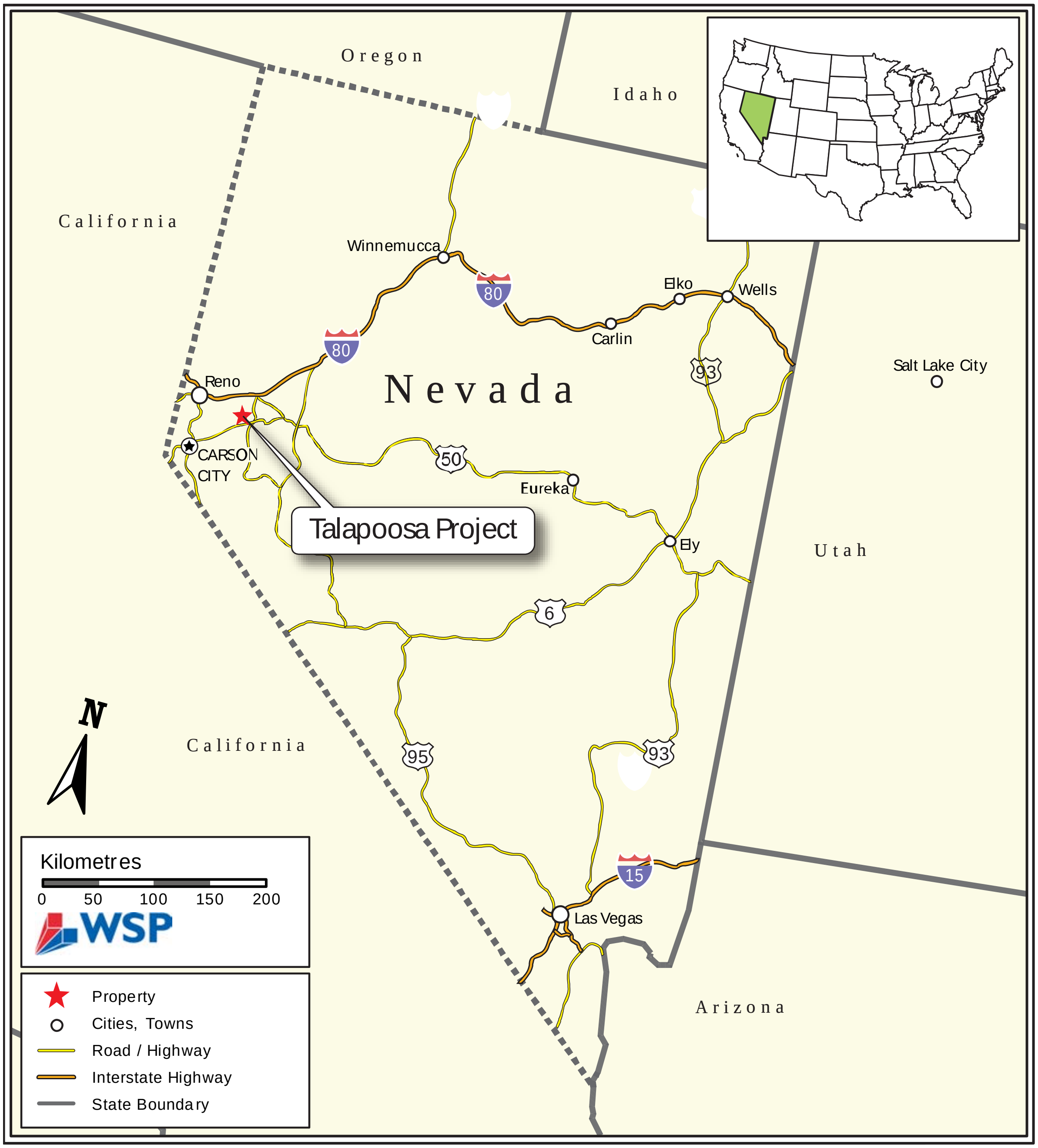
4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Project is located in the Talapoosa mining district in northwestern Nevada. The district lies in Lyon County approximately 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 lessee 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 require 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 Lyon 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 transactions:

- 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 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, T19N, R24E 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.

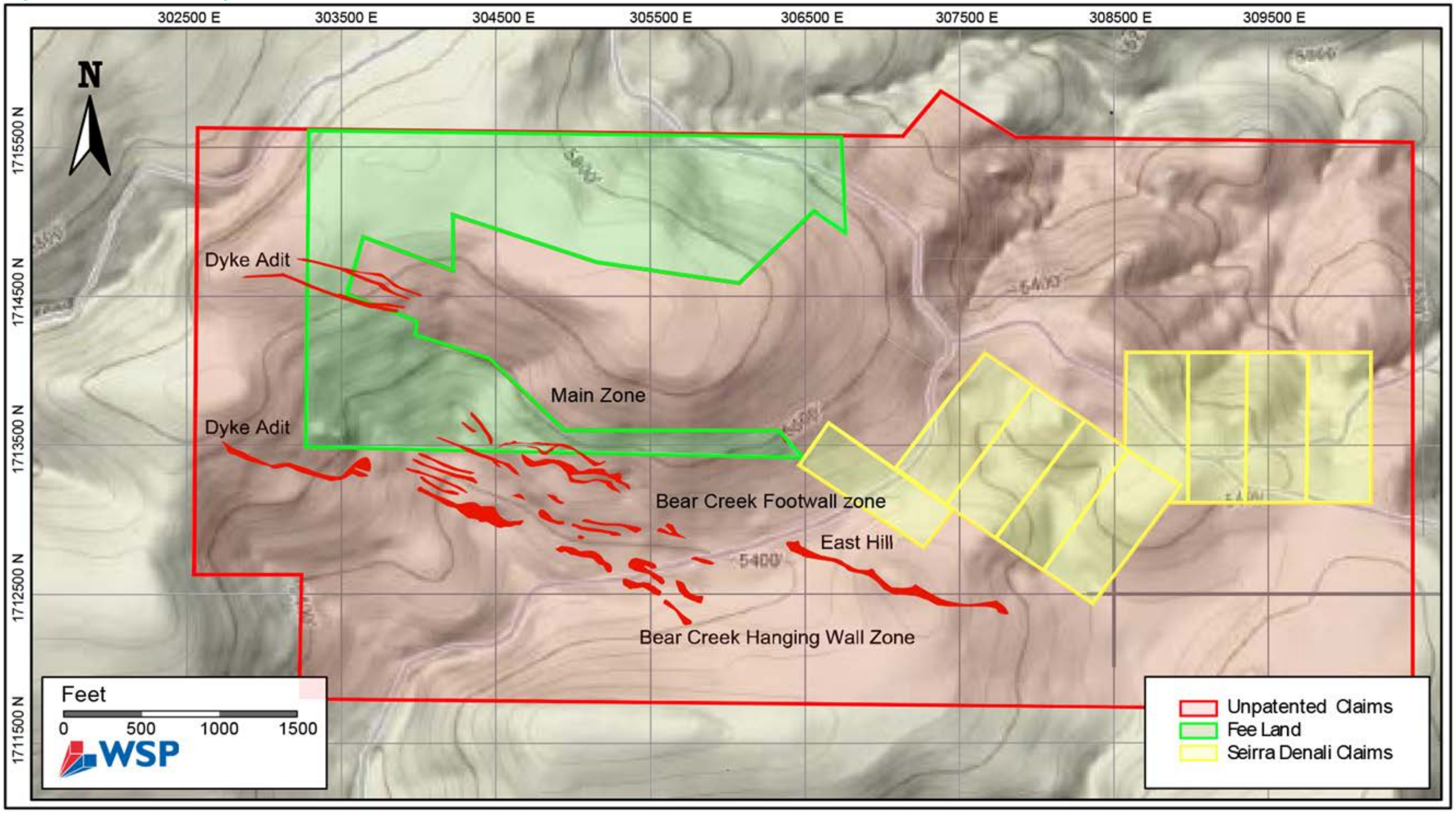
Table 4.1 – Claims Owned or Leased by American Gold within the Resource Area

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 12, 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 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 the Agreement, Timberline has also 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 that 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 September 1994, Talapoosa Mining Inc. (TMI) submitted a Plan of Operations (Plan) to the Carson City District Office of the Bureau of Land Management (BLM) for an open pit gold and silver mine. Proposed facilities included open pits, waste rock facilities, a heap leach facility, a water pipeline, and other ancillary facilities. The BLM issued a Decision on December 4, 1996 approving the Plan. The BLM completed an Environmental Impact Statement (EIS) as part of the Project review in October of 1996. An appeal filed by the Building and Construction Trades Council was dismissed in April 1998. A Reclamation Permit Application was submitted to the Nevada Division of Environmental Protection (NDEP) on October 3, 1996 and Reclamation Permit No. 0102 was issued for the Project on September 19, 1996. A reclamation bond cost was established for the Project, however a bond was not posted and no reclamation liability remains.

Exploration activities have been permitted through various actions with the BLM and NDEP since 1988. Talapoosa Mining Inc. filed an exploration Plan with the BLM in March of 1988 for 20 acres of disturbance. This Plan was approved in December of 1989 (NVN069733) and the case was closed in November of 1995 with no reclamation liability remaining.

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 (NVN89474) to the BLM, which included certain drill sites within the resource area. A reclamation bond in the amount of $15,000 was originally posted with the BLM and the case was ultimately closed in December 2012 with no reclamation liability remaining.

An Exploration Plan (NVN70006) was filed with the BLM by American Gold Capital US, Inc. in 1993 with an Environmental Assessment (EA) approved in 1994. Reclamation Permit 0070 was also issued for this project. Gunpoint Exploration US Ltd submitted an Interim Permit for Reclamation Application in September of 2011 to the NDEP. The NRP was revised to include the current and proposed exploration activities within the resource area.

In December of, 2011, American Gold requested bond release for re-vegetation and to re-categorize the acreage in the BMRR Permit 0070. The BLM and the NDEP conducted a site inspection December 21, 2011 and agreed to release the re-vegetation components in a 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

Permitting for the Project is discussed in Section 4.4 and Section 20. The Project has current permits in place for mining and exploration activities. A mining Plan was approved by the BLM with a NRP. Both remain active for the life of the Project. In 1996, a Water Pollution Control Permit was issued by NDEP for the Project, however this permit is no longer active, and will need to be re-applied for at the BMRR for approval.

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.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

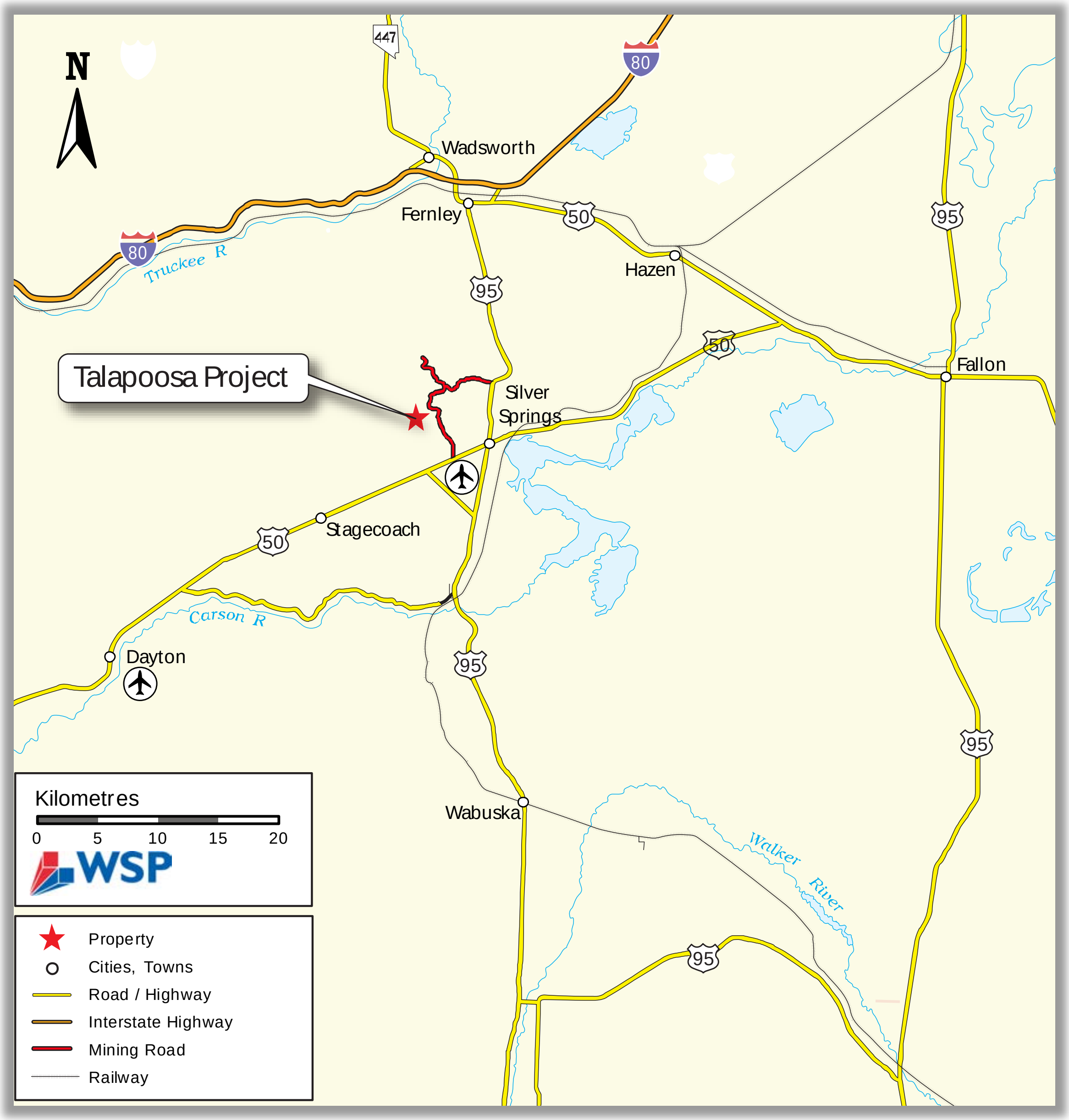
5.1 ACCESS

Year-round access to the Talapoosa district from Reno is via Interstate 80, east approximately 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 Talapoosa resource (Figure 5.1).

An alternate but 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 approximately 3 miles west to the resource area. This route is not recommended when road conditions are wet or muddy. The original Plan of Operations (Plan) called for a more direct alignment and improvement of this road, ultimately making it the main access to the Property.

Reno has an international airport with numerous regional flight schedules 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 approximately 13.4 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 50 in. per year (www.city-data.com).

Access to the Property is available year round if required.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The Project is located approximately 45 miles in road distance from Reno, whose metropolitan area has a population of approximately 225,000 (www.usa.com), and 30 miles in road distance from Nevada's capital, Carson City, with a population of approximately 55,000 (www.usa.com). The Reno / Sparks area is the closest major metropolitan area. Other population centers that are in the vicinity of the project are as follows:

- Silver Springs – Located approximately 4 miles southeast of the Project with a population of approximately 5,000.
- Fernley – Located approximately 18 miles northeast of the Project with a population of approximately 19,000.
- Yerington – Located approximately 30 miles south of the Project with a population of approximately 3,000.

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

A light-duty commercial power line passes through the Project running from the southern end of the Property and up the same canyon within which Ruby Road is located. Upgrades to the electric infrastructure will be required to advance the Project. It is anticipated that a new power line will be constructed along the same alignment as the exiting power line and will be extended approximately 2.5 miles to the plant area.

Water supply for the project will be leased from groundwater owners in the Silver Springs valley. Previous engineering studies have identified suitable areas for plant and ancillary facilities and also heap leach pad 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 approximately 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; however, this water may be occurring in a perched aquifer system implying that the rechargeable water table in the area is at a lower elevation.

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. Archaeological 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.
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.

Year	Company	Activity
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
1998	5	3,892	1
1992	16	7,966	3
1992 and 1993	23	9,545	3
1993	53	31,372	11

Description	Number of Holes	Feet	Percent
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. WSP 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

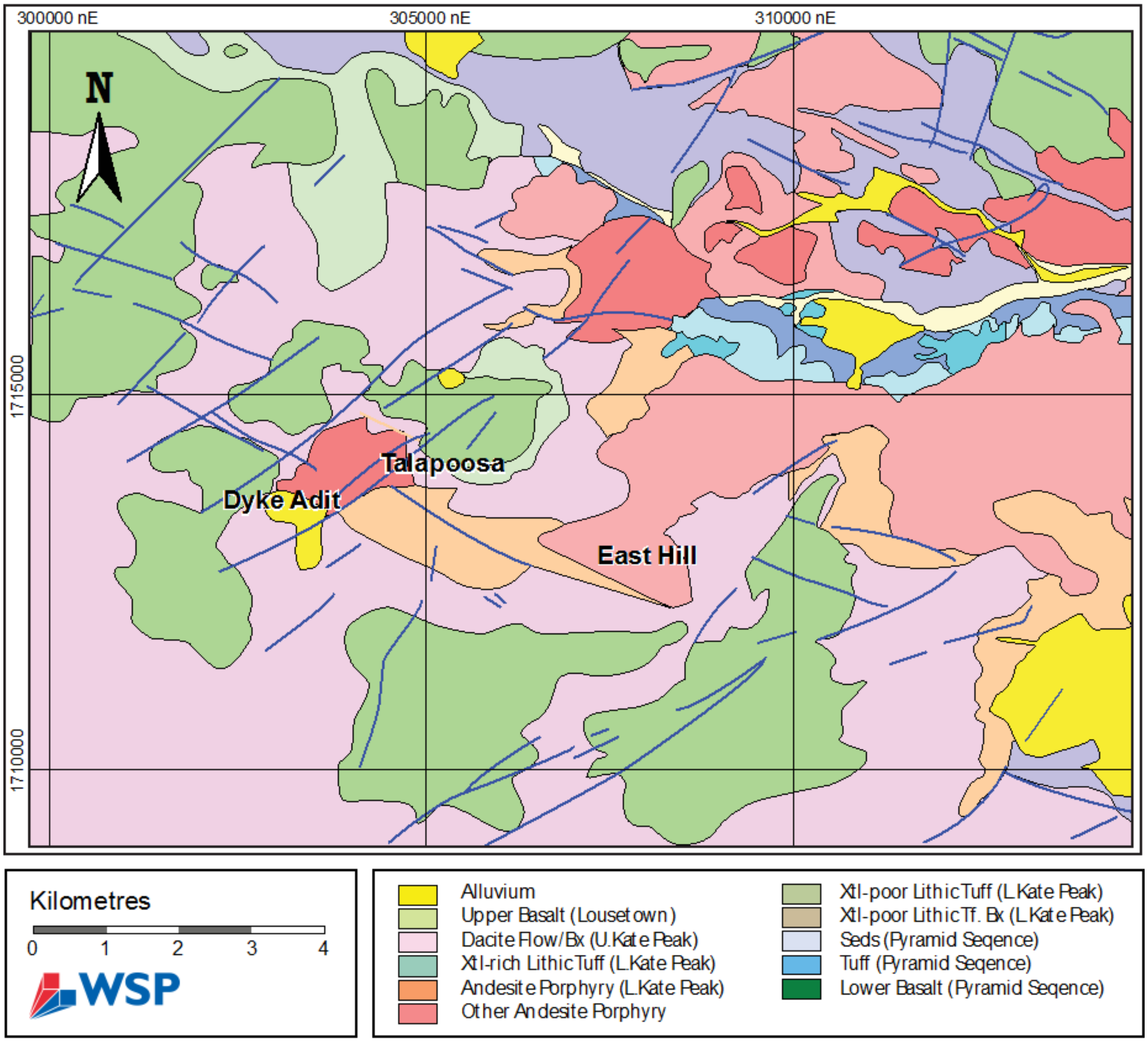
The Project lies in the western Basin and Range Province, a structural province of generally north-trending mountain ranges and intervening valleys formed by regional extension during Cenozoic Tertiary time. The Sierra Nevada on the California-Nevada border forms the western margin of the province. The 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).

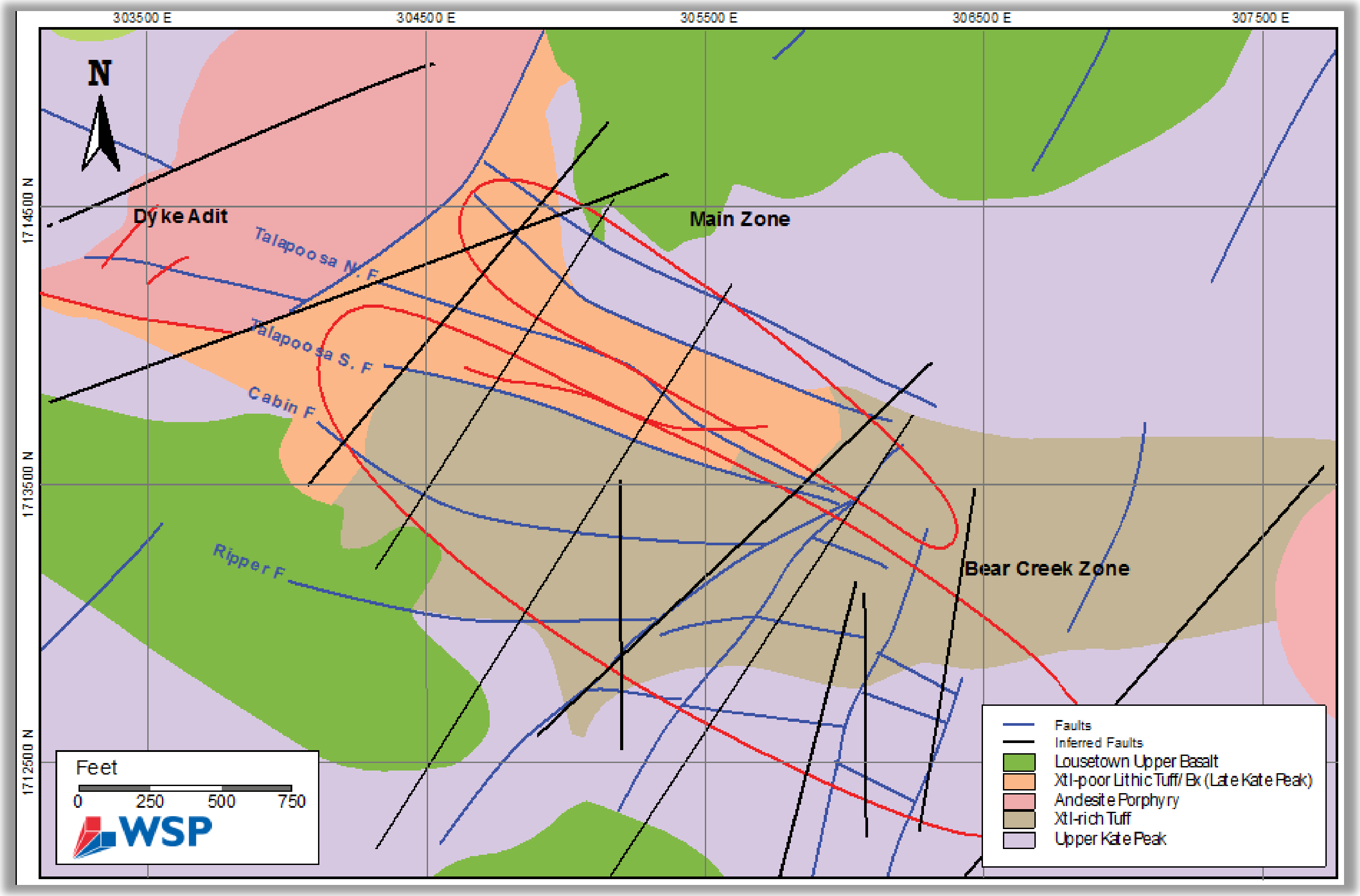
The Pyramid Sequence is the base of the geological package on the Project. It is a sequence of vesicular basalt, felsic ash-flow tuffs and hydrothermal eruption breccias associated with epithermal mineralization along the Appaloosa structure.

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 HW, Bear Creek FW 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 HW zone bounded by the Ripper Fault to south and the Cabin Fault to north. The HW 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 FW zone, bounded by the Cabin Fault to south and the Talapoosa (South) Fault to the north. The FW 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 HW zone at Bear Creek.

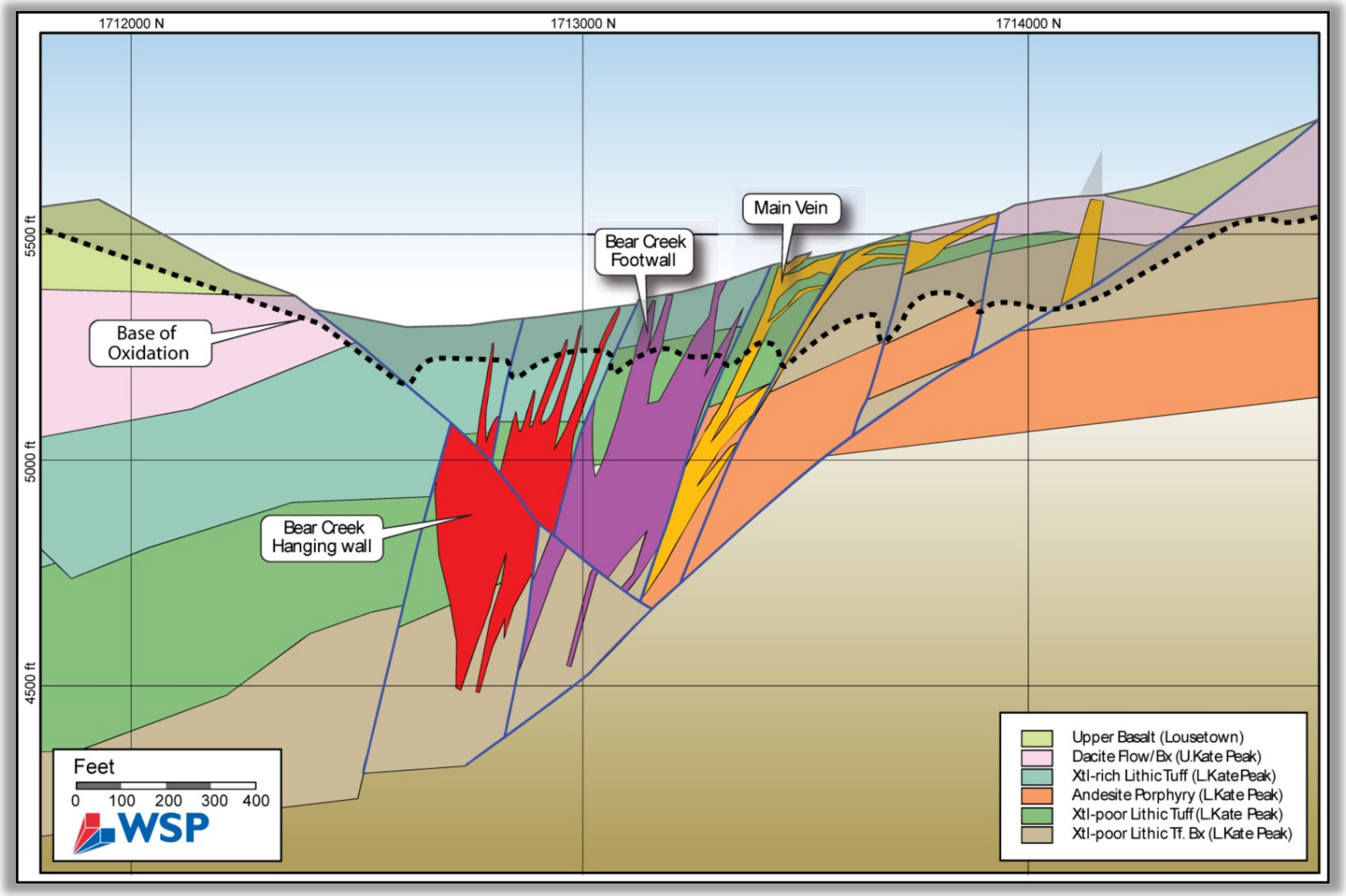
All of the domains are affected by a weathering profile extending down to as much as100 feet that results in an oxidized capping layer.

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 about 70° south, for the HW and FW zones at Bear Creek and for the eastern-most portion of the Main zone.
- Shallowly-dipping veins, at about 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).

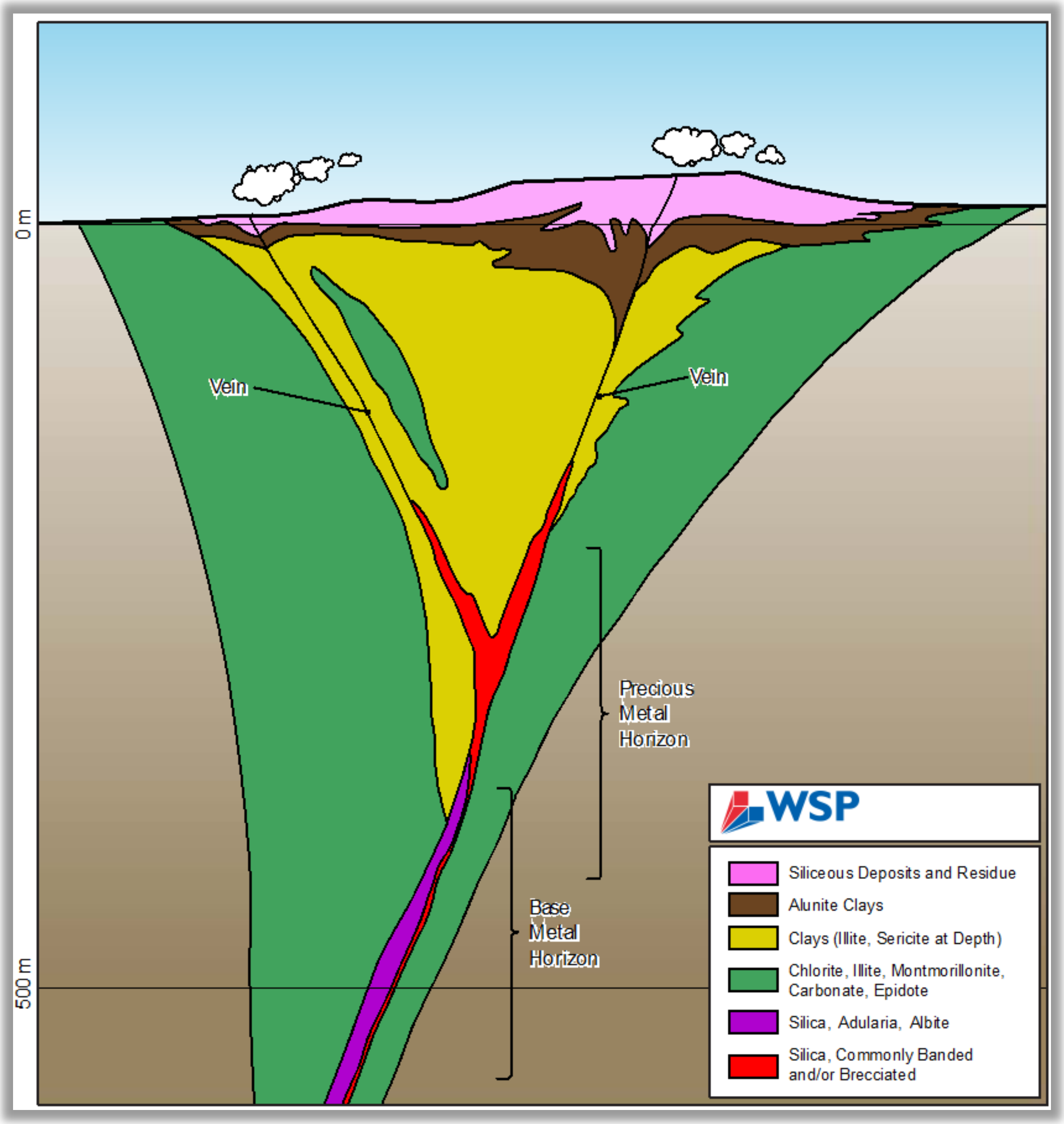
Emplacement of mineralization is generally restricted to within 1 km of the paleosurface (Panteleyev 1996). Veins typically have strike lengths in the range of hundreds to thousands of metres; productive vertical extent is seldom more than a few hundred metres. Vein widths vary from a few centimetres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

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 drilled 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 approximately 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 3,683 ft., 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 2,270 ft. 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 9,545 ft. of RC drilling in 23 holes and 2,267 ft. 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 27,072 ft. of RC drilling and 1,848 ft. 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 84,940.8 ft. 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 3,892.2 ft. (NCTAL-1 – NCTAL-5). Boart Longyear was the drilling contractor. Hole NCTAL-5 was reduced to BQ (1.43 in) from 652 to 901 ft. 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 validate the company's geological and structural reinterpretation of Talapoosa, determine the significance of the nugget effect on historic drill data, and to confirm the re-interpretation of the Bear Creek mineralization as two separate and steeply dipping vein zones, a HW-Type and FW-Type. 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.

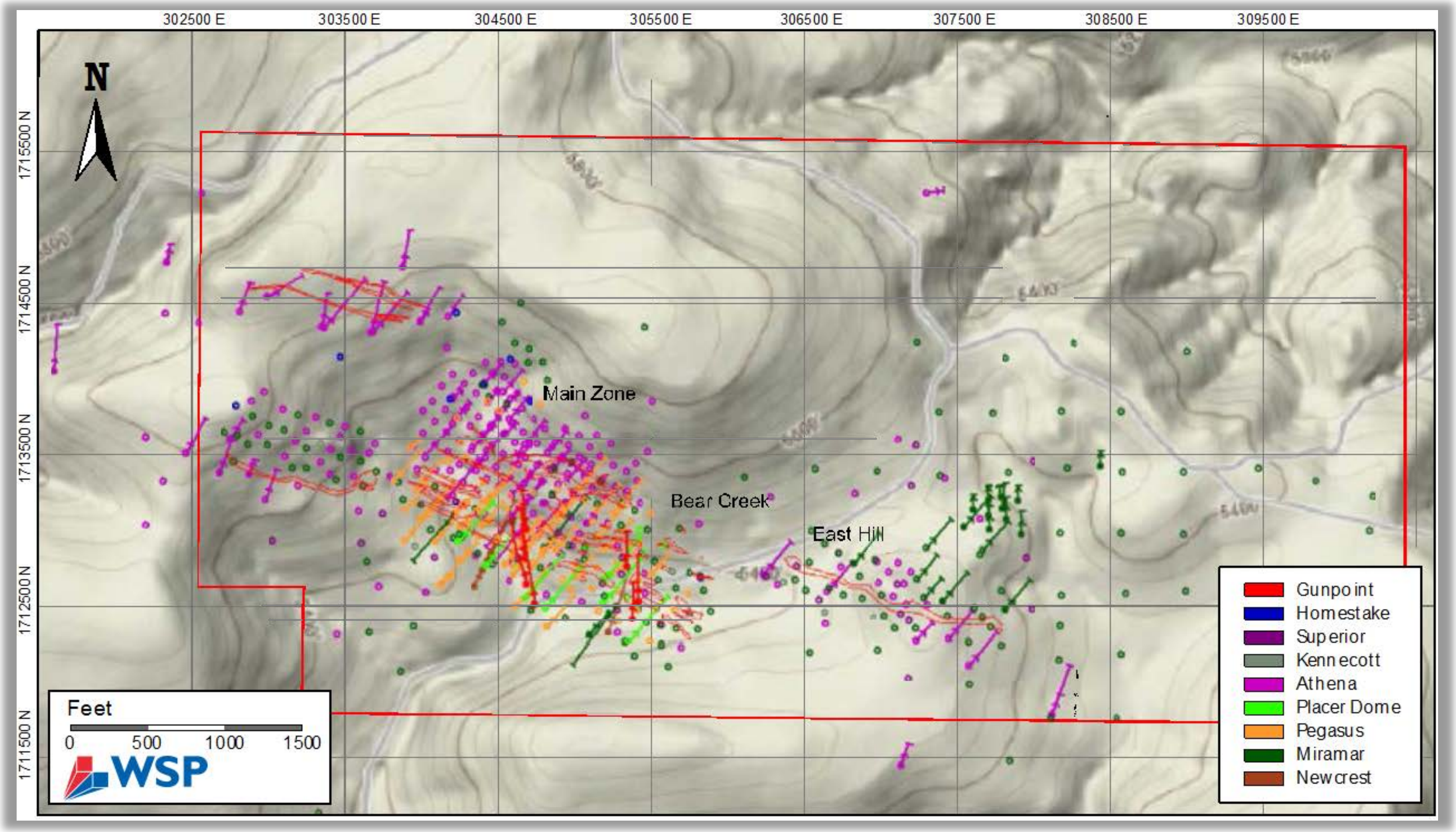
Table 10.2 – Gunpoint Drilling Collar Summary

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



Timberline Drilling Inc. of Elko, Nevada, completed the 2011 drilling program. 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 5 to 8 ft., 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 the 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

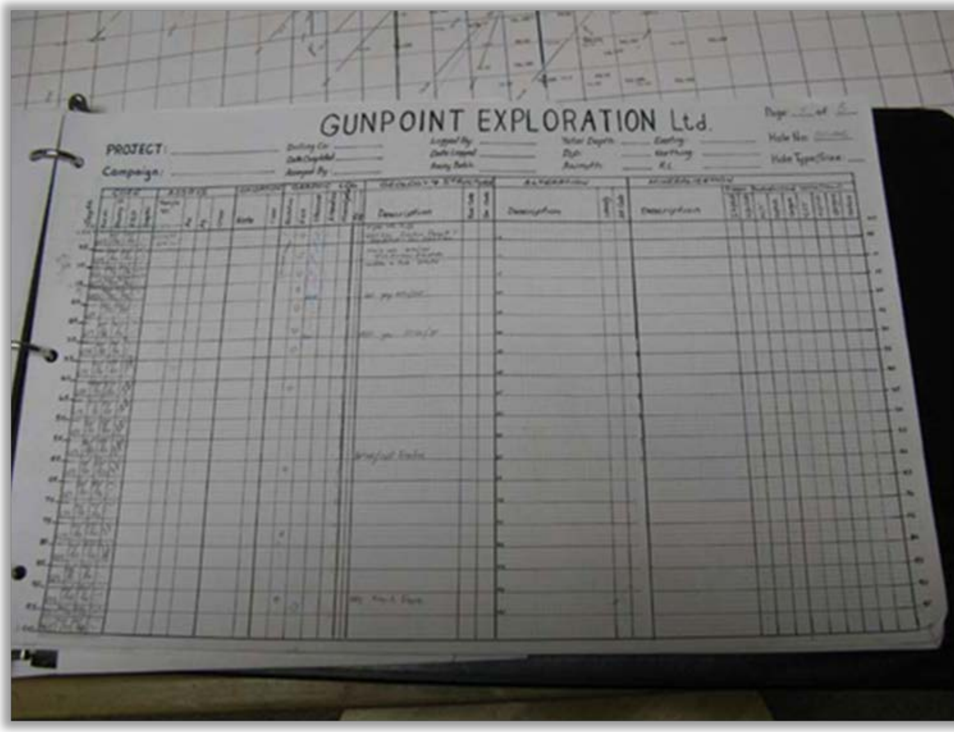


Figure 10.6 – Core Orientation Stand

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 between 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. The drill contractor collected half splits 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 regarding 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 regarding 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 that had detection limits for gold and silver of 0.003 and 0.03 oz/ton respectively.

KENNECOTT

There are no records regarding 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 regarding 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 were 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 regarding 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. Approximately 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 regarding 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 test work 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 were 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.
- 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 approximately 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

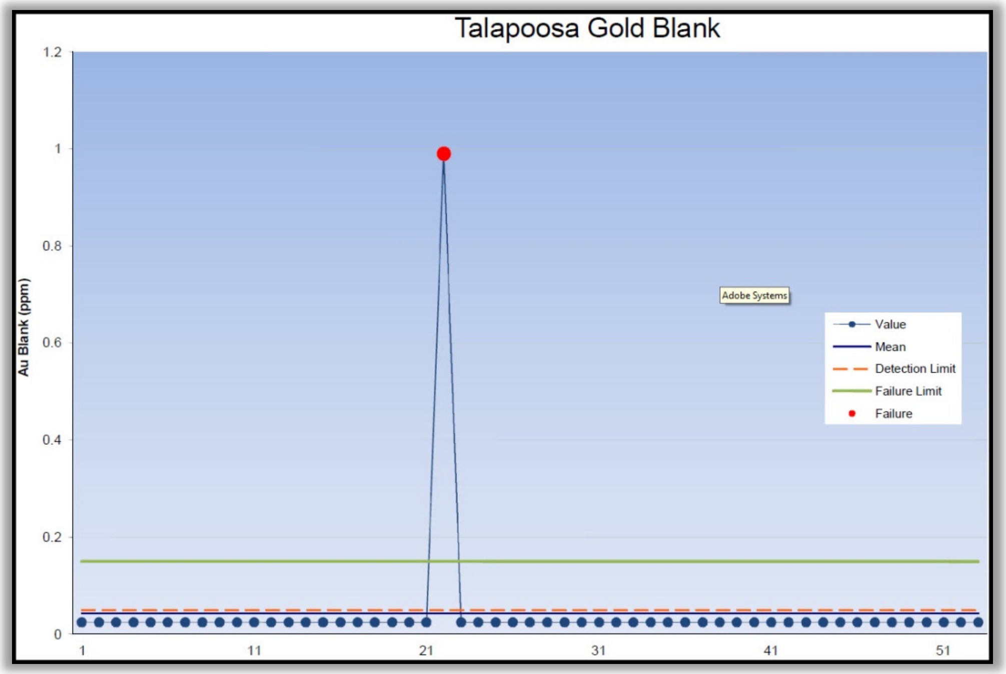
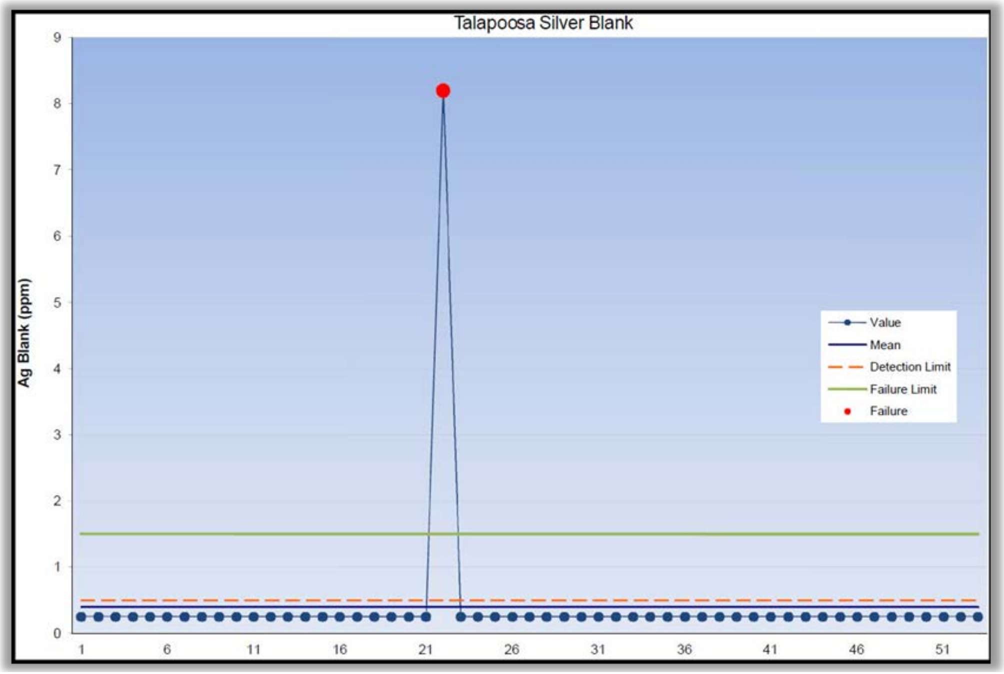


Figure 11.4 – Silver Blank Chart



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 Duplicates

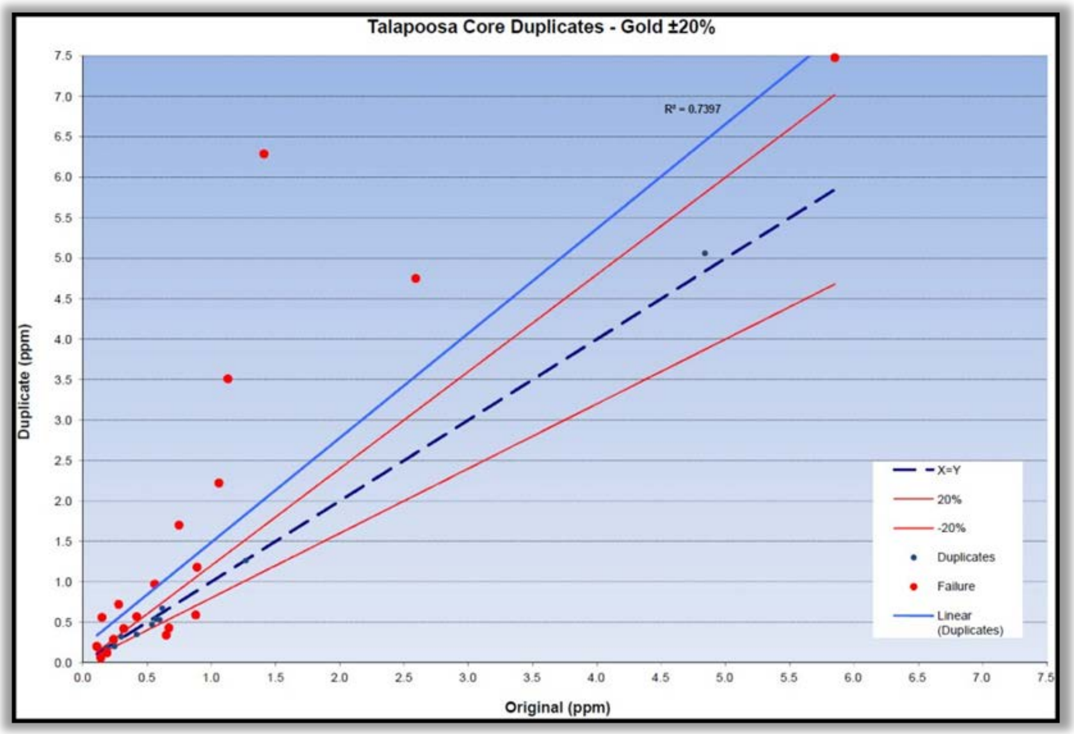
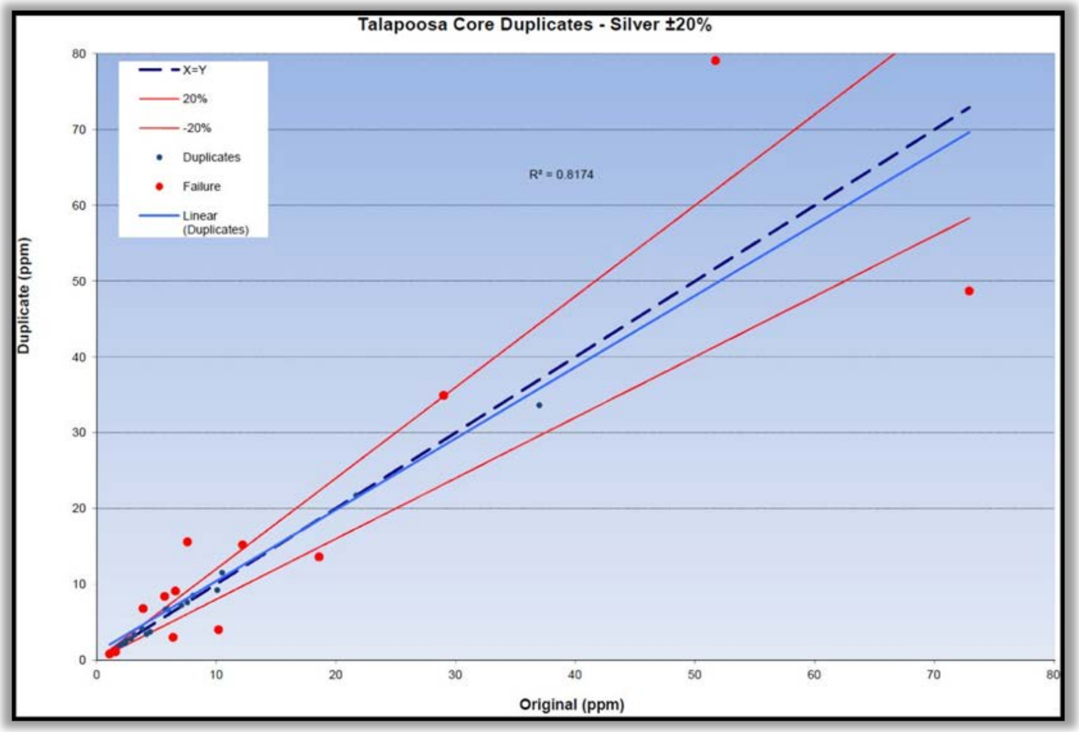


Figure 11.6 plots the silver assay duplicates.

Figure 11.6 – Silver Core Duplicates



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 dataset 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

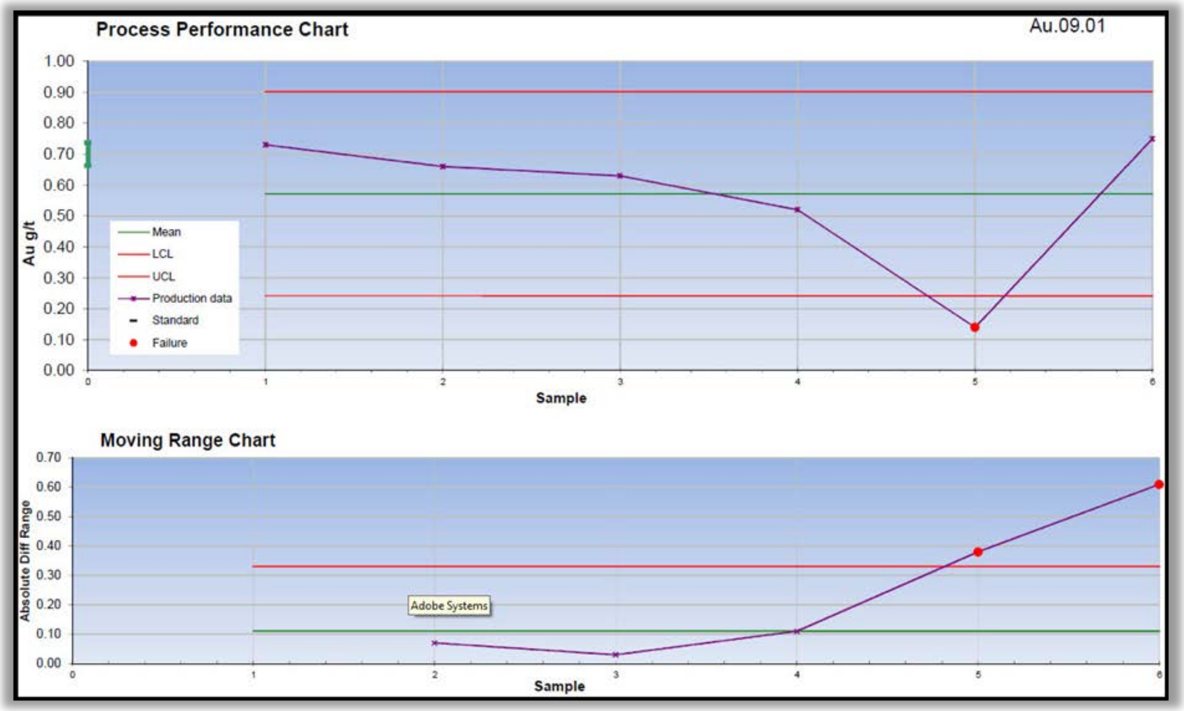


Figure 11.8 – SRM Au.09.01 – Silver Plot

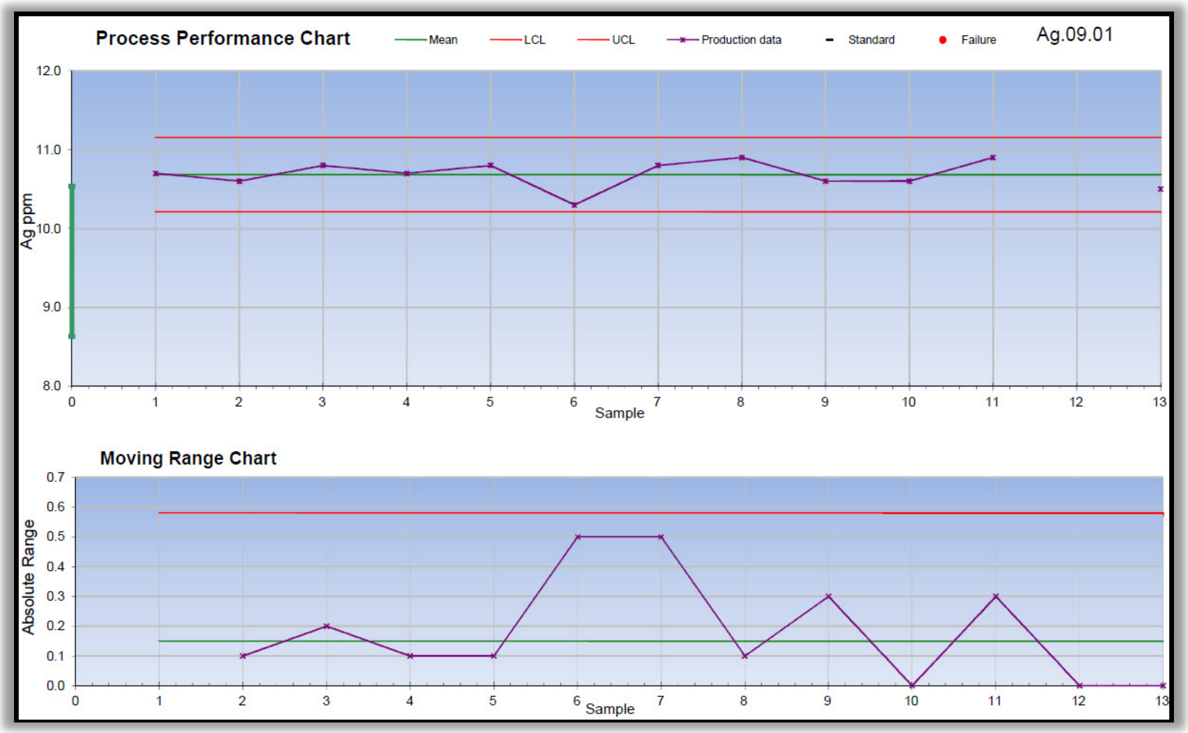


Figure 11.9 – SRM Au.09.03 – Gold Plot

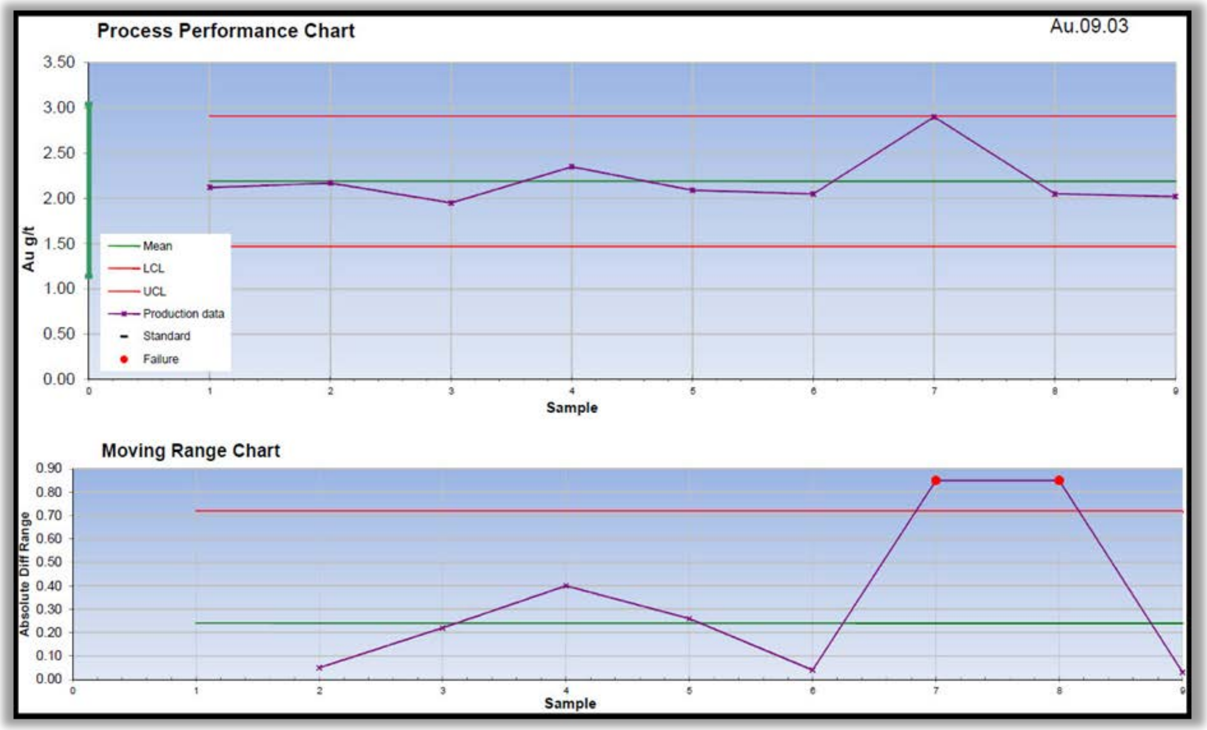


Figure 11.10 – SRM Au.09.03 – Silver Plot

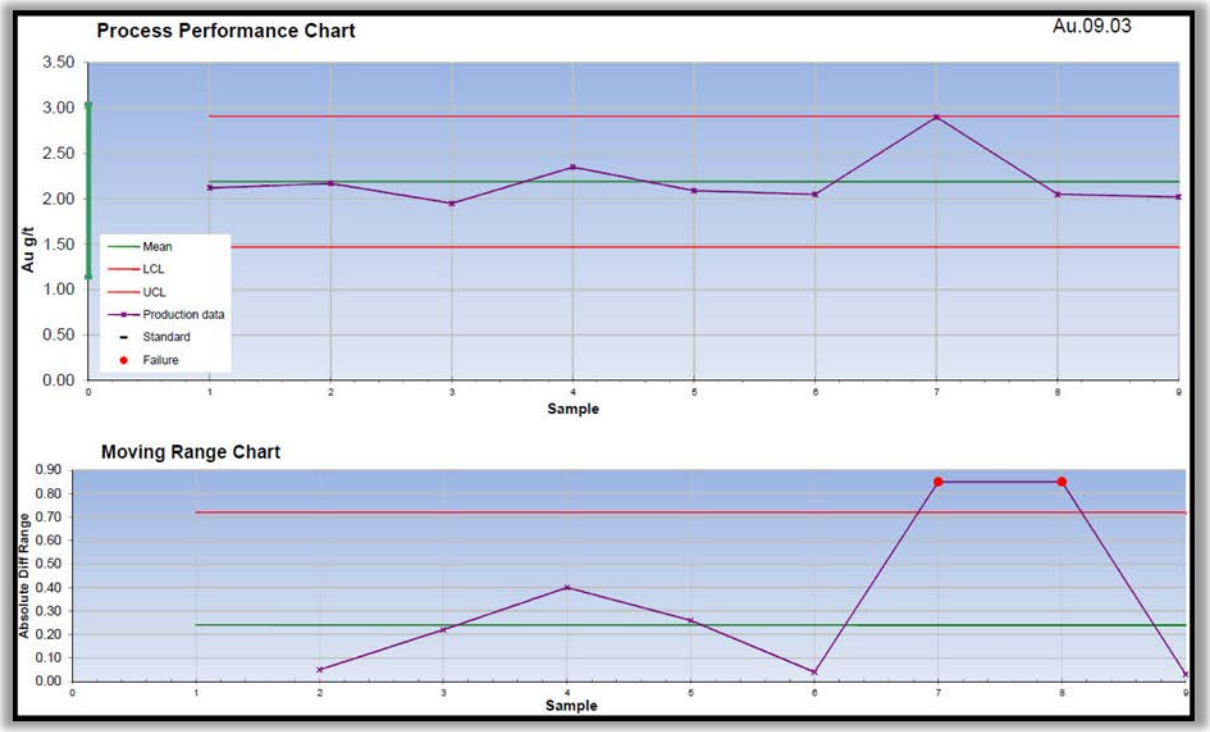


Figure 11.11 – SRM Au.09.04 – Gold Plot

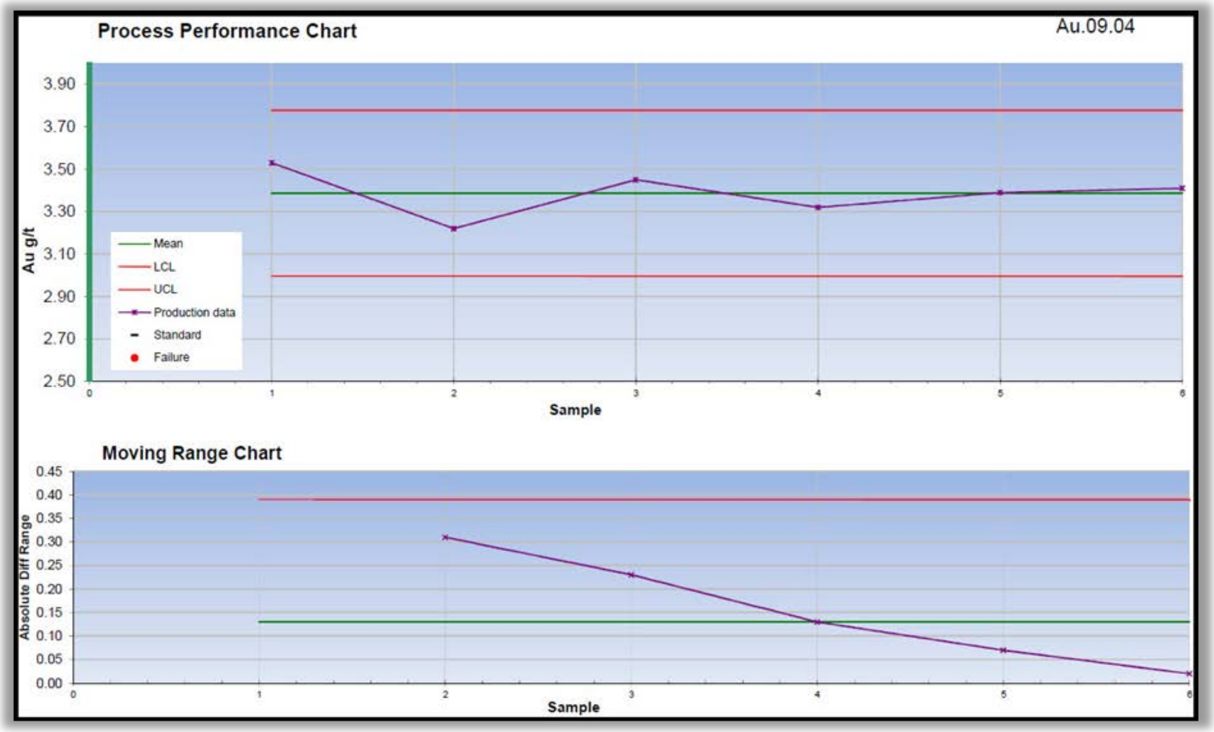


Figure 11.12 – SRM Au.09.04 – Silver Plot

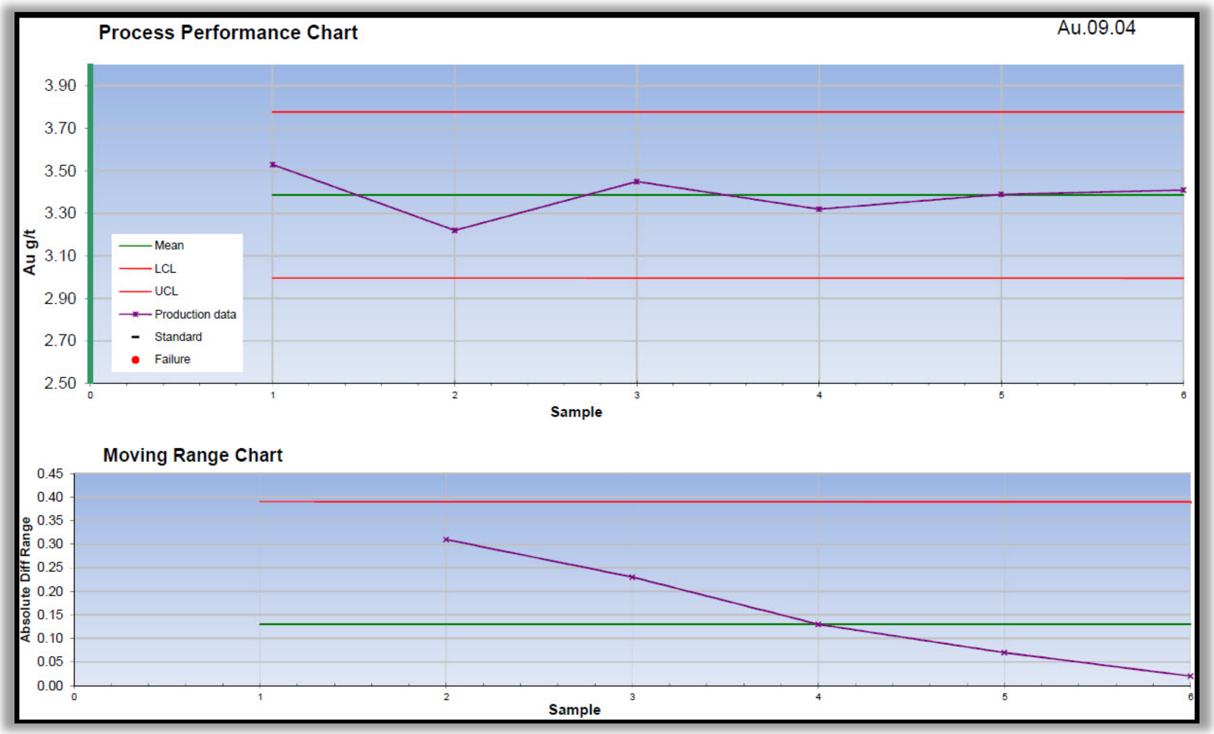


Figure 11.13 – SRM S107004X – Gold Plot

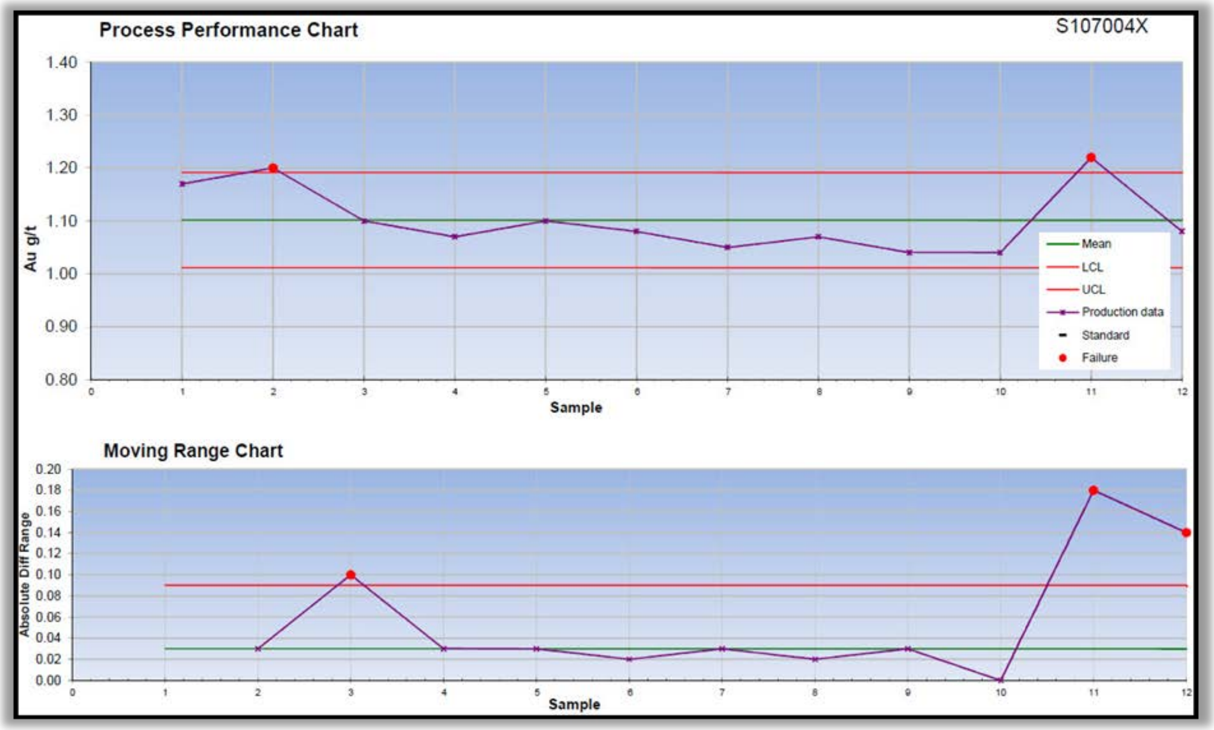
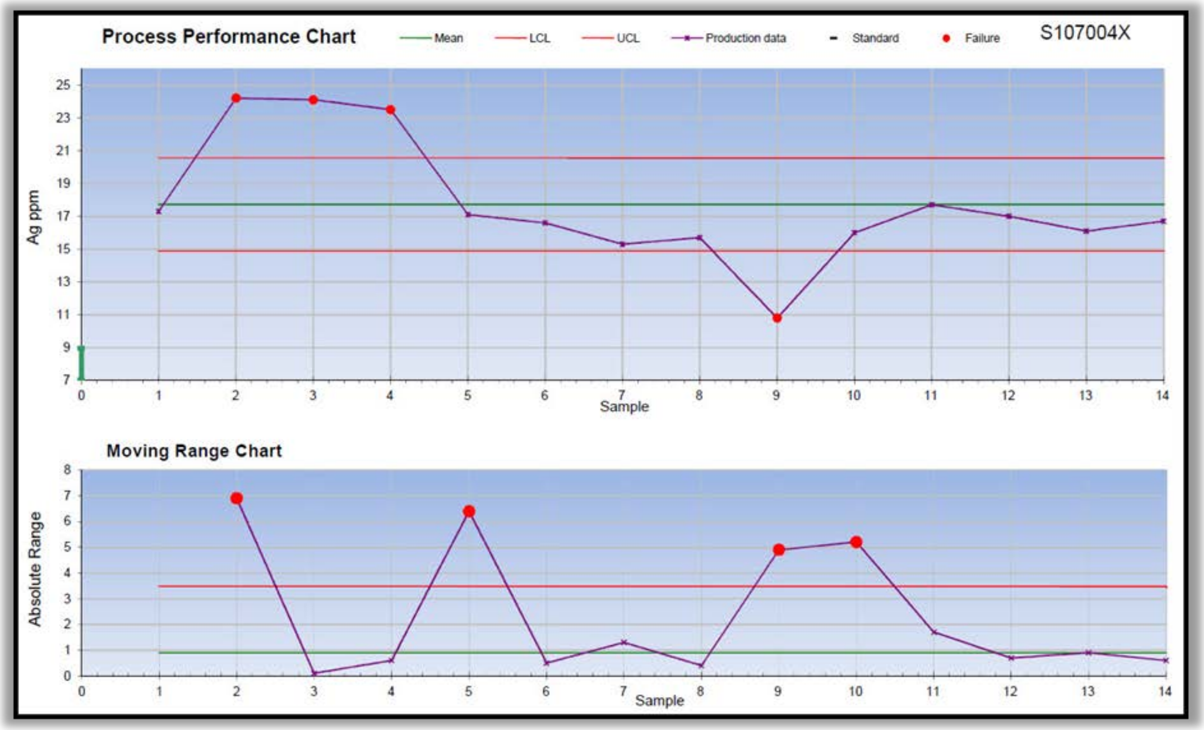


Figure 11.14 – SRM S107004X – Silver Plot



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 dataset 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

13.1 SUMMARY

A substantial amount of metallurgical test work has been performed over the years dating back to 1981 on various materials from the Talapoosa deposit. Results from these tests are varied but indicate that the resource materials contained within the PEA pit shell are amenable to industry standard cyanide heap leach extraction processes.

Several specific conclusions can be drawn from the results of this test work:

- The mineralized material at Talapoosa is amenable to heap leach cyanide extraction, but recoveries of gold and silver are sensitive to crush size;
- Mineralogical examination frequently found gold in solid solution with silver (electrum);
- Coarse gold has also been found in the deposit; and
- Gold recovery rates are slow and extended cyanide leach cycle times are required.

Results from these programs provided the basis from which the following estimated heap leach recoveries were used in the development of the PEA pit shell. These recovery estimates were made assuming heap leaching of an agglomerated nominal 1.7 mm (10 mesh) high pressure grinding roll product (Table 13.1). Estimates of gold and silver recoveries and reagent consumptions were made based on a review of the available column leach test data for the appropriate materials

Table 13.1 – Leach Recoveries

	Au Recovery	Ag Recovery	NaCN, kg/mt	Lime kg/mt	Cement kg/mt
Oxidized (HW and FW type)	77%	47%	0.6	0	5
HW Type (unoxidized)	65%	60%	0.8	2	5
FW Type (unoxidized)	59%	45%	0.8	2	5

Note that these are preliminary estimates of recovery and need substantial additional testing of each mineralized material type to further substantiate the expected extraction that may be achieved before advancing the project to a design-build stage.

13.2 MCCLELLAND LABORATORIES INC. – FEBRUARY 2015

The most recent metallurgical test work program at Talapoosa was undertaken by Gunpoint in late 2013 and completed in 2015 to evaluate ore types from the Bear Creek zone for amenability to heap leach cyanidation. The testing was conducted at McClelland Laboratories (Davis, 2015). Test work conducted included bottle roll and column leach tests, on HW- and FW-Type 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 (2013) Talapoosa resource summary. A more limited scope of work was also conducted on oxidized composites from the Bear Creek zone, and HW-Type material from the unoxidized Main zone.

Cyanidation test work 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 13.2.

Table 13.2 – 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 Unoxidized)				**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 Unoxidized)				**470**	**1.20**
GUN_L5	GTI-005	119	252	133	
GUN_L5	GTI-007	374	700	326	
GUN_L5 (Bear Creek Hanging Wall Unoxidized)				**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 Sulphide				**330**	**1.10**
GUN_L7	GTI-005	42.5	119	76.5	
GUN_L7 (Bear Creek Hanging Wall 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 Unoxidized)				**60**	**0.83**

Six of the composites tested represented material from the Bear Creek zone. Those included two HW-Type samples (GUN_L3 and GUN_L5) and FW-Type samples (GUN_L4 and GUN_L6), and one each representing oxidized material from the Bear Creek HW (GUN_L7) and FW-Type (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 HW-type zone (Baker, 2014).

Principal objectives of the test work program were to:

- Evaluate and compare heap leach and agitated cyanidation gold and silver recoveries of samples from the HW-Type and FW-Type.
- Evaluate the potential benefits of HPGR (high pressure grinding rolls) vs. conventional crush on both the Bear Creek HW and FW zones.
- Compare gold and silver recoveries between oxidized and unoxidized composites from the Bear Creek HW and FW zones.
- Confirm that the gold and silver recoveries from the unoxidized material from the Main zone were similar to those from the Bear Creek zone HW zone.

The test work 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 HW composite at 6.3 mm and 3.4 mm (conventionally crushed and HPGR product) feed sizes. Column tests were conducted on four composites comparing conventionally crushed 6.3mm feed and HPGR product (3.4 mm). 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) testing was conducted on a select column leached residue, to evaluate permeability of the leached agglomerates under simulated commercial heap stack height compressive loadings.

Summary results from the bottle roll, and column tests are shown in Tables 13.3 and 13.4, respectively. Correlation between the bottle roll (1.7 mm feed size) and most comparable column leach test results (HPGR) was poor with the latter typically producing higher recoveries.

Table 13.3 – Summary Results, Bottle Roll Tests, Gunpoint 2013 Metallurgical Testing

Composite	Material Type	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	Bear Creek HW	6.3 mm	31.4	1.02	23.1	14.7	<0.07	1.0
GUN_L3	Bear Creek HW	3.4 mm	48.5	1.03	36.7	14.7	0.13	1.2
GUN_L3	Bear Creek HW	HPGR	50.0	1.04	36.4	14.3	0.15	1.4
GUN_L3	Bear Creek HW	1.7 mm	59.3	0.91	41.5	13.0	0.13	1.3
GUN_L3	Bear Creek HW	75 µm	82.4	0.85	58.9	12.9	0.10	1.5
GUN-L4	Bear Creek FW	1.7 mm	53.8	0.80	30.2	6.3	0.13	1.6
GUN-L4	Bear Creek FW	75 µm	73.4	0.94	54.1	7.4	<0.07	1.6
GUN_L5	Bear Creek HW	1.7 mm	45.8	0.83	39.1	17.9	0.10	2.9
GUN_L5	Bear Creek HW	75 µm	68.0	0.97	51.7	14.5	0.21	3.6
GUN-L6	Bear Creek FW	1.7 mm	38.2	1.10	38.4	19.8	0.21	2.8
GUN-L6	Bear Creek FW	75 µm	71.8	1.17	57.1	14.7	0.13	2.0
GUN_L7	Bear Creek HW oxide	1.7 mm	72.1	0.43	38.6	4.4	0.19	5.1
GUN_L7	Bear Creek HW oxide	75 µm	92.1	0.63	77.4	5.3	0.16	6.3
GUN_L8	Bear Creek FW oxide	1.7 mm	54.3	0.46	40.3	6.2	0.13	3.7
GUN_L8	Bear Creek FW oxide	75 µm	76.8	0.56	48.8	4.1	0.29	5.1
GUN_L9	Main zone oxide	1.7 mm	59.2	0.76	53.1	22.8	0.17	2.3
GUN_L9	Main zone oxide	75 µm	78.9	1.09	50.9	26.7	0.26	3.1

Bottle roll test results (Table 13.3) showed that gold recoveries at the 1.7 mm 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 (72.1%) for the GUN_L7 composite. As described above, GUN_L6 was one of the two Bear Creek FW unoxidized composites. GUN_L7 was the Bear Creek HW oxide composite. Silver recoveries from the 1.7 mm feed size bottle roll tests were low (30.2% - 53.1%).

Milling the composites to 80%-75 µm generally improved gold recovery for agitated cyanidation (bottle roll testing) by between 19.6% and 23.1% (Table 13.3). 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%.

Column leach tests (Table 13.4) were completed on four of the Gunpoint composite samples at variable feed sizes with results confirming a relationship of increased recovery with decreased particle size. Column test gold recoveries from the four conventionally crushed 6.3 mm feeds were 60.3% (GUN_L-3),39.2% (GUNLL-4), 42.2% (GUN_L-5), 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 3.4 mm 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 6.3 mm and 3.4 mm (conventionally crushed) feed sizes. Gold recoveries for the 3.4 mm composite GUN_L-4 feed was approximately 17% higher than obtained at the 6.3 mm feed size.

Comparative column tests were conducted on HPGR product (3.4 mm feed size) for all four of the composites tested at the 6.3 mm 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 3.4 mm 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 3.4 mm feed size, so it was not possible to directly compare recoveries from conventionally crushed and HPGR product, at a 3.4 mm 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 3.4 mm). 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 possibly by preferential rock breakage during HPGR grinding. Further testing would be required to determine the degree to which each improved gold recovery.

Figures 13.1 to 13.4 show column test gold extraction versus leach time, and demonstrate that HPGR and the finest of the conventional crush particle size tests typically showed best recoveries for the four Gunpoint composites. Gold recovery rates in the column tests were generally not substantially different for the finest conventionally crushed and HPGR products. It is notable that gold extraction was continuing although at a slow rate when leaching was terminated for all column feeds. Heap leach cycles of 120 days will be required to maximize gold recoveries.

Overall, comparative results between column tests on conventionally crushed and HPGR product samples were somewhat inconclusive with respect to the benefits of HPGR crushing vs. conventional crushing to comparable size. 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 crushing may have 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 3.4 mm feed size were substantially higher than obtained at a 6.3 mm feed size. As such, crushing to 1.7mm (10 mesh) is estimated to provide a further increase in recovery compared to a 3.4 mm (6 mesh) product as tested.

Table 13.4 – Summary Results, Column Leach Tests, Gunpoint 2013 Metallurgical Testing

Composite	Material Type	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.	Reagent Requirements, kg/mt ore Lime Added	Reagent Requirements, kg/mt ore Cement Added
GUN_L3	HW	6.3 mm	205	60.3	1.41	35.8	14.8	2.08	1.3	----
GUN_L3	HW	3.4 mm	205	59.8	1.02	46.5	12.9	1.64	----	3.0
GUN_L3	HW	HPGR	205	66.9	1.24	49.6	13.3	1.63	----	3.0
GUN-L4	FW	6.3 mm	205	39.2	1.02	28.6	10.5	1.36	1.6	----
GUN-L4	FW	3.4 mm	205	55.8	0.95	35.1	9.7	1.52	----	3.0
GUN-L4	FW	HPGR	205	56.3	0.96	37.3	10.2	1.42	----	3.0
GUN_L5	HW	6.3 mm	206	42.2	1.02	54.8	14.6	1.40	2.9	----
GUN_L5	HW	HPGR	204	58.8	1.02	66.1	16.5	1.79	----	3.5
GUN-L6	FW	6.3 mm	205	44.3	1.22	38.0	17.1	1.45	2.2	----
GUN-L6	FW	HPGR	204	57.5	1.13	48.4	15.9	1.85	----	3.5

Figure 13.1 - Composite L-3 (Bear Creek HW Zone) Column Test Leach Rate Profiles for Gold

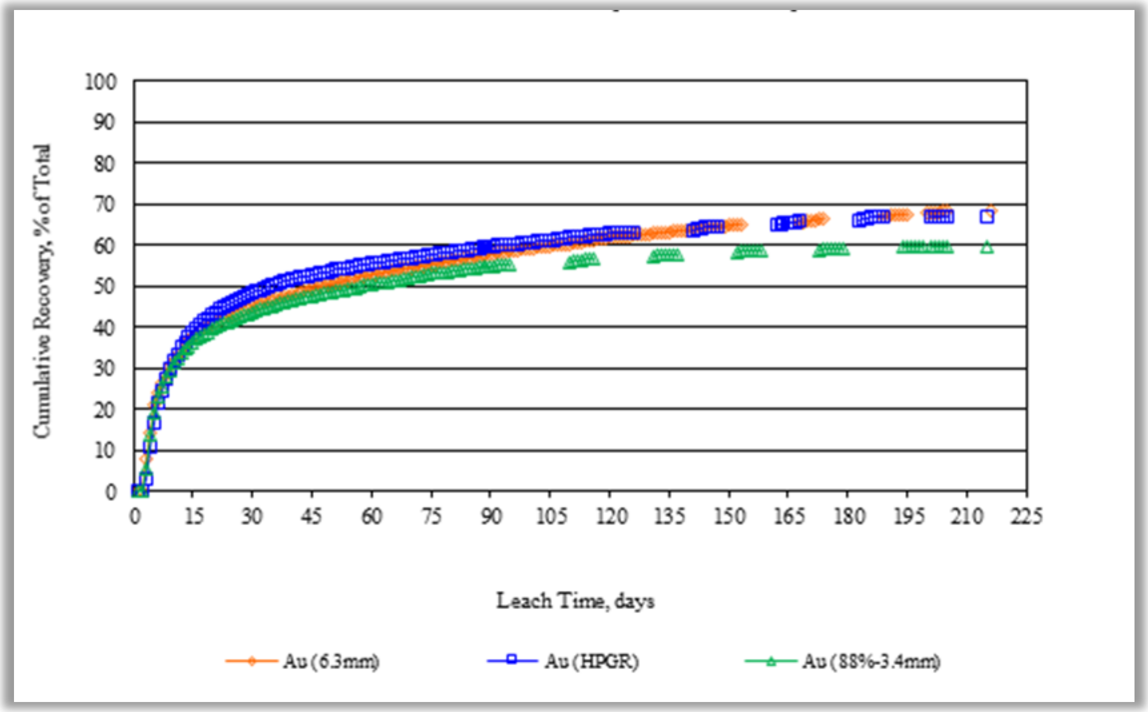


Figure 13.2 – Composite L-4 (Bear Creek FW Zone) Column Test Leach Rate Profiles for Gold

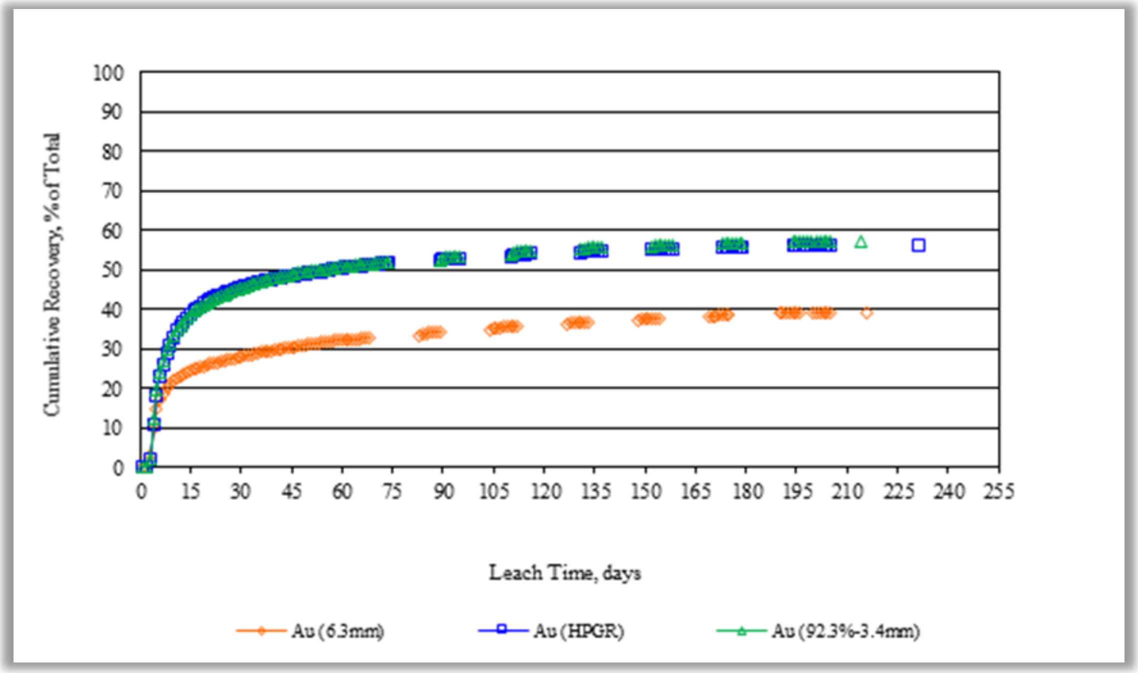


Figure 13.3 – Composite L-5 (Bear Creek HW Zone) Column Test Leach Rate Profiles for Gold

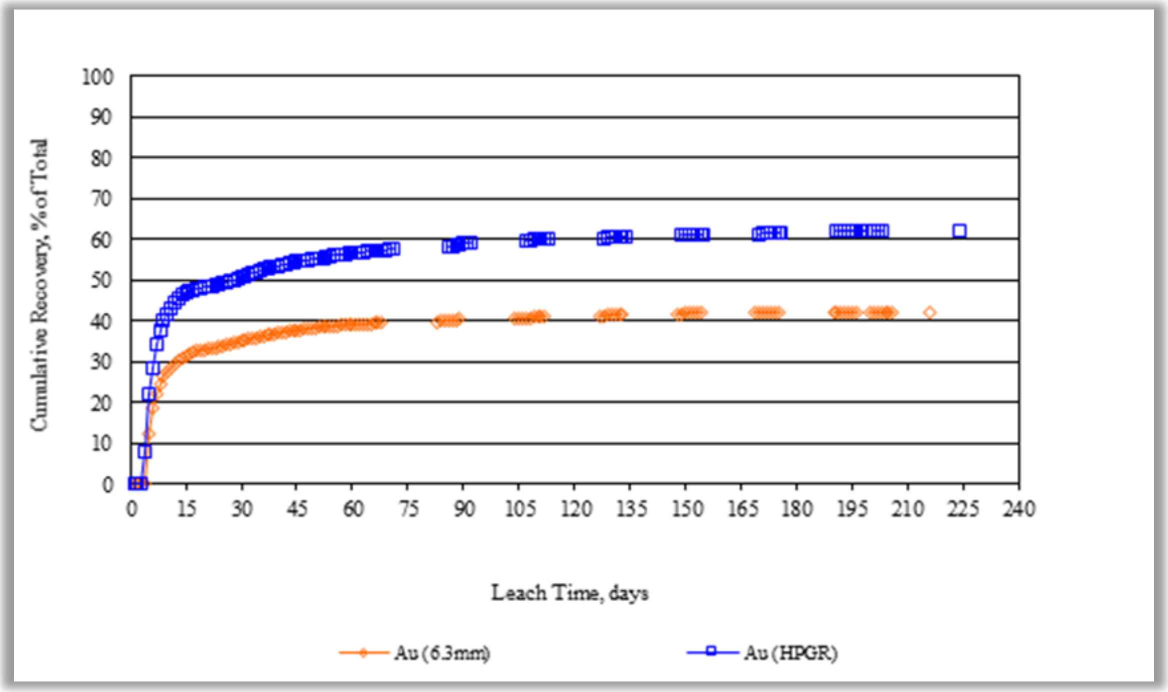
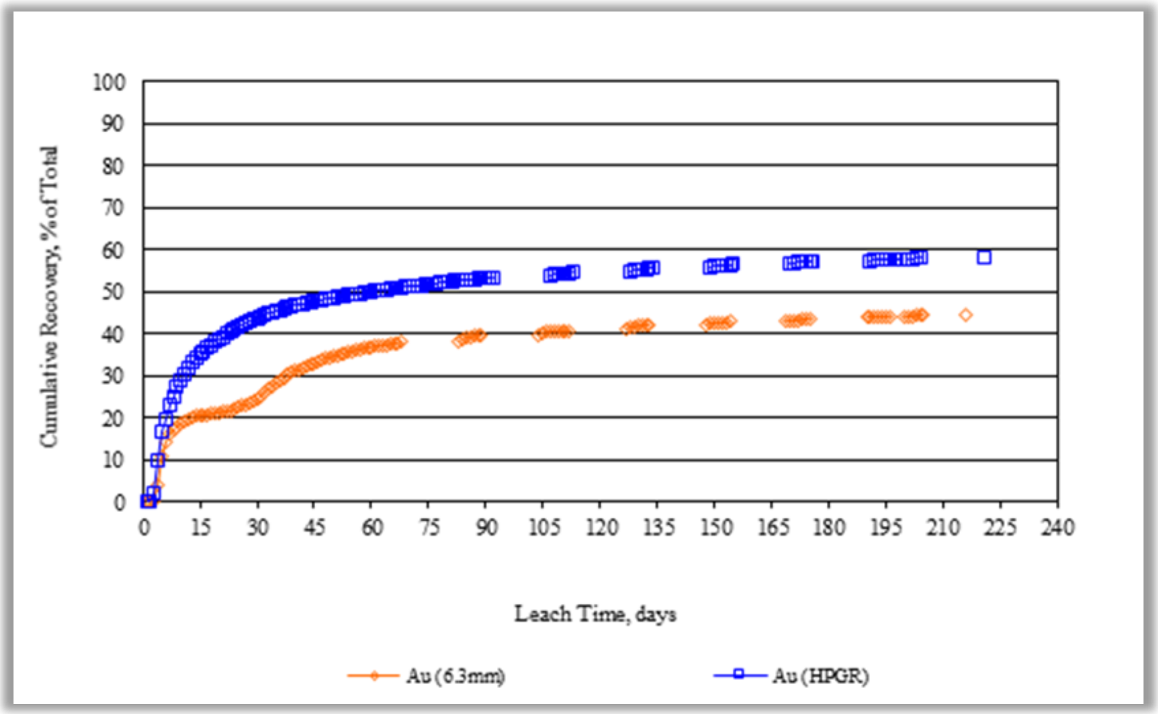


Figure 13.4 – Composite L-6 (Bear Creek FW Zone) Column Test Leach Rate Profiles for Gold



Comparison between simulated heap leach and milling/cyanidation test results (reflected in bottle roll tests) show that, for the two HW-Type composites (GUN_L3 and GUN_L5), milling/cyanidation treatment (75 µm) (Table 13.3) resulted in a gold recovery that was 9% to 15% higher than obtained by column testing (considering the maximum cyanide leach test recovery) (Table 13.4). In the case of the two FW Type 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 test work 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 may not be useful for predicting crushed ore heap leach gold recoveries.

Load/permeability test work was completed on the four Gunpoint composite columns. Of these four, the composite GUN_L-3 residue showed a hydraulic conductivity of 7.8×10^{-3} cm/sec. at a simulated heap stack height of 25 m. That was significantly higher than the equivalent solution application rate used during leaching (3.3×10^{-4} cm/sec.). This result indicates acceptable permeability (up to a 25 m stack height) for the one of the four HPGR products as tested.

The remaining three Gunpoint composite column tests displayed ponding problems when rinsed with fresh water after column leaching. The poor permeability during column testing of these three indicates that further optimization of agglomerating conditions will be required for heap leaching of the HPGR product.

Conclusions reached from the test work program are summarized as follows.

- All of the Bear Creek zone and the Main zone composites tested were sensitive to feed size with respect to gold and silver recovery.
- 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 with respect to improved gold recovery.
- 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.
- 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 HW oxide composite was readily amenable to agitated cyanidation at the 1.7 mm and 75 µm feed sizes with recoveries of 72.1% and 92.1%, respectively.
- Gold recoveries from agitated cyanidation at the 1.7 mm and 75 µm feed sizes were similar for the Main zone composite and the Bear Creek HW GUN_L-3 composite.
- Agitated cyanidation gold recovery rates were rapid at the milled feed size (75 µm) but much slower at coarser feed sizes.

13.3 HISTORIC TEST PROGRAMS

Between 1981 and 1999, there were 12 metallurgical test work programs conducted on the Property, by various stakeholders. Testing focused mainly on heap leaching, but included some work relevant to potential mill / concentrator processing. Results from the recently completed test work for Gunpoint generally supported results from the earlier heap leach test work.

There is considerable inconsistency in terminology between the various historic testing programs making it necessary to adopt a standardized terminology when describing the different programs (Table 13.5).

Table 13.5 – Correlation of Various Terminology Used in Previous Test work with Current Nomenclature

Term	Description
Oxide	Refers to all oxidized material, from all zones and both primary types of mineralization, (HW type and FW type). In some cases of historic sampling, composites comprised a mixture of oxidized and unoxidized material; in this report, these composites are identified by the suffix '-mixed'. Many of the Main zone samples were oxide and have simply been referred to as oxide in this review of the metallurgy.
HW Type	Refers to unoxidized mineralization from Main zone, East Hill, Dike Adit and the HW section of the Bear Creek zone. In much of the pre-2014 test work, the unoxidized Bear Creek zone was sub-divided; in some of the earlier work it was referred to as Lower Bear Creek. Elsewhere, this material has been referred to as 'unoxidized".
FW Type	Refers to unoxidized mineralization from the FW section of the Bear Creek zone. In some of the pre-2014 test work this material has been referred to variously as Upper Bear Creek, sulphide or simply Bear Creek.

The historic test work programs focused primarily on evaluation of material from the Bear Creek and Main zones. A smaller amount of work was conducted on material from the Dyke Adit and East Hill zones. Details of those test work programs were summarized in the previous technical report (McCracken and McPartland, 2015) and are summarized in subsequent sub-sections.

The recent Gunpoint work mainly considered HW and FW Type material from the deeper unoxidized portion of the Bear Creek zone (see Section 13.2). While the upper portion of the Bear Creek zone is oxidized to a depth of approximately 100 feet, the majority of the zone is located below the depth limit of oxidation. In the Gunpoint program on Bear Creek zone material, particular care was taken to separate HW type and FW Type mineralization when making composites.

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 recovery of the gold and silver by flotation was generally higher for the unoxidized material than for the oxidized material. Very little testing has been conducted to evaluate further processing of flotation concentrates for recovery of contained gold and silver.

Work in the 1980s showed that the predominantly oxidized Main zone material was for the most part more amenable to simulated heap leach processing than the Bear Creek zone unoxidized material, the bulk of which composite samples were a mix of HW and FW Type mineralization. Available mineralogy generally indicated that the majority of the gold was present as electrum (gold silver), so the leach kinetics were slow, leading to extended heap leach cycle times.

Both oxidized and the unoxidized mineralization was 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. Test work included evaluation of recoveries achieved at various feed size distributions as prepared by different reduction equipment.

In historic tests, the use of high-pressure grinding rolls (HPGR) for size reduction to approximately 6 mesh generally resulted in increased gold and silver recoveries, but it is noted that improvements may have resulted simply from generation of a finer particle size distribution than conventional crushing. In the case of unoxidized Bear Creek zone material, a significant portion of the contained gold and silver is only recoverable from sulphide mineral grains after liberation through fine crushing.

As with recent McClelland Laboratories study (Section 13.2), available historic metallurgical test work results summarized below by program suggest that conventional heap leaching at a relatively fine crush size is likely the best approach for processing the oxide and unoxidized materials from the Property. While gold and silver recoveries from the unoxidized material were lower than for the oxidized material, it has been demonstrated that acceptable recoveries were obtained from the unoxidized materials using the same process (simulated heap leaching at a relatively fine crush size).

Further, historic work shows that grinding to finer sizes followed by flotation treatment gives recoveries of over 90% into rougher flotation concentrates for the unoxidized materials tested. Such flotation concentrates would likely be reground and cleaned. Cleaner flotation concentrates would subsequently be treated by one of several regrinding / oxidation / cyanidation methods or be direct shipped to a smelter as sulphide feed.

13.3.1 SUMMARY OF HISTORIC RECOVERY RESULTS

Extensive historic column leach testing has been conducted on oxidized materials, represented mainly by samples from the Main Zone. Most of those column leach tests were conducted on material coarser in size than the 1.7 mm HPGR product considered for the PEA. A summary of results from all column leach tests conducted on oxidized samples is presented in Table 13.6 and gold recovery versus feed size data are graphically presented in Figure 13.5.

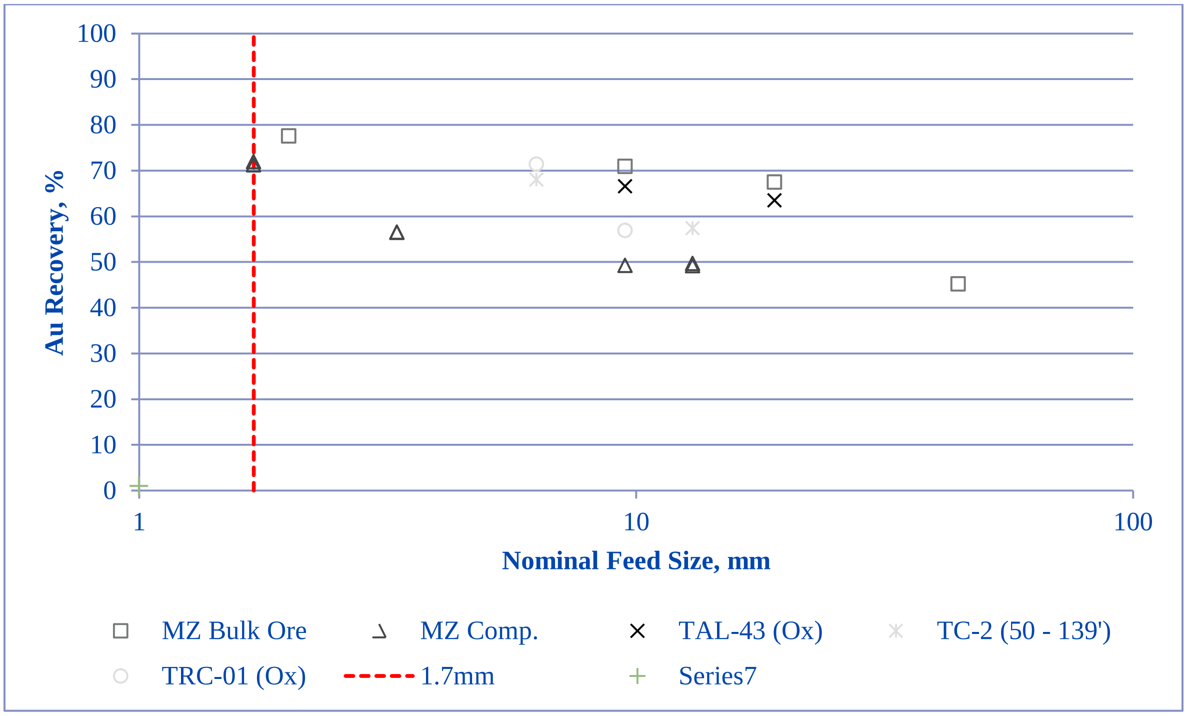
Table 13.6 - Summary of Results from all Column Leach Tests Conducted on Oxidized Samples

Study	**Sample	Leach Time, days	Crusher Type	Nominal Feed Size	Recoveries % Au	Recoveries % Ag	Head Grades gAu/mt	Head Grades gAg/mt	Reagents Required kg/mt NaCN Cons.	Reagents Required kg/mt Lime Added
MLI (5/94)	MZ Bulk Ore[J]	531	Conv.	45mm	45.2	36.0	2.50	47.2	N/A	N/A
MLI (5/94)	MZ Bulk Ore	531	Conv.	19mm	67.5	64.7	2.64	42.8	N/A	N/A
Bateman (1989)	Glory Hole (Ox)	101	Conv.	19mm	37.8	22.8	0.82	31.5	0.65	10.0*
Bateman (1989)	TAL-43 (Ox)	101	Conv.	19mm	63.5	23.8	0.27	5.5	0.69	10.0*
DML (94-97)	MZ Comp.	122	Conv.	13mm	49.2	23.3	0.94	13.9	0.66	6.2
DML (94-97)	MZ Comp.	325	Conv.	13mm	49.7	41.8	1.43	10.3	1.79	4.3
MLI (1989, Phase II)	TC-2 (50 - 139')	54	Conv.	13mm	57.4	31.3	1.61	11.0	1.03	5
MLI (5/94)	MZ Bulk Ore	531	Conv.	9.5mm	71.0	64.1	2.13	43.0	N/A	N/A
DML (94-97)	MZ Comp.	122	Conv.	9.5mm	49.3	16.2	0.87	21.7	0.66	5.6
Bateman (1989)	TRC-01 (Ox)	112	Conv.	9.5mm	56.9	31.8	2.23	35.3	1.24	10.0*
Bateman (1989)	T-01 (Ox)	116	Conv.	9.5mm	65.6	24.9	1.10	13.4	0.82	10.0*
Bateman (1989)	Glory Hole (Ox)	101	Conv.	9.5mm	32.5	21.8	0.96	32.9	0.83	10.0*
Bateman (1989)	TA-2 & 3 (Ox)	112	Conv.	9.5mm	66.1	41.2	0.51	7.5	0.85	10.0*
Bateman (1989)	TAL-43 (Ox)	101	Conv.	9.5mm	66.6	49.0	0.31	3.4	0.70	10.0*
Bateman (1989)	TRC-13 (Ox)	112	Conv.	9.5mm	38.0	15.5	0.62	16.1	1.17	10.0*
MLI (1989, Phase II)	TC-2 (50 - 139')	54	Conv.	6.3mm	68.1	48.8	1.61	11.3	0.42	5.0
MLI (1989, Phase II)	TC-4 (0 - 57')	54	Conv.	6.3mm	64.7	37.2	1.75	26.7	1.14	5.0
MLI (7/94)	Comp. 1	421	Conv.	6.3mm	72.2	45.5	1.85	18.9	3.58	3.5
MLI (7/94)	Comp. 1	405	Conv.	6.3mm	73.2	50.0	1.92	17.8	3.83	3.5
MLI (7/94)	Comp. 2	57	Conv.	6.3mm	75.0	38.9	0.41	6.2	1.10	3.5
MLI (7/94)	Comp. 2	57	Conv.	6.3mm	83.3	41.2	0.41	5.8	1.24	3.5
Bateman (1989)	TRC-01	79	Conv.	6.3mm	71.4	35.0	2.33	32.2	1.35	10.0*
DML (94-97)	MZ Comp.	122	Conv.	3.3mm	56.5	44.0	0.91	9.8	0.64	5.8
DML (94-97)	MZ Comp.	321	Conv.	3.3mm	56.6	63.8	1.07	9.5	1.78	4.0
MLI (5/94)	MZ Bulk Ore	531	Conv.	2mm	77.6	67.3	1.99	43.0	N/A	N/A
DML (94-97)	MZ Comp.	295	HPGR	1.7mm	71.3	85.5	1.05	9.5	1.61	4.0
DML (94-97)	MZ Comp.	295	HPGR	1.7mm	72.0	80.9	1.08	10.4	1.54	4.0

* Cement added during agglomeration pretreatment.

Note: For crusher type, Conv. denotes conventional laboratory (jaw or cone) crushers. HPGR denotes high pressure grinding rolls.
**Historic terminology

Figure 13.5 Gold Recovery vs. Normal Feed Size in Column Tests on Historic



Extensive column leach testing has also been conducted on unoxidized material, represented mainly by samples from the Bear Creek Zone (Table 13.7). Most of the column tests were conducted in the 6.3 mm or finer feed size range, including a significant number of tests conducted on samples prepared using HPGR. Gold recovery versus feed size data for the unoxidized samples is shown graphically in Figure 13.6.

Table 13.7 - Summary of Histocial Metallurgical Column Leach Test Results for Bear Creek Unoxidixed Samples

				Nominal					Reagents Required	
		Leach Time,	Crusher	Feed	Recoveries		Head Grades		kg/mt	
Study	**Sample	days	Type	Size	% Au	% Ag	gAu/mt	gAg/mt	NaCN Cons.	Lime Added
Bateman (1989)	TA-10 (Sulf)	105	Conv.	9.5mm	43.7	25.6	0.72	17.1	0.73	10.0*
Bateman (1989)	T-3 (Sulf)	106	Conv.	9.5mm	80.3	69.2	0.51	8.9	1.08	10.0*
Bateman (1989)	T-08 (Sulf)	115	Conv.	9.5mm	45.4	23.8	0.45	6.2	1.22	10.0*
MLI (2015)	L-3 (BC HW UnOx)	205	Conv.	6.3mm	60.3	35.8	1.41	14.7	2.08	1.3
MLI (2015)	L-5 (BC HW UnOx)	206	Conv.	6.3mm	42.2	54.8	1.03	14.7	1.40	2.9
MLI (2015)	L-4 (BC FW UnOx)	205	Conv.	6.3mm	39.2	28.6	1.03	10.6	1.36	1.6
MLI (2015)	L-6 (BC FW UnOx)	205	Conv.	6.3mm	44.3	38.0	1.23	17.1	1.45	2.2
MLI (7/94)	Comp. 3	362	Conv.	6.3mm	36.8	39.5	0.65	13.0	2.15	3.5
MLI (7/94)	Comp. 4	665	Conv.	6.3mm	61.9	45.8	2.16	16.5	3.78	3.5
MLI (7/94)	Comp. 5	675	Conv.	6.3mm	44.6	59.7	2.54	23.0	3.55	3.5
MLI (5/94)	Upper BC Sulf. Cuttings	223	Conv.	6.3mm	59.0	59.6	1.34	30.5	N/A	N/A
MLI (5/94)	Lower BC Sulfide	231	Conv.	6.3mm	45.5	46.7	1.13	20.6	N/A	N/A
MLI (5/94)	Lower BC Sulfide	231	Conv.	6.3mm	53.3	39.7	1.03	25.0	N/A	N/A
DML (94-97)	Bear Creek No. 1	99	Conv.	3.3mm	37.7	31.9	0.58	5.5	0.81	1.2
DML (94-97)	Bear Creek No. 1	99	Conv.	3.3mm	52.2	34.9	0.63	5.3	0.80	1.2
DML (94-97)	Bear Creek HG	275	Conv.	3.3mm	67.2	70.5	3.09	26.7	1.52	3.6
DML (94-97)	Bear Creek LG	270	Conv.	3.3mm	36.0	62.1	0.37	1.8	1.39	2.8
MLI (2015)	L-3 (BC HW UnOx)	205	Conv.	2.8mm	59.8	46.5	1.02	12.9	1.64	3.0
MLI (2015)	L-4 (BC FW UnOx)	205	Conv.	2.8mm	55.8	35.1	0.95	9.7	1.52	3.0
DML (94-97)	Bear Creek No. 2	239	Conv.	1.7mm	28.6	40.4	0.98	6.2	1.46	2.4
DML (94-97)	Bear Creek No. 2	239	Conv.	1.7mm	46.2	64.4	1.01	5.8	1.55	2.8
DML (94-97)	Bear Creek No. 1	111	Conv.	1.7mm	43.2	33.0	0.81	7.7	0.88	1.4
MLI (5/94)	Lower BC Sulfide	212	VSI	3.3mm	52.6	55.4	1.30	19.2	N/A	N/A
DML (94-97)	Bear Creek No. 1	89	VSI	1.7mm	47.2	31.5	0.88	8.0	0.77	1.2
MLI (2015)	L-3 (BC HW UnOx)	205	HPGR	2.4mm	66.9	49.6	1.24	13.3	1.63	3.0*
MLI (2015)	L-6 (BC FW UnOx)	205	HPGR	2.3mm	44.3	38.0	1.22	15.9	1.85	3.5*
MLI (2015)	L-5 (BC HW UnOx)	204	HPGR	2.1mm	58.8	66.1	1.03	16.5	1.79	3.5*
MLI (2015)	L-4 (BC FW UnOx)	205	HPGR	2.1mm	56.3	37.3	0.96	10.2	1.42	3.0*
DML (94-97)	Bear Creek No. 1	366	HPGR	1.7mm	57.8	83.0	0.87	4.8	1.77	4.1
DML (94-97)	Bear Creek No. 1	119	HPGR	1.7mm	47.9	N/A	0.86	N/A	0.91	1.2

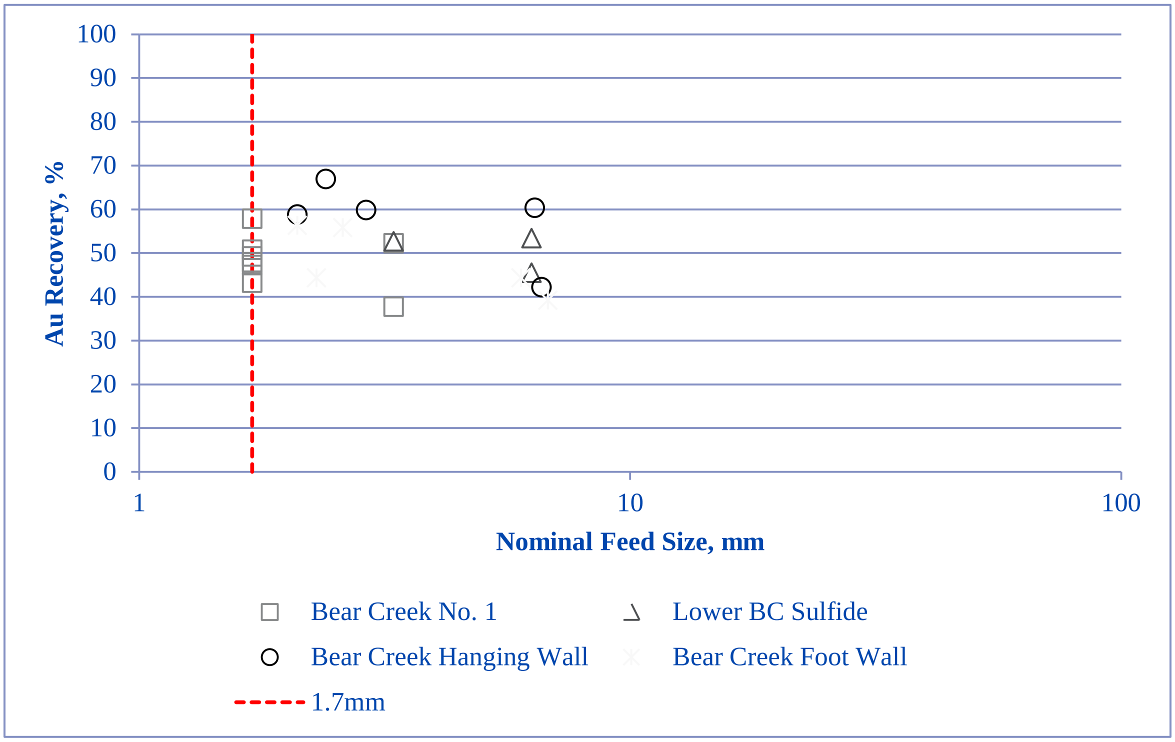
Study	**Sample	Leach Time, days	Crusher Type	Nominal Feed Size	Recoveries % Au	Recoveries % Ag	Head Grades gAu/mt	Head Grades gAg/mt	Reagents Required kg/mt NaCN Cons.	Reagents Required kg/mt Lime Added
DML (94-97)	Bear Creek No. 1	111	HPGR	1.7mm	49.2	36.6	0.91	9.7	0.94	1.2
DML (94-97)	Bear Creek No. 1	128	HPGR	1.7mm	50.7	50.6	1.08	7.6	1.06	1.7
DML (94-97)	Bear Creek No. 2	269	HPGR	1.7mm	46.3	43.8	1.09	8.5	1.38	3.4
DML (94-97)	Bear Creek No. 2	246	HPGR	1.7mm	46.5	67.9	1.21	5.7	1.36	2.9
DML (94-97)	Bear Creek No. 2	239	HPGR	1.7mm	50.0	50.0	1.01	8.0	1.51	3.1
DML (94-97)	Bear Creek No. 2	239	HPGR	1.7mm	49.4	63.5	1.04	6.1	1.51	2.8
DML (94-97)	Bear Creek No. 2	328	HPGR	1.7mm	51.8	69.8	1.07	6.7	1.56	2.4
DML (94-97)	Bear Creek No. 2	204	HPGR	1.7mm	54.4	86.7	1.12	5.2	1.30	2.5

* Cement added during agglomeration pretreatment.

Note: For crusher type, Conv. denotes conventional laboratory (jaw or cone) crushers. VSI denotes verticle shaft impact type crusher. HPGR denotes high pressure grinding rolls.

**Historic terminology

Figure 13.6 - Gold Recovery vs. Normal Feed Size in Column Tests on Unoxidized Historic Composite Samples



13.3.2 GOLD RECOVERY ESTIMATION

Gold recovery for the oxidized material was estimated at 77%, considering the reasonably consistent improvement in gold recovery with decreasing feed size observed with the oxidized samples (Table 13.6 and Figure 13.5). Silver recovery was more conservatively estimated at 47%, considering the less consistent size versus recovery trend.

Gold and silver recoveries for heap leaching of the unoxidized hanging wall and footwall materials at a 1.7 mm feed size were estimated as 2% higher than the average column test recoveries from the corresponding 2014 composites. Those composites were tested at an 80%-2.2 mm (average) feed size. These estimates are reasonable, considering the size versus recovery relationship observed for the four 2014 composites. Cyanide consumptions(Table 13.7) for the unoxidized materials were estimated as the average column test consumption for each material type (HW and FW-Type) through a 60-day leach cycle, which is considered to be a standard duration laboratory column leach testing. Cement and lime consumptions for heap leaching of the unoxidized materials (Table 13.7) were estimated as being significantly higher than that used during column testing of the HW and FW samples, as it was noted during testing that further optimization of base additions will be required.

Cyanide consumption for the oxidized material was estimated by averaging column test cyanide consumption for all of the oxide material column leach tests at a 6.3 mm and finer feed size, divided by a factor of 3. This is a common approach for discounting column test cyanide consumptions to predict commercial cyanide consumptions for relatively “clean” oxide ore types. The cement requirement was based on the average lime or cement requirement for the 6.3 mm and finer column leach tests on oxidized samples.

13.3.3 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 that 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 regarding 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 (All of Newcrest's metallurgical composites were comprised of HW Type mineralization from Bear Creek zone) 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 13.8. 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 13.9.

Table 13.8 – 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 13.9 – Oretest – Summary of Gravity Test Results at 150 µm

	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
Head Assays													
Calculated Head	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	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,	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	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	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
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 13.10. 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 13.11.

Table 13.10 – Oretest – Summary of Flotation Results at P_{80}=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
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 13.11 – Oretest – Summary of Flotation Test Results at P_{80}=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
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 suggests that increased gold recoveries could lead to increased silver recoveries. 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 13.12.

Table 13.12 – 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 Tables 13.13 and 13.14. 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 13.13 – Oretest – Summary of Bottle Roll Cyanide Leach Test at P_{80}=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 13.14 – Oretest – Summary of Bottle Roll Cyanide Leach Test at P_{80}=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 13.15.

Table 13.15 – Oretest – Results of Oxygen Addition to Vat Leach

COMPOSITE	GOLD RECOVERY (%)		SILVER RECOVERY (%)		NACN (KG/T)		LIME (KG/T)	
	Leach	Leach + O_2	Leach	Leach + O_2	Leach	Leach + O_2	Leach	Leach + O_2
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 13.16.

Table 13.16 – Oretest – Results of Lead and Oxygen Addition to Vat Leach

Composite	Gold Recovery (%)		Silver Recovery (%)		NaCN (kg/t)		Lime (kg/t)	
	Leach	Leach + O_2 + PbO	Leach	Leach + O_2 + PbO	Leach	Leach + O_2 + PbO	Leach	Leach + O_2 + 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 13.17.

Table 13.17 – 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 13.18.

Table 13.18 – 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 13.19.

Table 13.19 – 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)

13.3.4 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 13.20.

Table 13.20 – 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.

13.3.5 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 (Mixed HW and FW Type) and No. 2 (HW Type) 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 13.21.

Table 13.21 – 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	-	-

Head Analysis	Units and Elements	BC No. 1	BC No. 2	HG	LG
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 13.22.

Table 13.22 – 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 13.23. 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 13.24.

The tests showed that there was only a small percentage of free gold in the residue (2.1% and 3.8% respectively). Approximately 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 13.25.

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 13.26. The higher-grade material had a higher gold recovery (67.2%) than the lower-grade material (36%).

Table 13.23 – 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 13.24 – 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 13.25 – 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 13.26 – 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 (Table 13.27). 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 13.27 – 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
	P_{80} = 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 13.28. 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 products are presented in Table 13.28.

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 13.29. The gold extraction at the 841 µm crush size was 58% and 65% for silver.

Table 13.28 – 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 13.29 – 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 13.30. 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 13.31. 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 13.32. 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 sample were completed. The results showing equipment type and gold recovery are presented in Table 13.33. 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 13.30 – 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 13.31 – 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 13.32 – 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 13.33 – 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 13.34 – 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 13.34. The gold kinetics were fast for the high-grade sample and slower for the low grade.

13.3.6 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), with TAL-328, TAL-329 and TAL-331 being from mixed Oxide and FW-Type Bear Creek zone material, and TAL-300 from Dike Adit (HW-Type material). 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 13.35. 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 13.36. 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 13.37. 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 that 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 13.38. 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 13.39. 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 13.40. Again, the majority of the gold was leached in the finer fraction of material.

Table 13.35 – 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 13.36 – 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 13.37 – 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 13.38 – 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 13.39 – 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 13.40 – 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 13.41.

Table 13.41 – 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)

13.3.7 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.

13.3.8 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 that 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.

13.3.9 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.

13.3.10 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, with Bear Creek 1 compromising a mix of HW Type and FW Type material, and Bear creek 2 comprised exclusively of HW Type material, 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 13.42.

Table 13.42 – 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 13.43.

Table 13.43 – 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 13.44. 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 13.44 – 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 13.45.

Table 13.45 – 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 13.46.

Table 13.46 – 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 (A mix of HW and FW Type material) 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 (oxidized) and Bear Creek No. 1 (mix of HW and FW Type mineralization) composites. The preliminary results were presented. These have been summarized in Tables 13.47 and 13.48. 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 13.47 – 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 13.48 – 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 is 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) was tested with the Bear Creek Composite No. 1 sample. The results are presented in Table 13.49.

Table 13.49 – 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 13.50.

Table 13.50 – 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)

13.3.11 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, BC Comps 3 and 4 are from HW Type from Bear Creek mineralization, the other BC composites are presumably a mix of HW and FW Type mineralization. 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 13.51. The column leach results for the Main zone composites can be found in Table 13.52. The column leach results for the Bear Creek zone composites are shown in Table 13.53. Table 13.54 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 13.53. 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 13.51 – McClelland 1994 – Bottle Roll Leach Results

Sample	Head Assay (g/t)				Extraction (%)												NaCN Consumption (kg/t)	Lime Consumption (kg/t)
	Calculated		Assayed		Au						Ag							
	Au	Ag	Au	Ag	2 h	6 h	24 h	48 h	72 h	96 h	2 h	6 h	24 h	48 h	72 h	96 h		
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 13.52 – McClelland 1994 – Main 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	34 d	50 d	200 d	300 d	398 d	5 d	34 d	50 d	200 d	300 d	398 d		
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 13.53 – 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	3.5 6
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 13.54 – 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	3.5 6

Source: McClelland (1994)

13.3.12 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.

13.3.13 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 13.55. 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 Oxide from the Main zone. Composite 3 contains 131 intervals (200 m) and represents the low-grade section of the HW-Type mineralization from Bear Creek, similarly for Composites 4 and 5. 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 13.55 – 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 13.56.

Table 13.56 – 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 13.57.

Table 13.57 – 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 tests were in the 79 to 80% passing -75 µm range. Below are descriptions of the details of each test; Table 13.58 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 13.58 – 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 13.59.

Table 13.59 – 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
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. FW Type from Bear Creek, HW Type from Bear Creek HW Type from Main zone composites as presented in Table 13.60 were created.

Table 13.60 – 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 13.61.

Table 13.61 – 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, 123pprox.. 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 HW-Type composite from Bear Creek. 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 13.62.

Table 13.62 – 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.

13.3.14 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 Oxide from the Main zone bulk material, (HW Type and FW Type material from Bear Creek). 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 Oxide 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 Oxide sample gravity pre-concentrations test work showed that this sample was not amenable to pre-concentration by gravity separation.

HW Type and FW Type material from Bear Creek, 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.

13.3.15 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.

13.3.16 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 approximately 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 approximately 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.

13.3.17 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).

13.3.18 MCCLELLAND LABORATORIES INC. – 1989

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 report's 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. Composites of Main zone HW Type material from TAL-151 and TAL-154 was mixed. FW Type samples from Bear Creek were taken from TAL-151 (Main zone, Mixed oxide and unoxidized), and the Bear Creek zone composite from TAL-127, TAL-129 and TAL-157, was a mix of HW-Type and FW-Type mineralization. The flotation tests were completed at a grind of $P_{80=}$-75µm. The results from these tests have been tabulated in Table 13.63. Flotation was carried out at $P_{80=-}$75µm and the concentrates were subjected to a grind at 100%-37µm prior to intensive cyanidation.

Table 13.63 – 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 13.64.

Table 13.64 – 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 13.65.

Table 13.65 – 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)

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. Oxide from the Main zone material was sampled from TAL-5, TAL-6, TAL-9, TAL-27 and TAL-58 and HW-Type (Bear Creek) material from TAL-127, TAL-130 and TAL-148. 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}=-53µm. The results from these tests have been summarized in Table 13.66. The results were not as good as those from the previous flotation/cyanidation test work but the previous tests were done at finer grind, longer residence time, and on 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 13.67.

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 13.68. 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 13.69.

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 13.70 and the column leach tests in Table 13.71.

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 13.66 – 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 13.67 – 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 13.68 – 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 13.69 – 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 13.70 – 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 13.71 – 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)

13.3.19 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 of oxide, 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. Most of the material represented Oxide from the Main zone, however oxidized Bear Creek FW Type material was also represented in composites taken from T-02, T-03, T-08 and TA-10.

The results from the column leach test work are presented in Table 13.72. 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 13.73. The size fractions of the residues from the bottle roll leach are presented in Table 13.74. 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 13.72 – 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 13.73 – 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 13.74 – 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)

13.3.20 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 13.75. The measured and calculated head assays were in good agreement. The material from T-RC-1 through to T-RC-33 consist of oxide (Main Zone).

Table 13.76 presents the bottle roll leach results from the composite and individual drillhole samples. Table 13.77 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 13.75 – 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
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 13.76 – 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 13.77 – 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

13.3.21 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 Oxide and FW Type samples respectively. The head assays for the samples are tabulated in Table 13.78.

Table 13.78 – 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 13.79.

Table 13.79 – Hazen – Screened Feed Bottle Roll Leach Results

Composite	Size (µm)	Residual		Recovery	
		Gold g/t	Silver g/t	Gold %	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

Composite	Size (µm)	Residual		Recovery	
		Gold g/t	Silver g/t	Gold %	Silver %
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 FW Type samples behaved similarly for the gold solubilisation, but the oxide sample had better silver solubilisation.

13.3.22 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 Oxide material taken from the 15 to 20 m intervals of drill core TA-3 (Bear Creek FW-zone), and deeper primary mineralized material from the 100 to 150 m intervals of drillhole TA-4 (FW-Type from Bear Creek). 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 13.80. Results from the pulverized material can be found in Table 13.81.

Table 13.80 – 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 13.81 – 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 ground to a finer sizes, which were subjected to agitated leach, had higher recoveries. Drillhole TA-3 is described as representing the Oxide (Bear Creek zone) and FW-Type (Bear Creek zone) material and drillhole TA-4 the unoxidized material. 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 indicated that heap leach may not be a feasible option due to low column leach precious metal recoveries.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

Tetra Tech completed a resource estimation of the Talapoosa deposit in 2013. Timberline updated the NI43-101 Technical Report and re-issued the resource with an effective date of the resource as March 24, 2015. Mr. Todd McCracken, P. Geo, the QP who completed the resource estimation with Tetra Tech, remains as the QP with WSP.

Historically, the Talapoosa deposit is made up of four different resource zones: Bear Creek, Main zone, East Hill zone and Dyke Adit zone. The Bear Creek zone has been subdivided into HW and FW zones.

14.2 DATABASE

Timberline maintains the borehole database in MineSight® containing header, survey, assays and lithology tables. A copy of these 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 was for all surface boreholes on the Property. There are 40,723 gold assays and 36,601 silver assays within the database (Table 14.1).

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/st)	40723	-2	5.389	-0.108	0.478
Ag (oz/st)	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 with an absent field.

The database has all significant data, and each sample interval is assigned an integer code representation that reflects the quality of that particular assay. 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	st/ft^3	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 dataset. At a minimum, 2% of the dataset should have specific gravity measurements. Currently, the specific gravity dataset 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 HW Vein	303966.06	305833.46	1712178.51	1713217.08	4476.92	5370.38	104,010,965.5
Bear Creek FW 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 HW Zone	303591.27	305869.35	1712177.83	1713357.19	4462.62	5414.89	282,393,020.0
Bear Creek FW 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

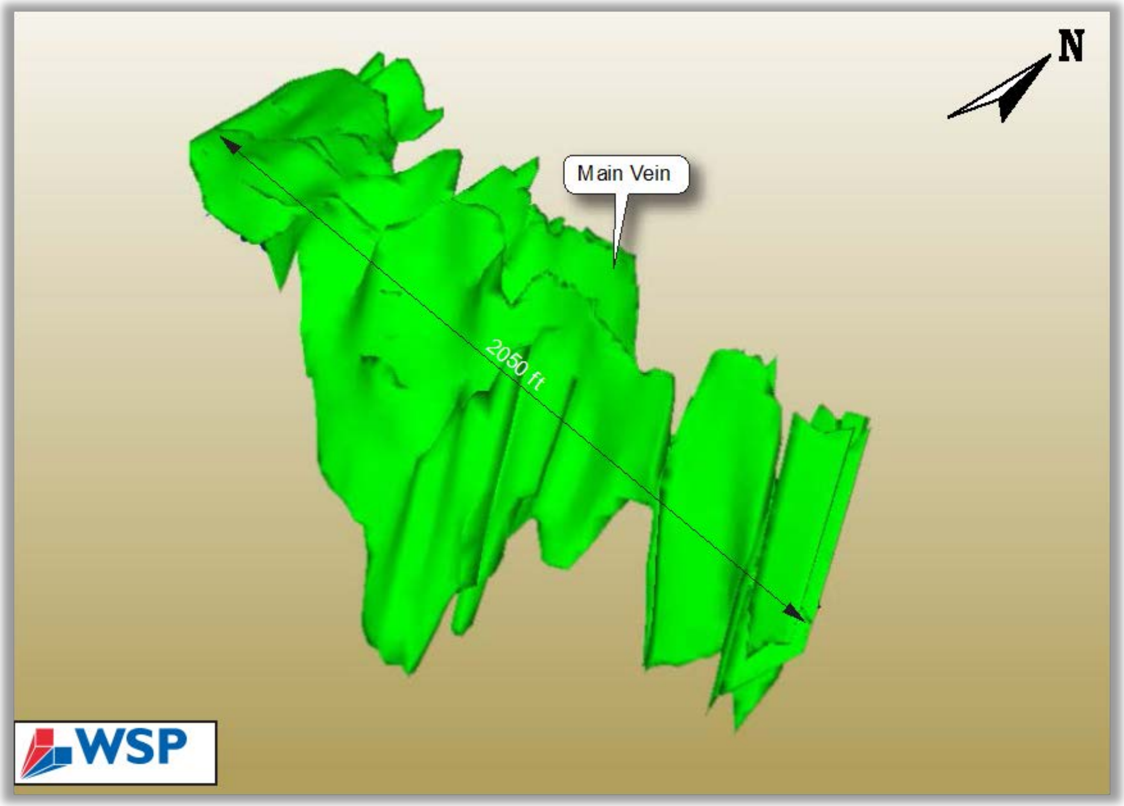


Figure 14.3 – Oblique View Bear Creek FW Vein

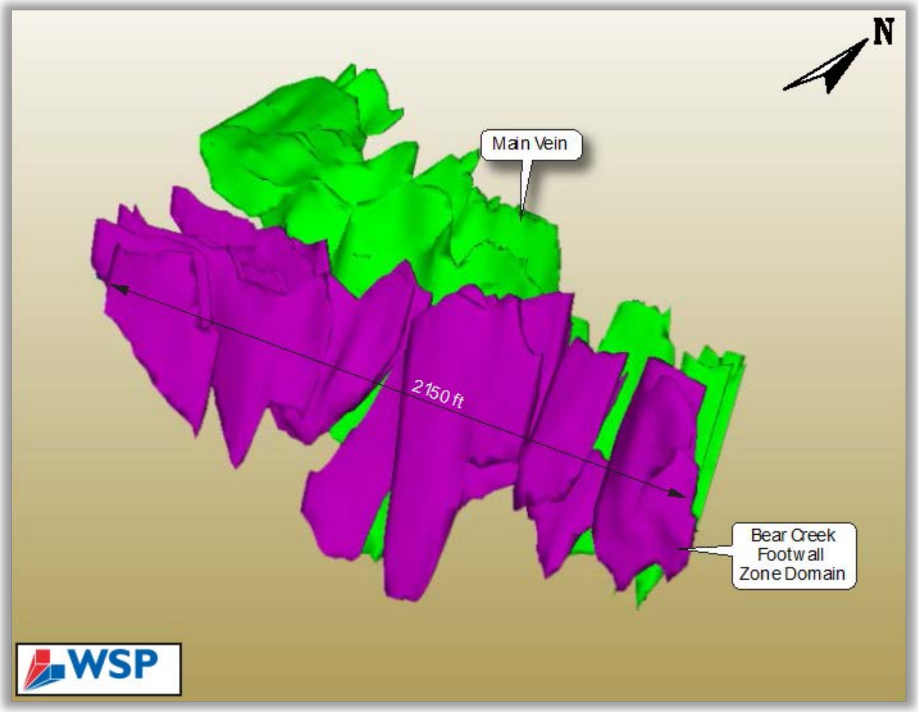


Figure 14.4 – Oblique View Bear Creek HW Vein

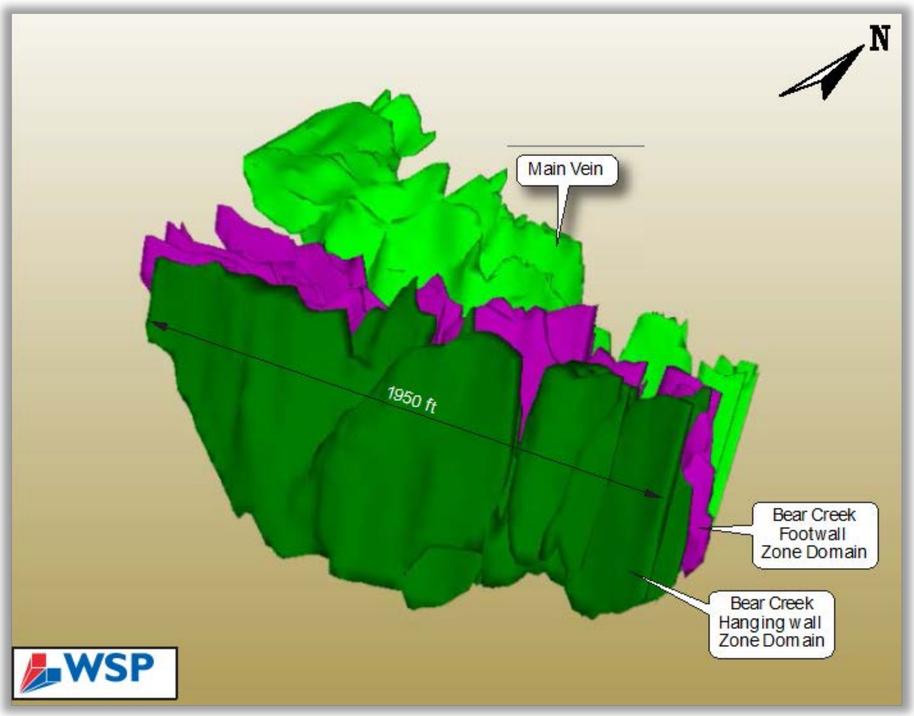


Figure 14.5 – Oblique View Dyke Adit Vein

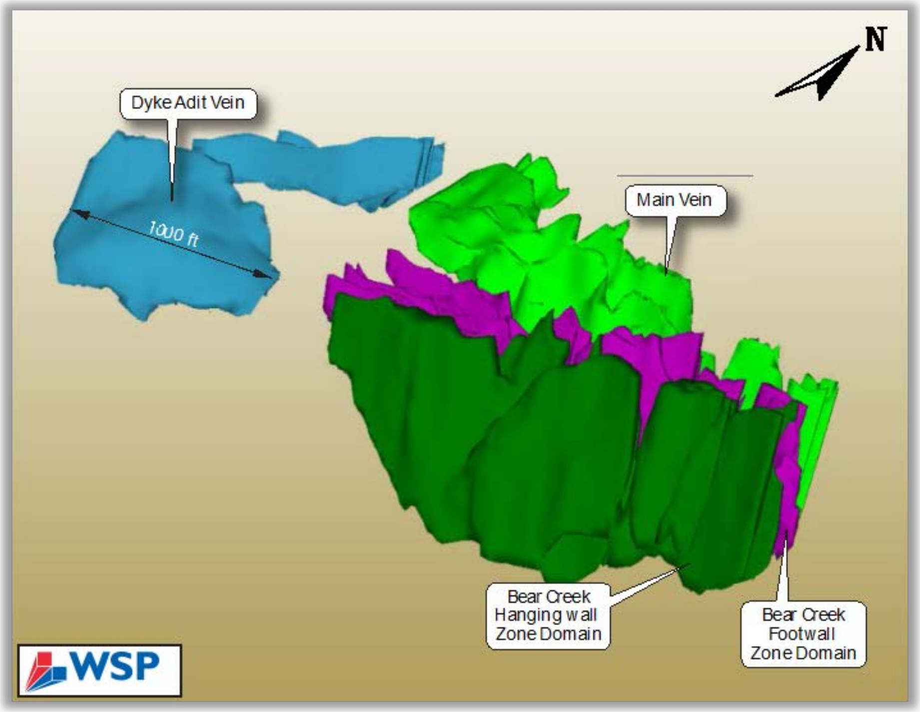


Figure 14.6 – Oblique View East Hill Vein

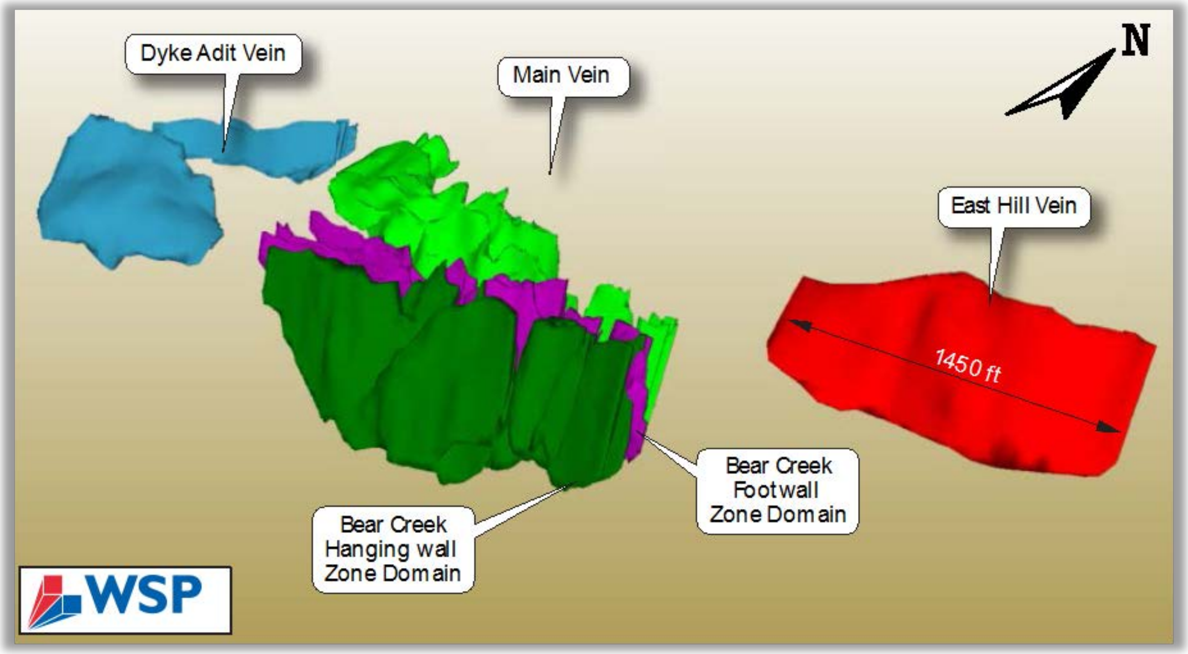


Figure 14.7 –Oblique View Main Zone

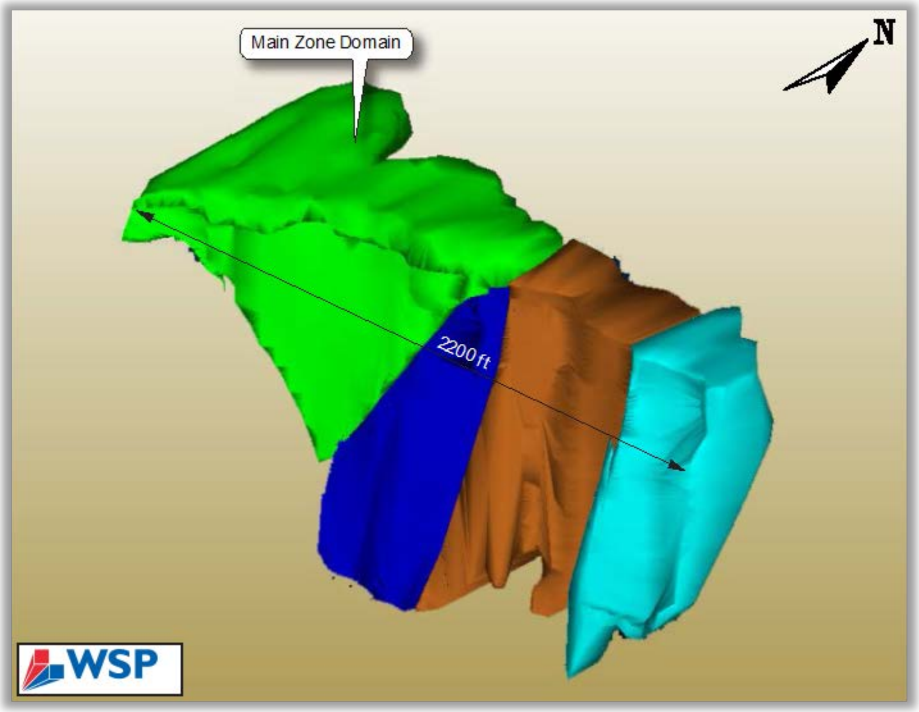


Figure 14.8 – Oblique View Bear Creek FW Zone

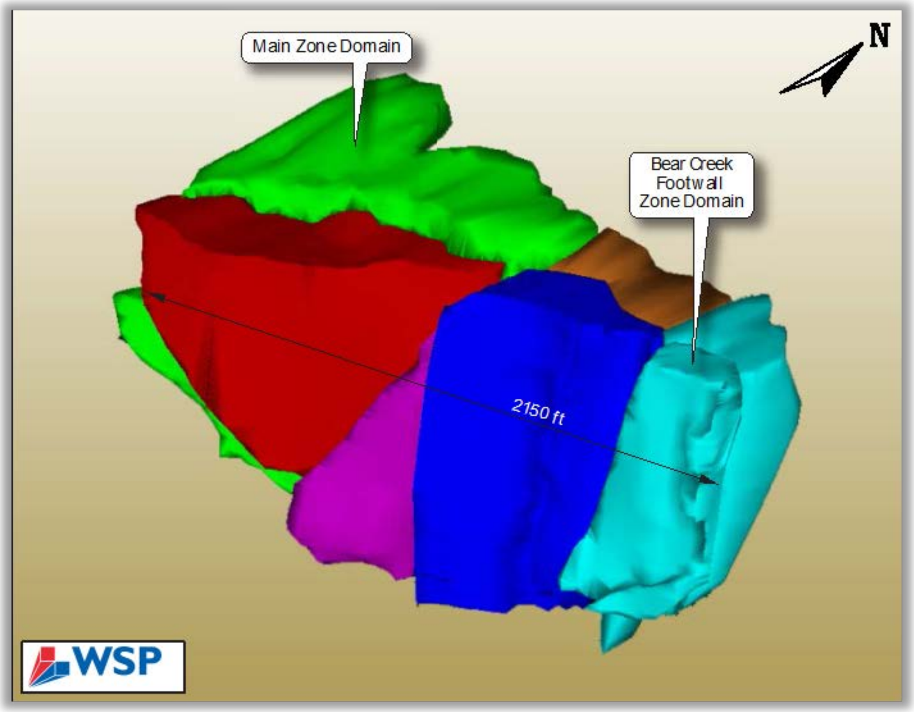


Figure 14.9 – Oblique View Bear Creek HW Zone

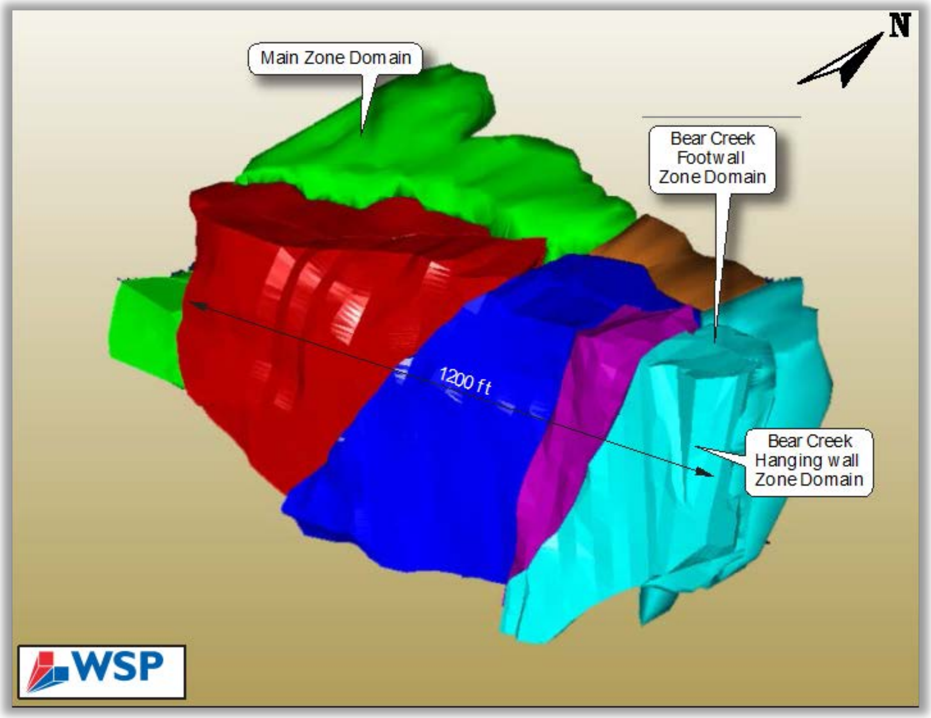


Figure 14.10 – Oblique View Dyke Adit Zone

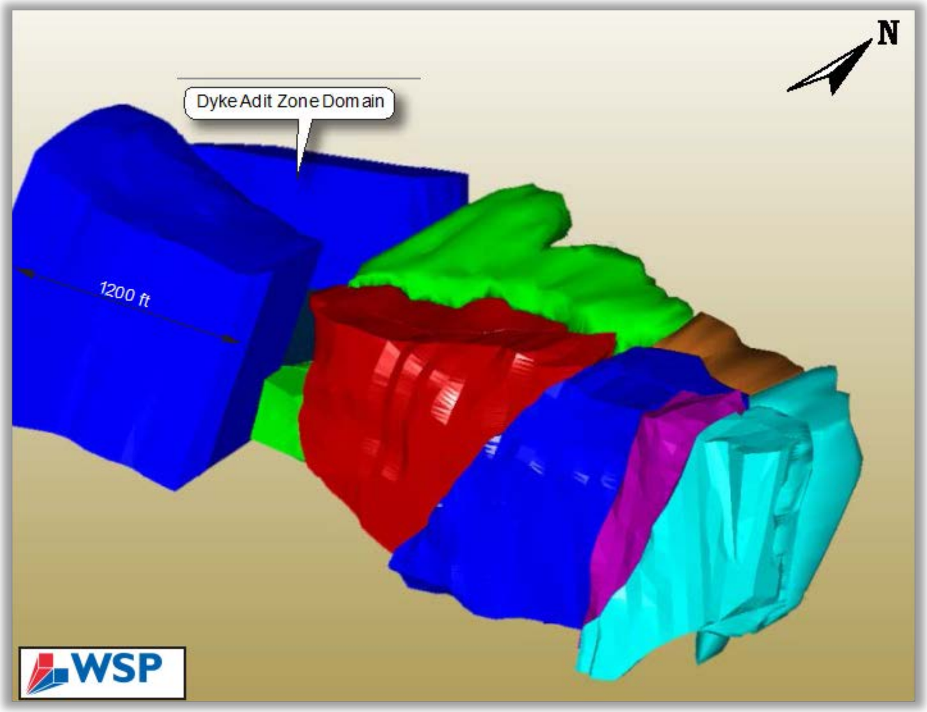
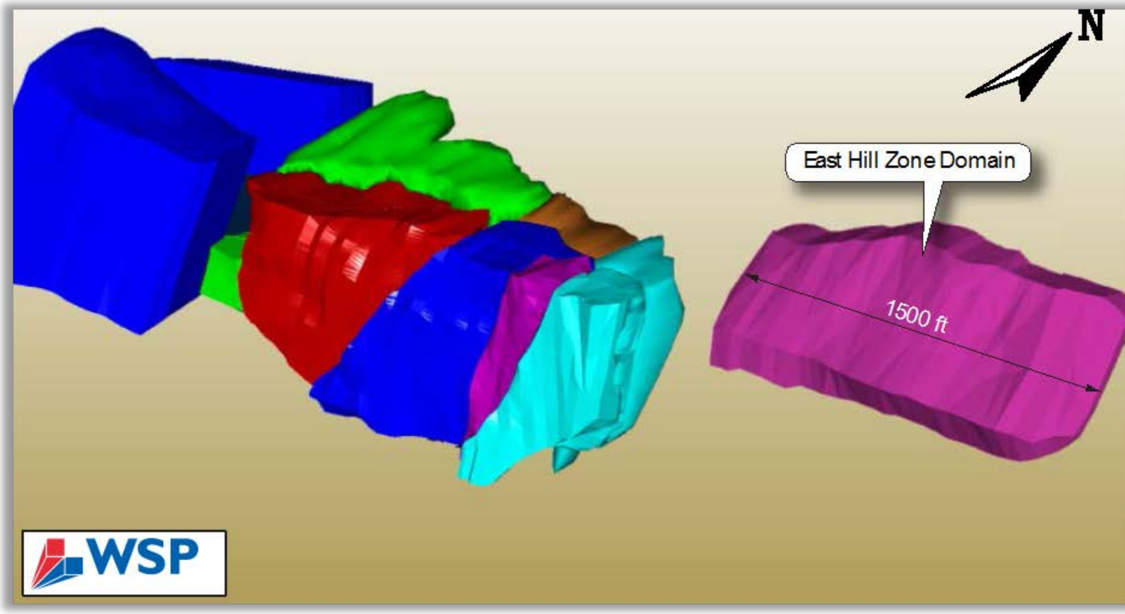


Figure 14.11 – Oblique View East Hill Zone



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 FW 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 HW 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
	Ag	1,641	0.0022	41.7560	0.4205	1.4843

Zone	Field	N Samples	Minimum	Maximum	Mean	Standard Deviation
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 FW 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 HW 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 HW Vein Gold Histogram Plot

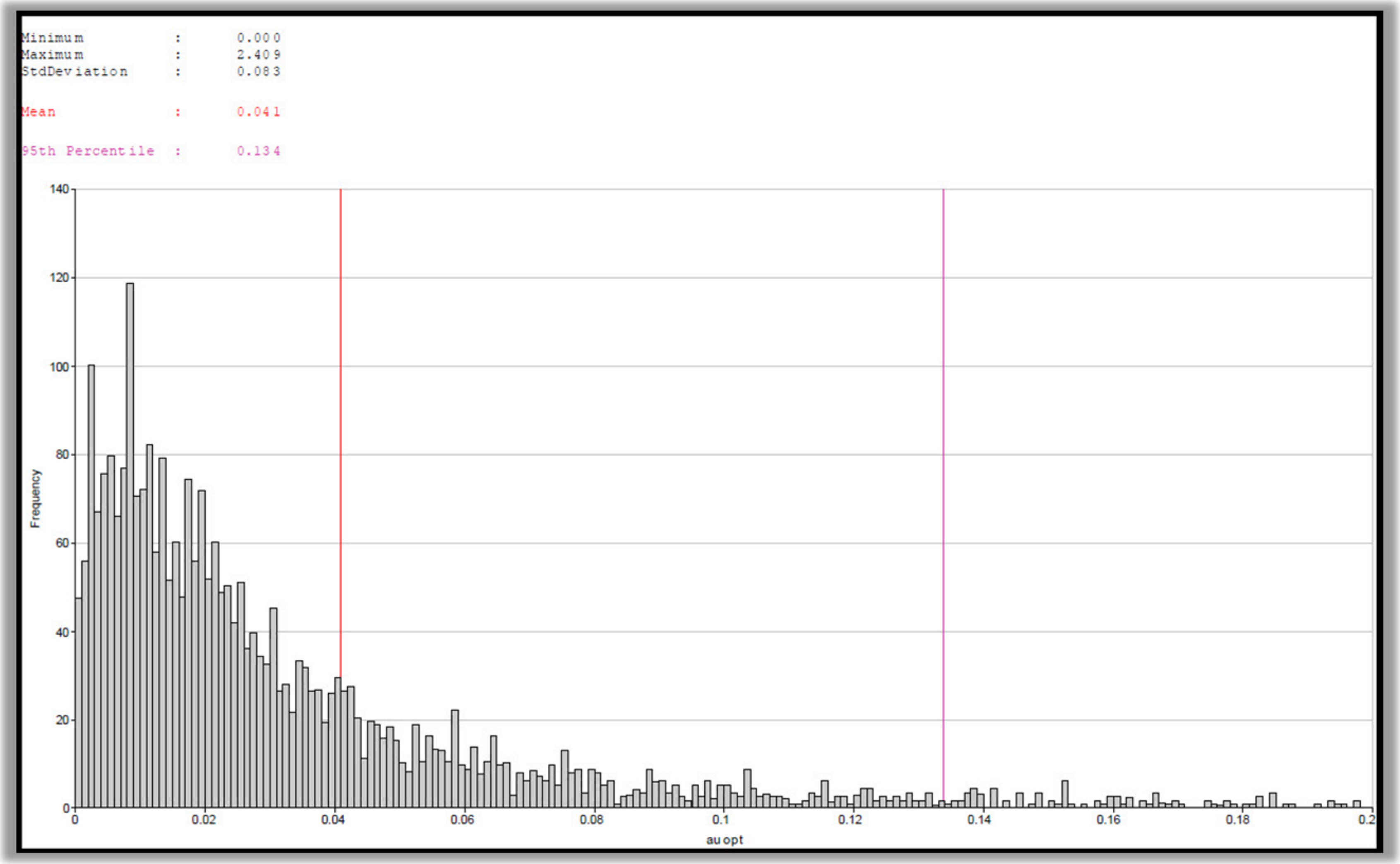


Figure 14.13 – Bear Creek FW Vein Gold Histogram Plot

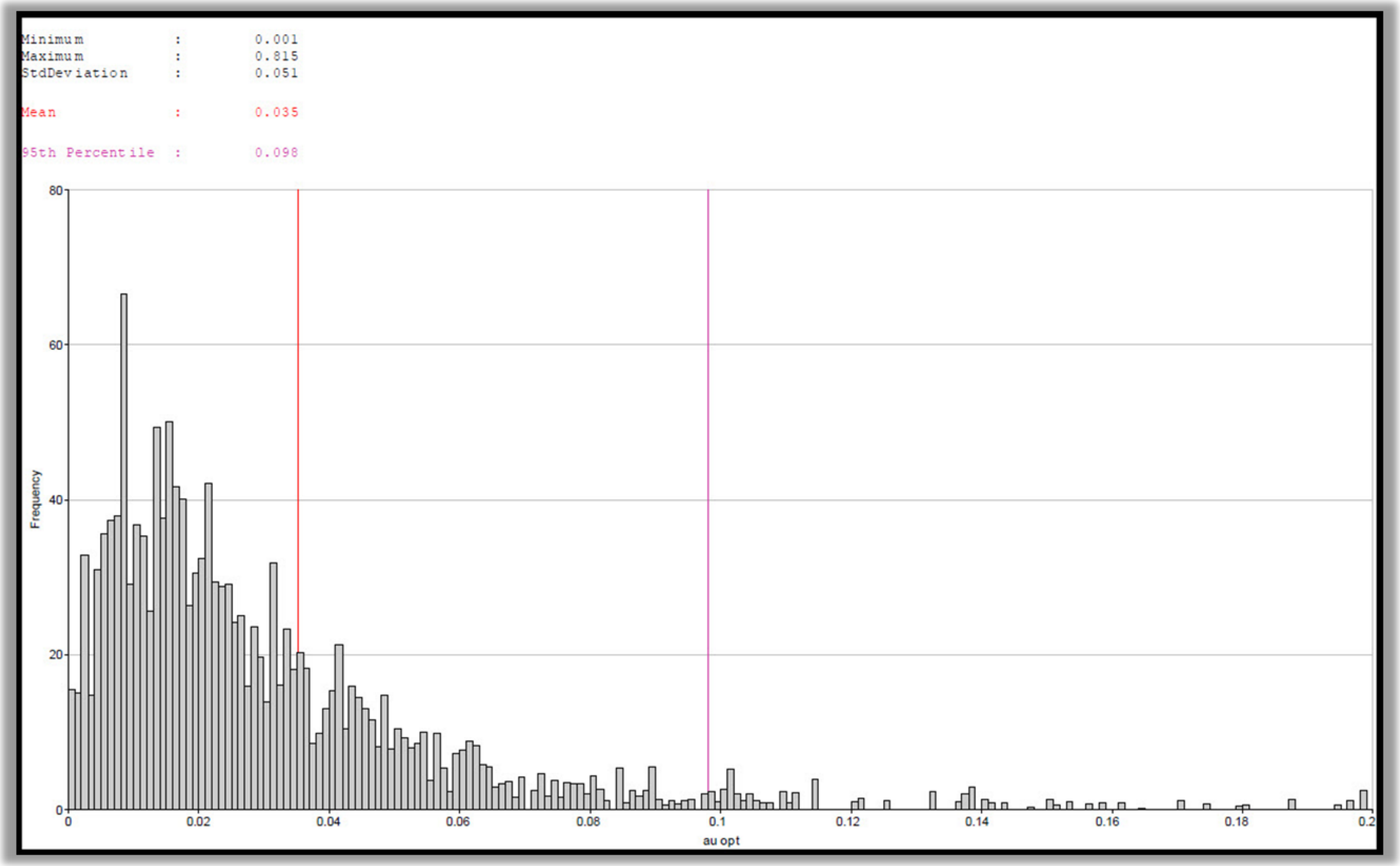


Figure 14.14 – Main Vein Gold Histogram Plot

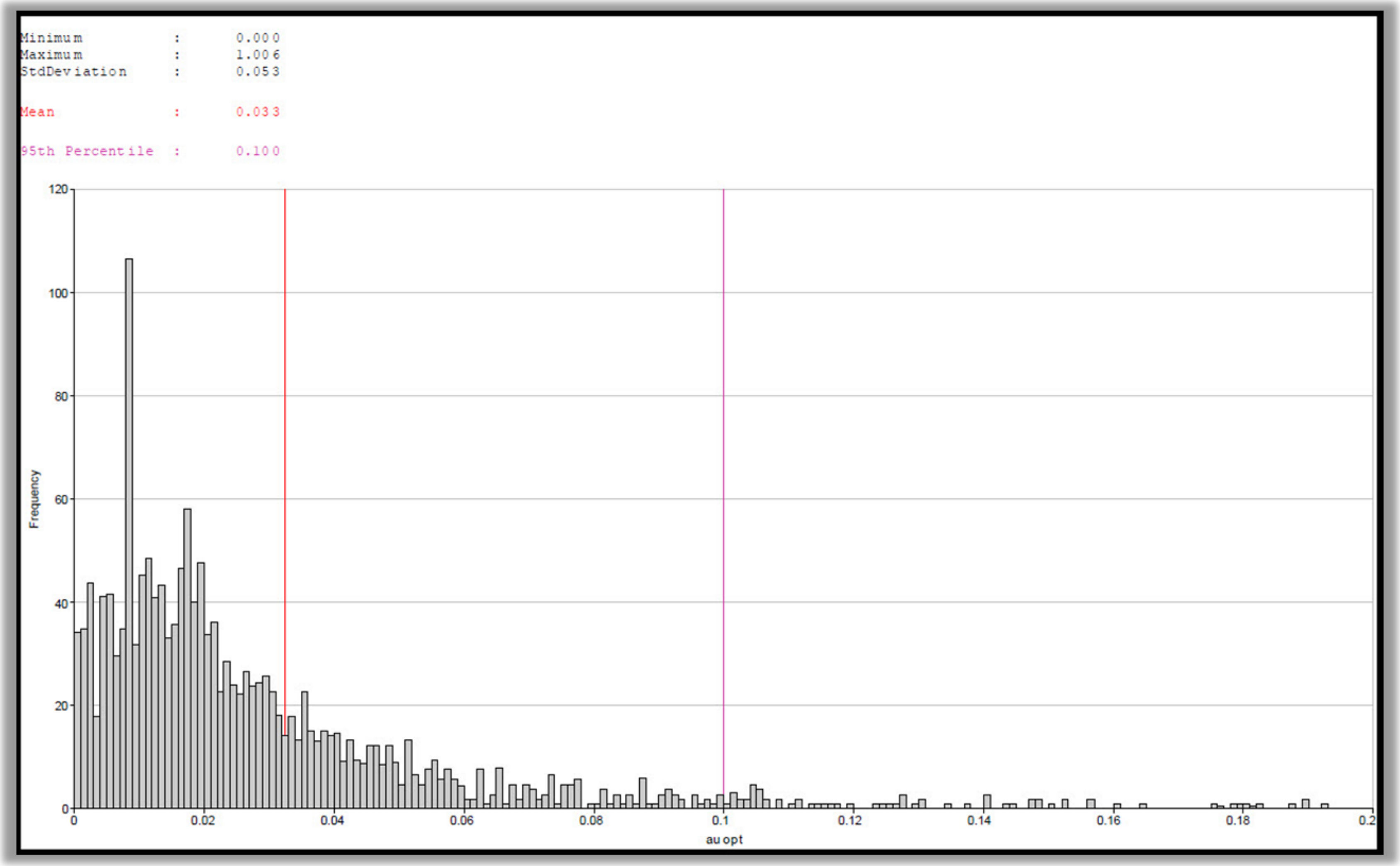


Figure 14.15 – Dyke Adit Vein Gold Histogram Plot

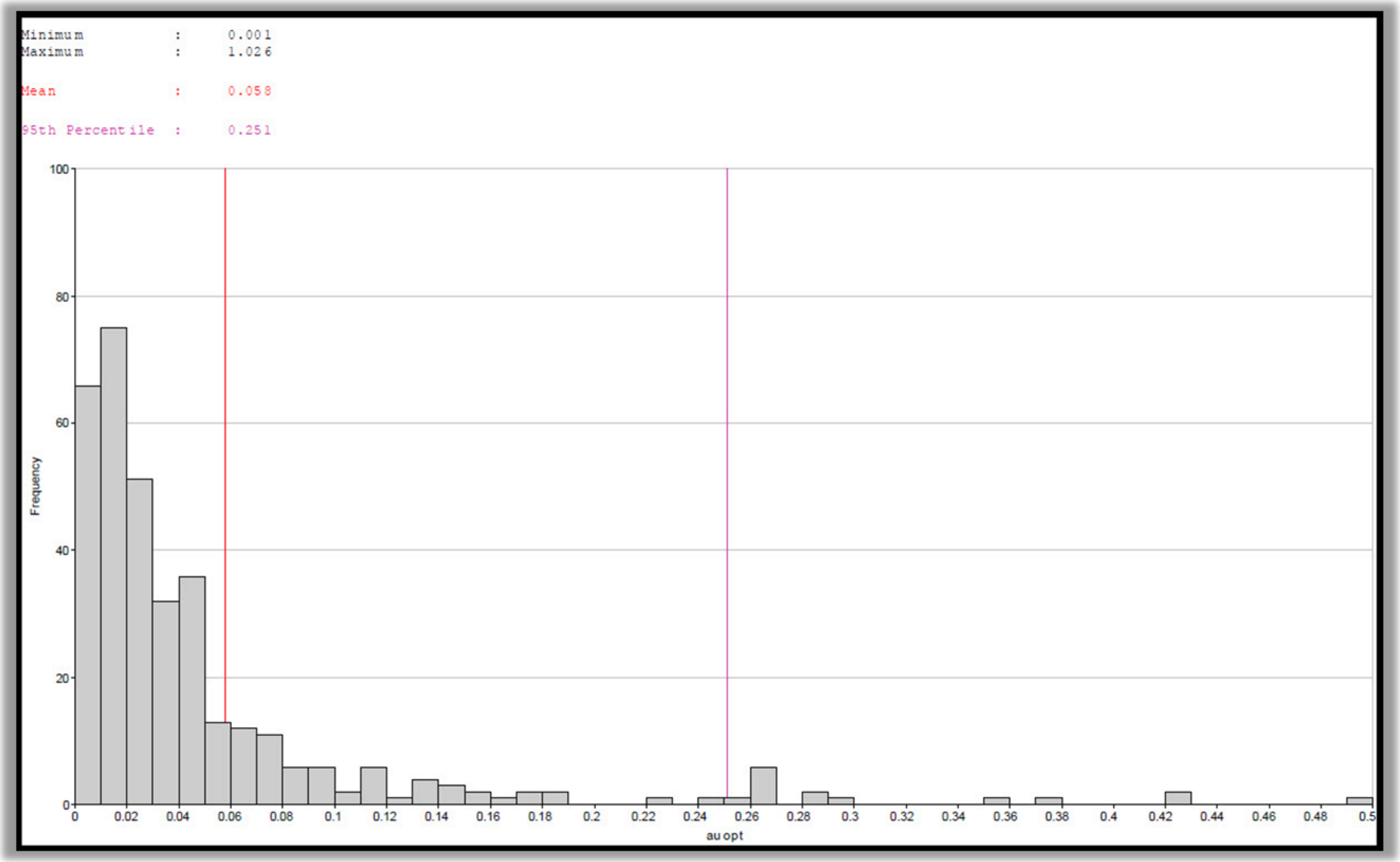


Figure 14.16 – East Hill Vein Gold Histogram Plot

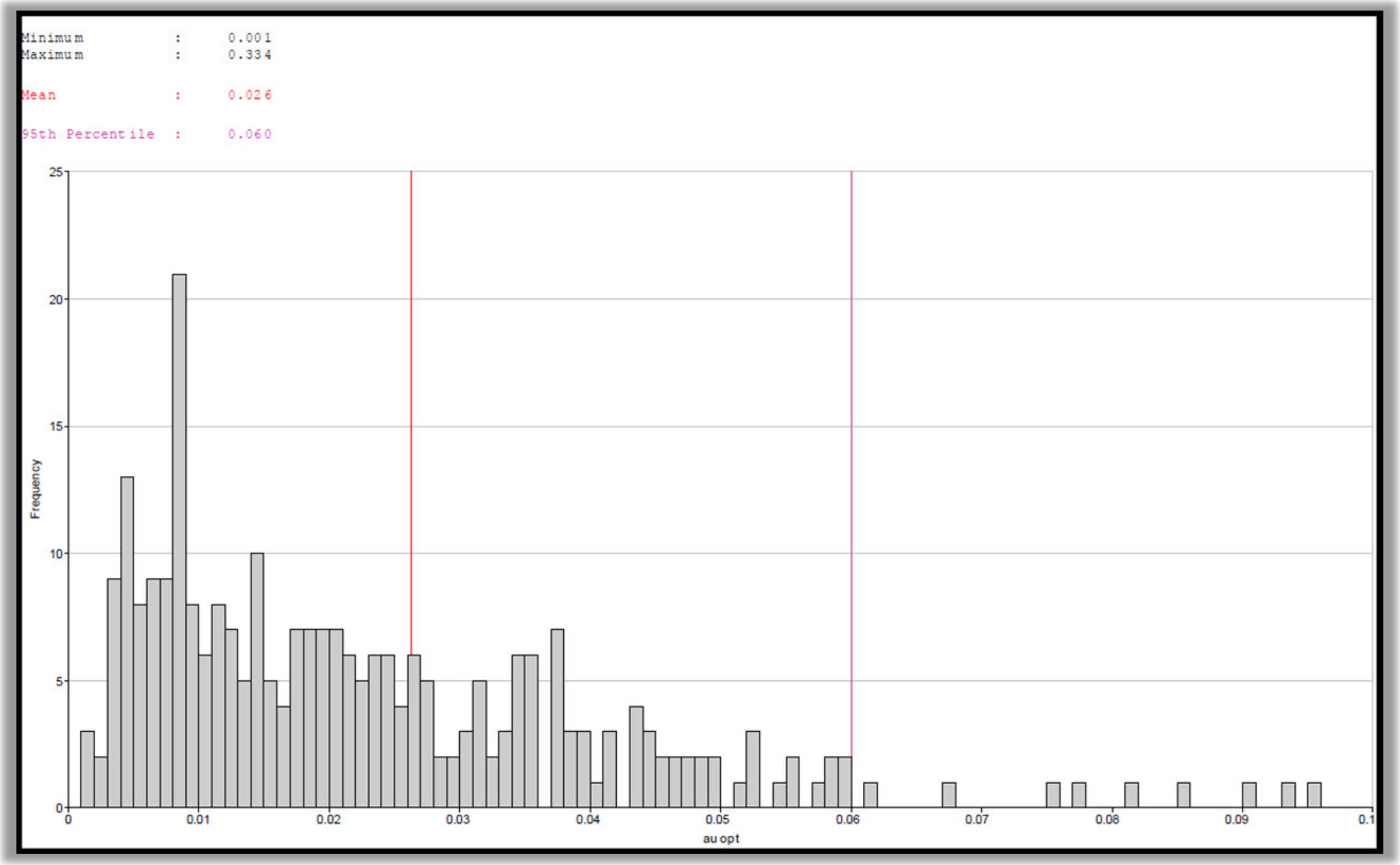


Figure 14.17 – Bear Creek HW Zone Histogram Plot

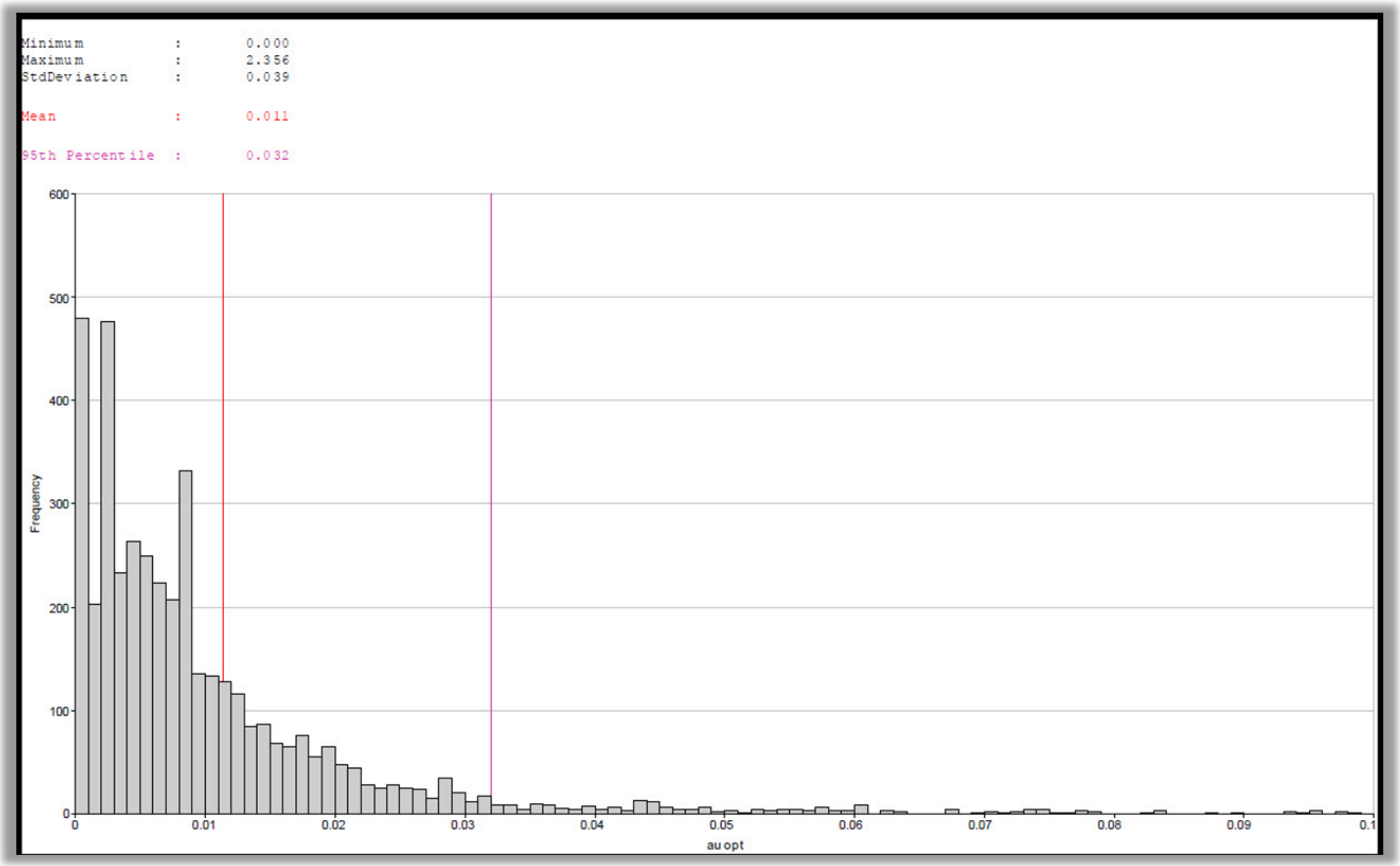


Figure 14.18 – Bear Creek FW Zone Gold Histogram Plot

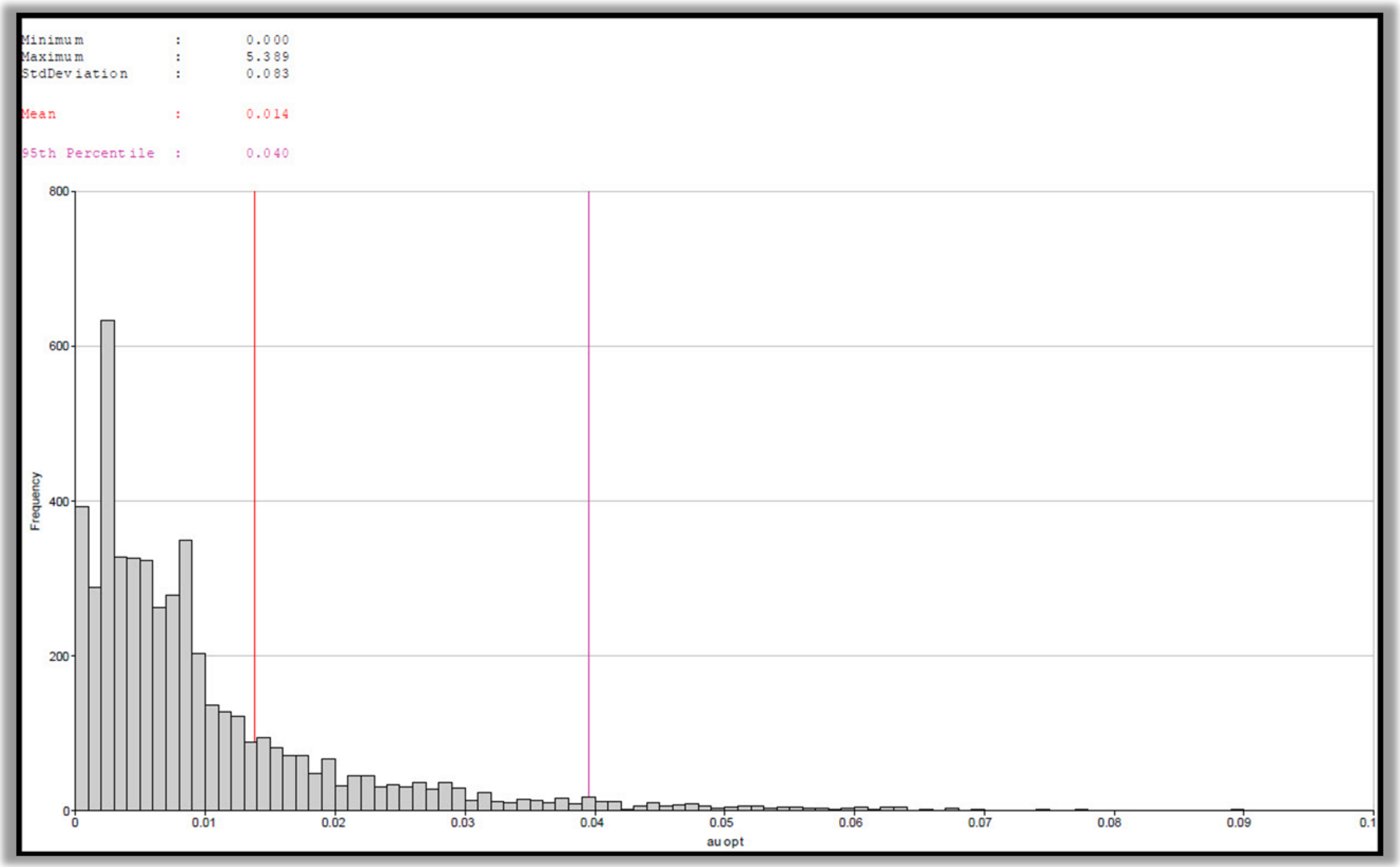


Figure 14.19 – Main Zone Gold Histogram Plot

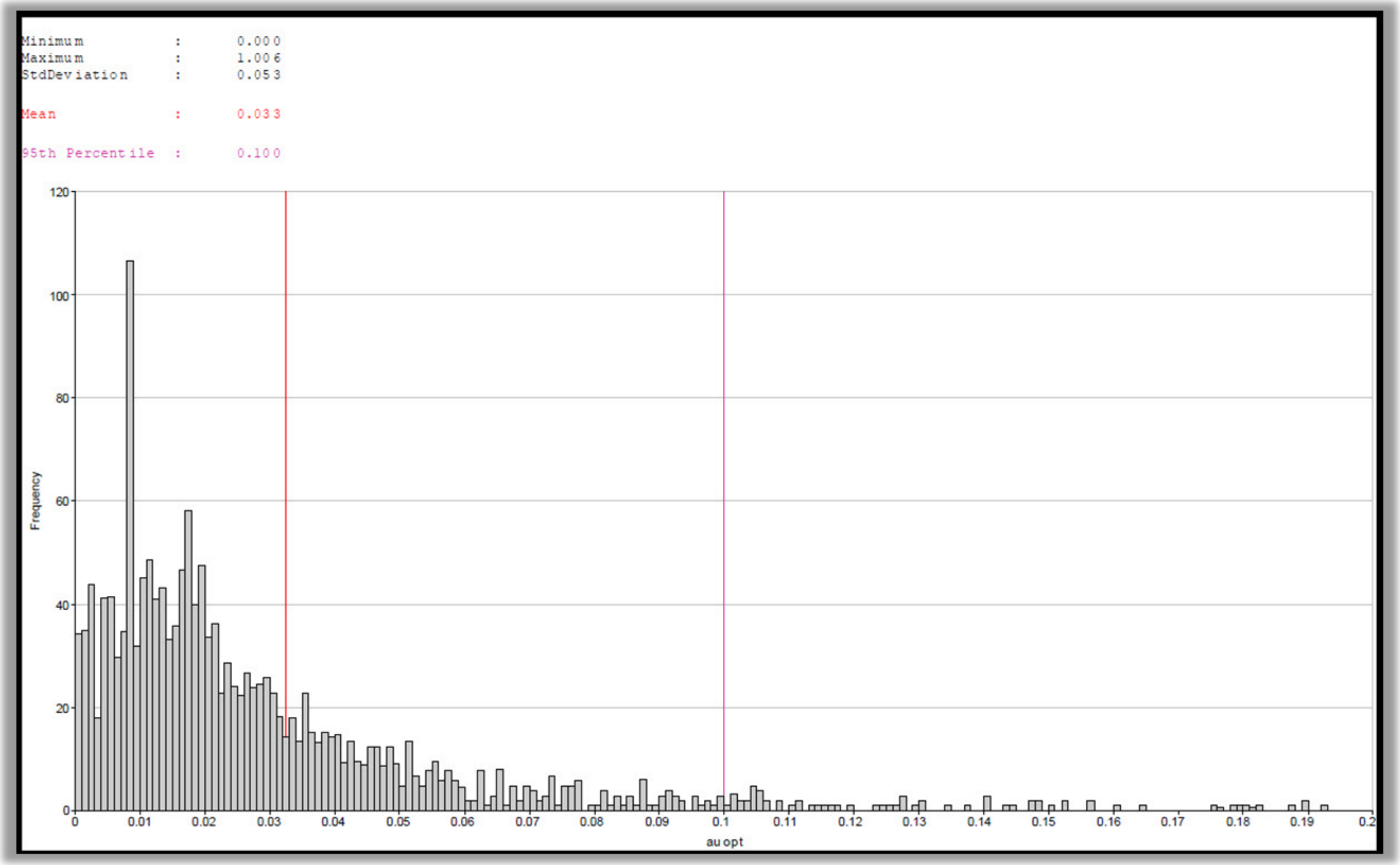


Figure 14.20 – Dyke Adit Zone Gold Histogram Plot

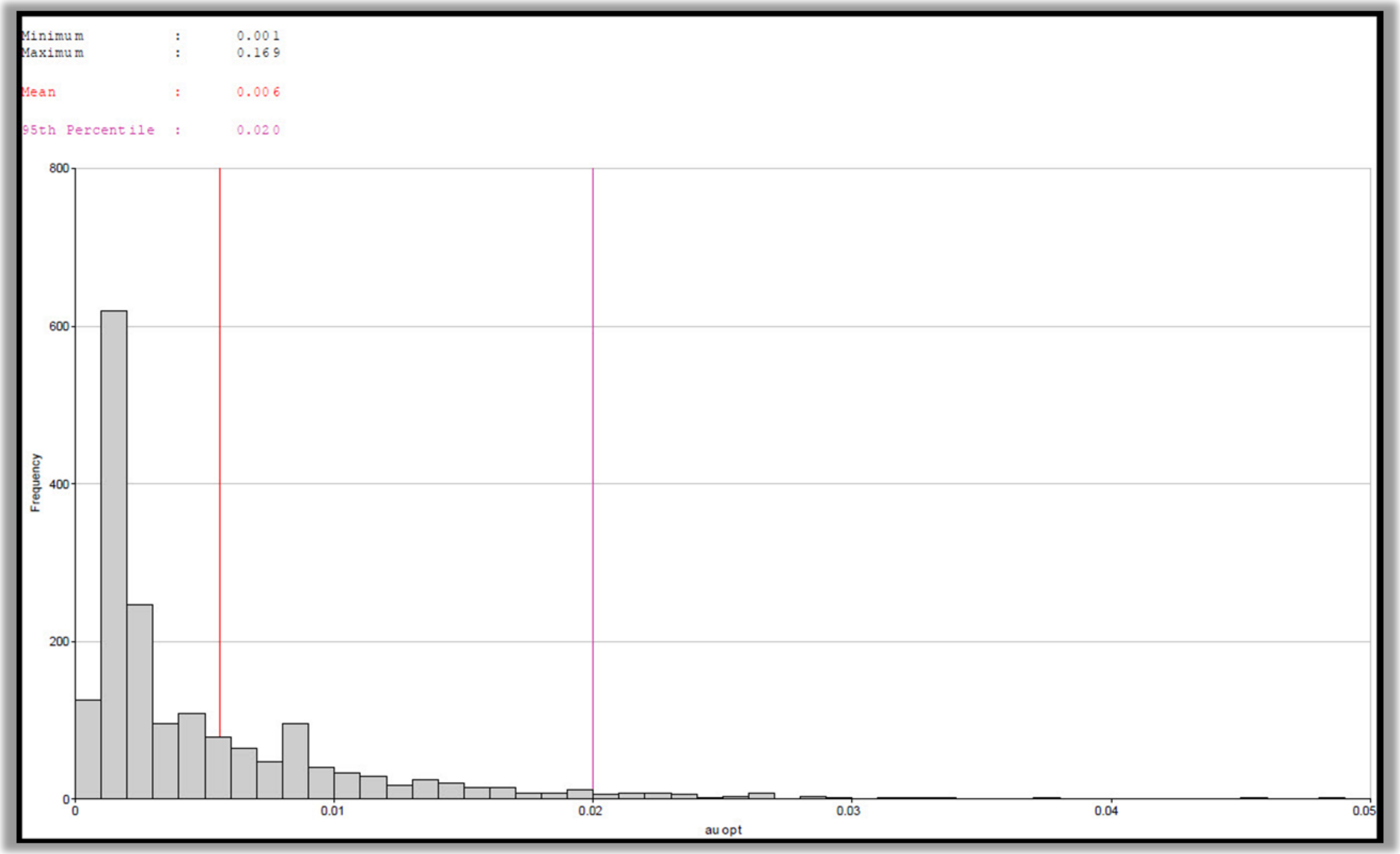
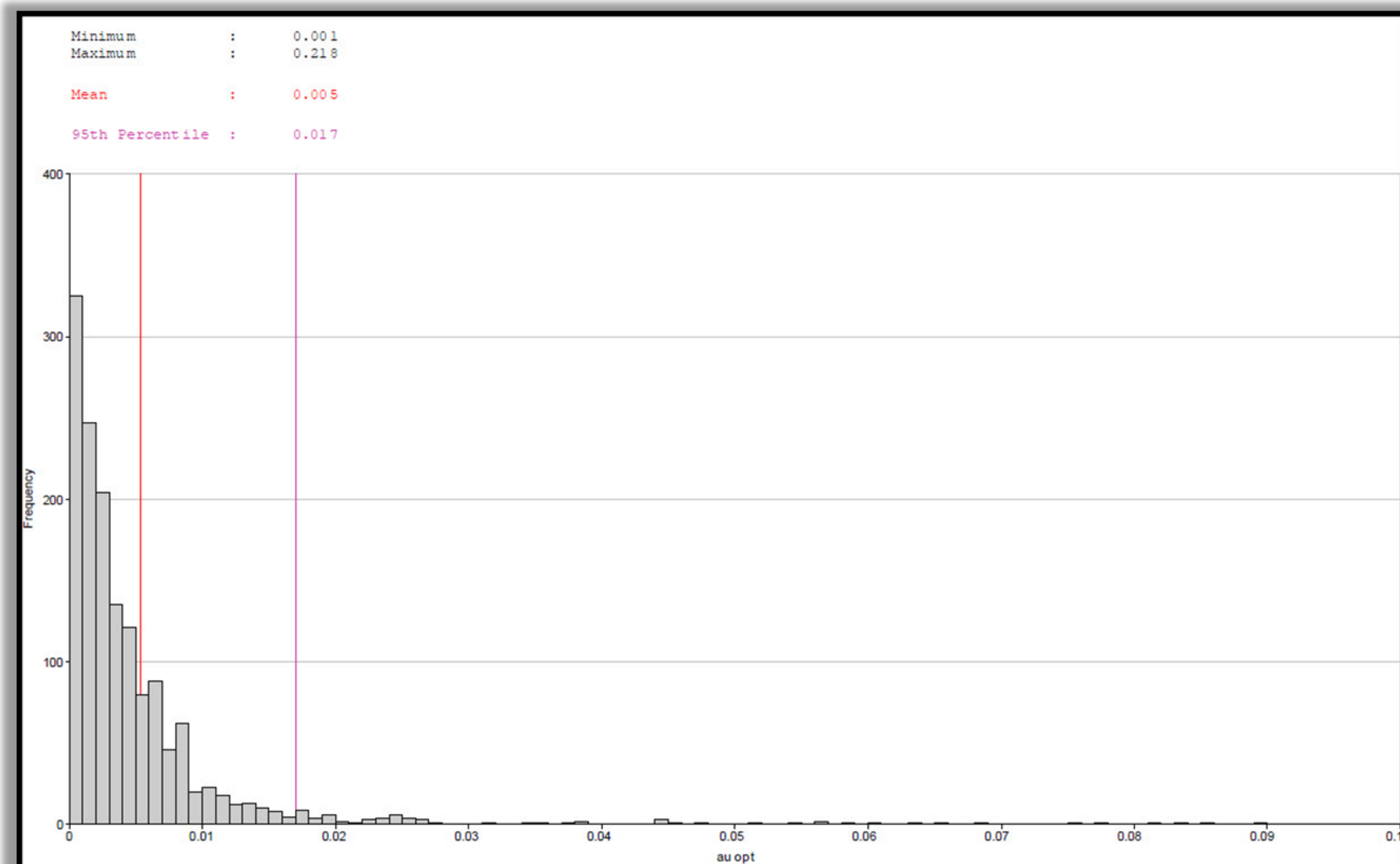


Figure 14.21 - East Hill Zone Gold Histogram Plot



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 dataset.

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
FW 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
HW 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
FW 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
HW 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 COMPOSITING

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
FW 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
HWVein	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
FW 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
HW 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 Figures 14.22 to 14.41 depict 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 HW Vein Gold Variogram

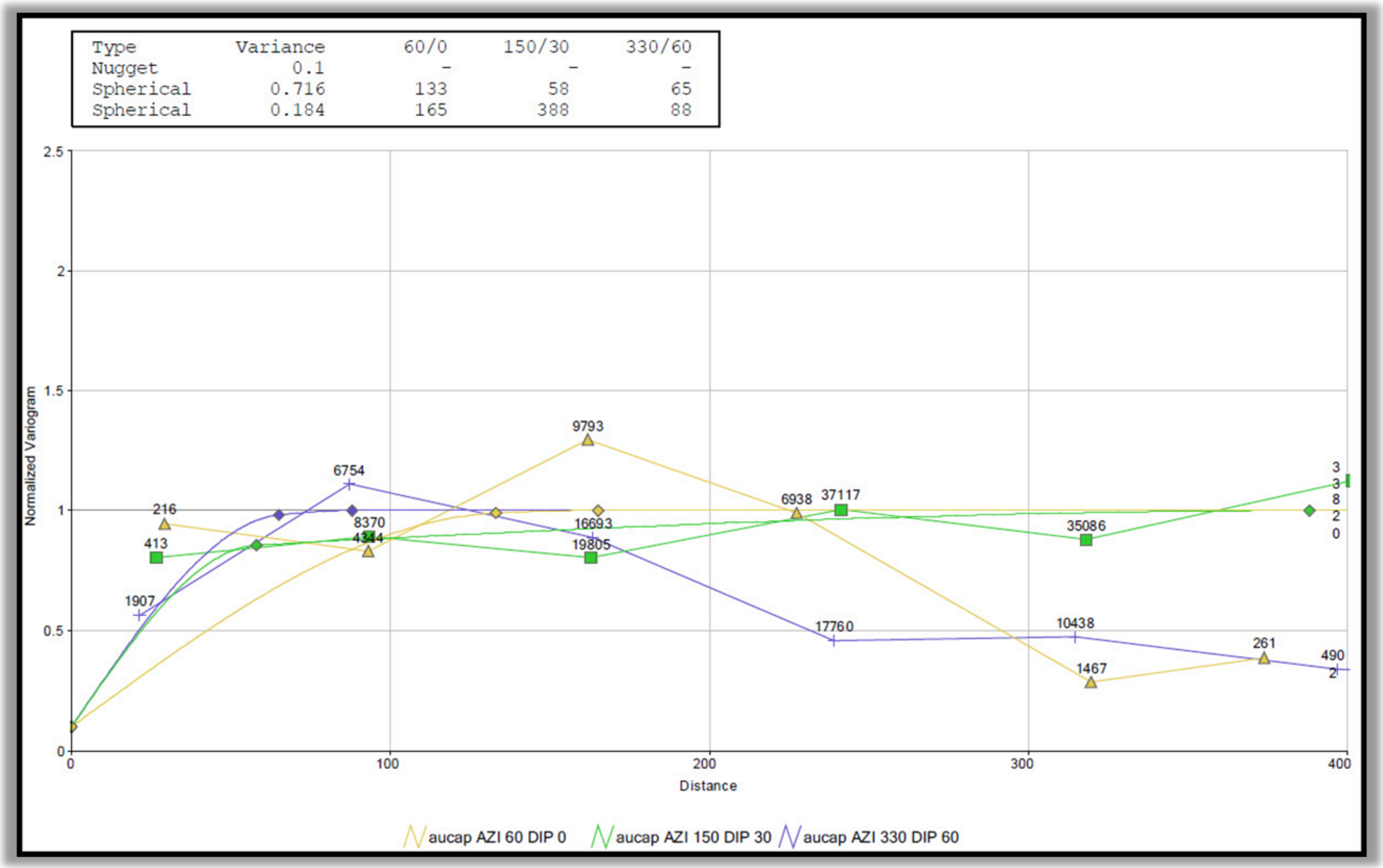


Figure 14.23 – Bear Creek HW Vein Silver Variogram

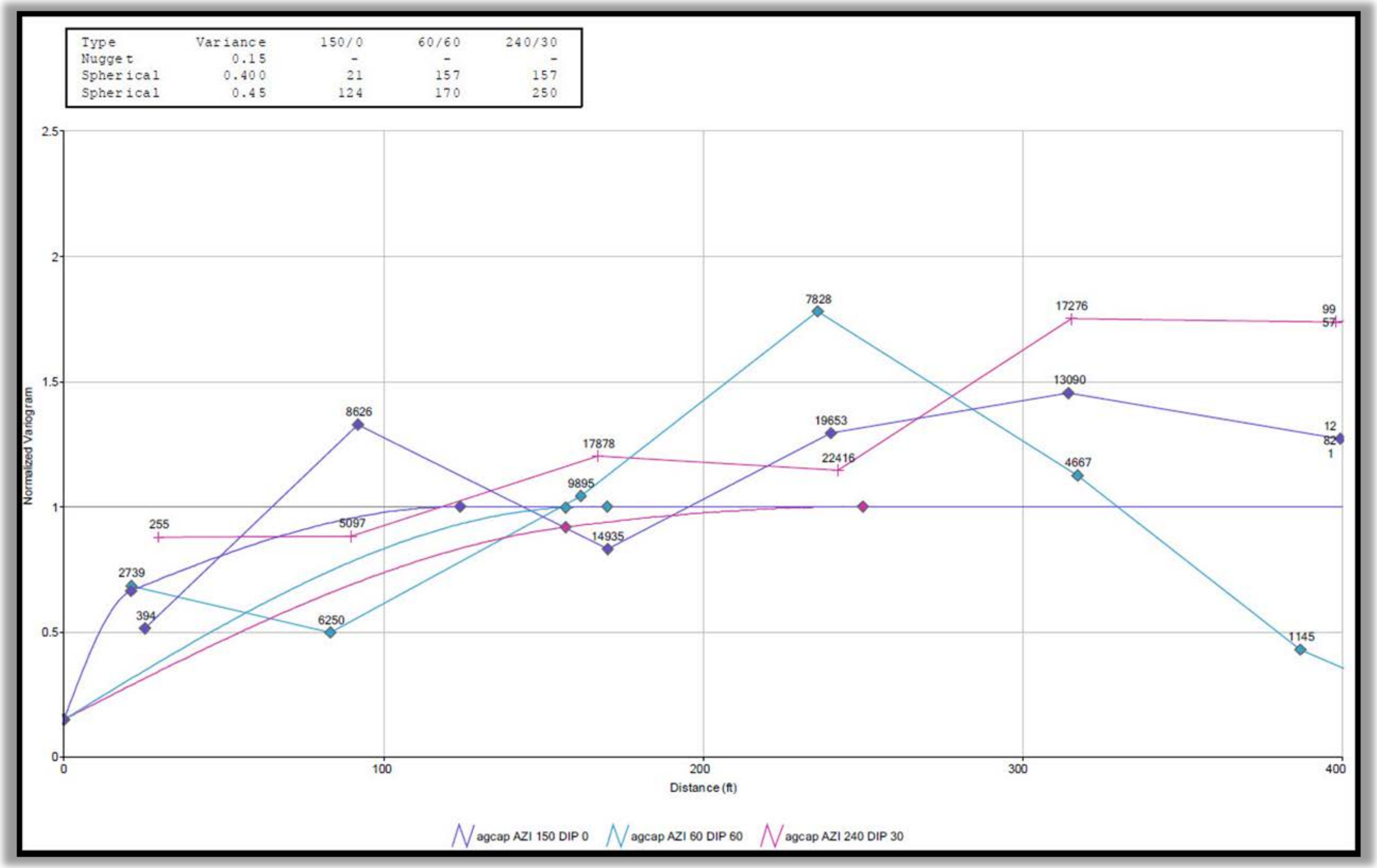


Figure 14.24 – Bear Creek FW Vein Gold Variogram

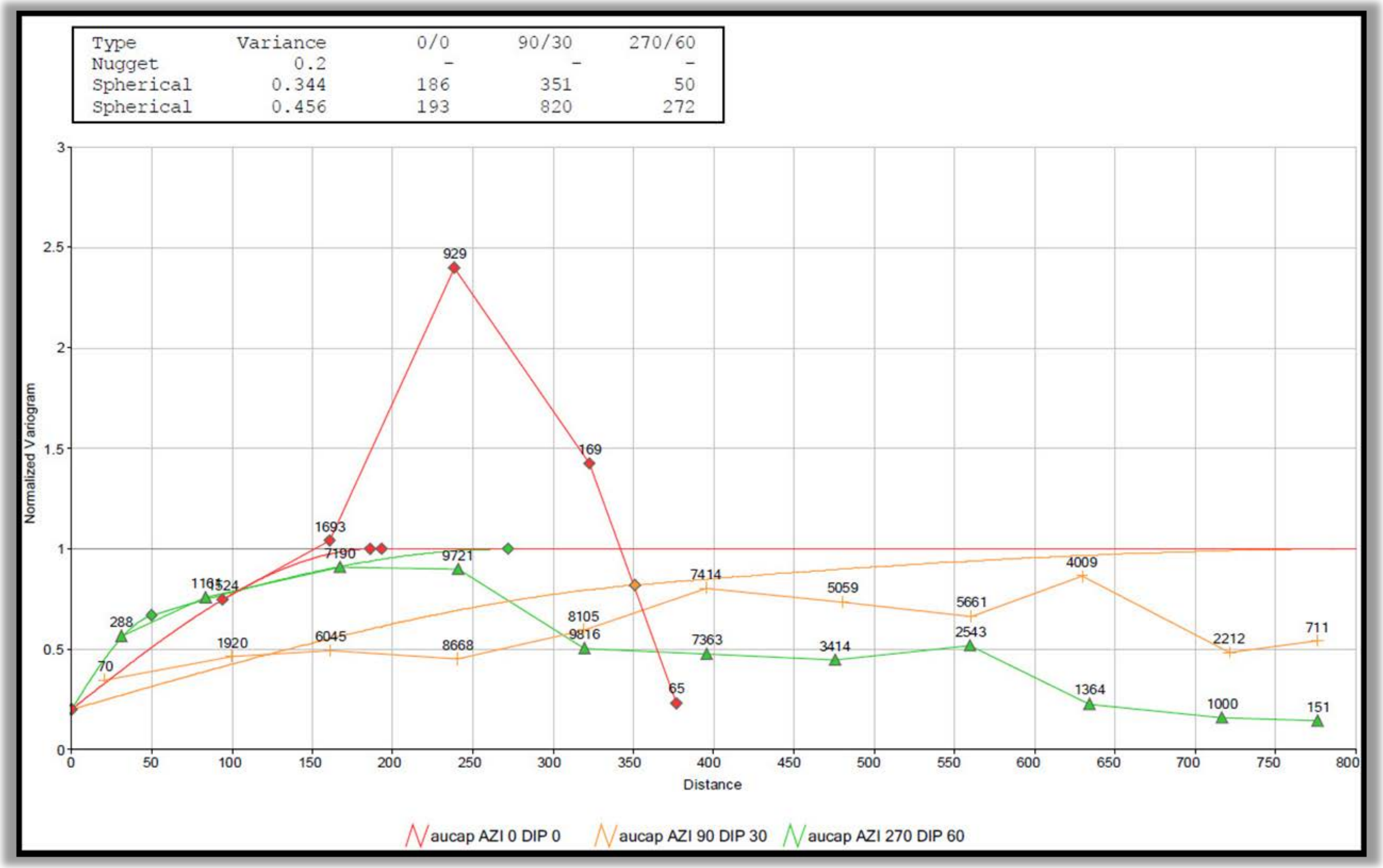


Figure 14.25 – Bear Creek FW Vein Silver Variogram

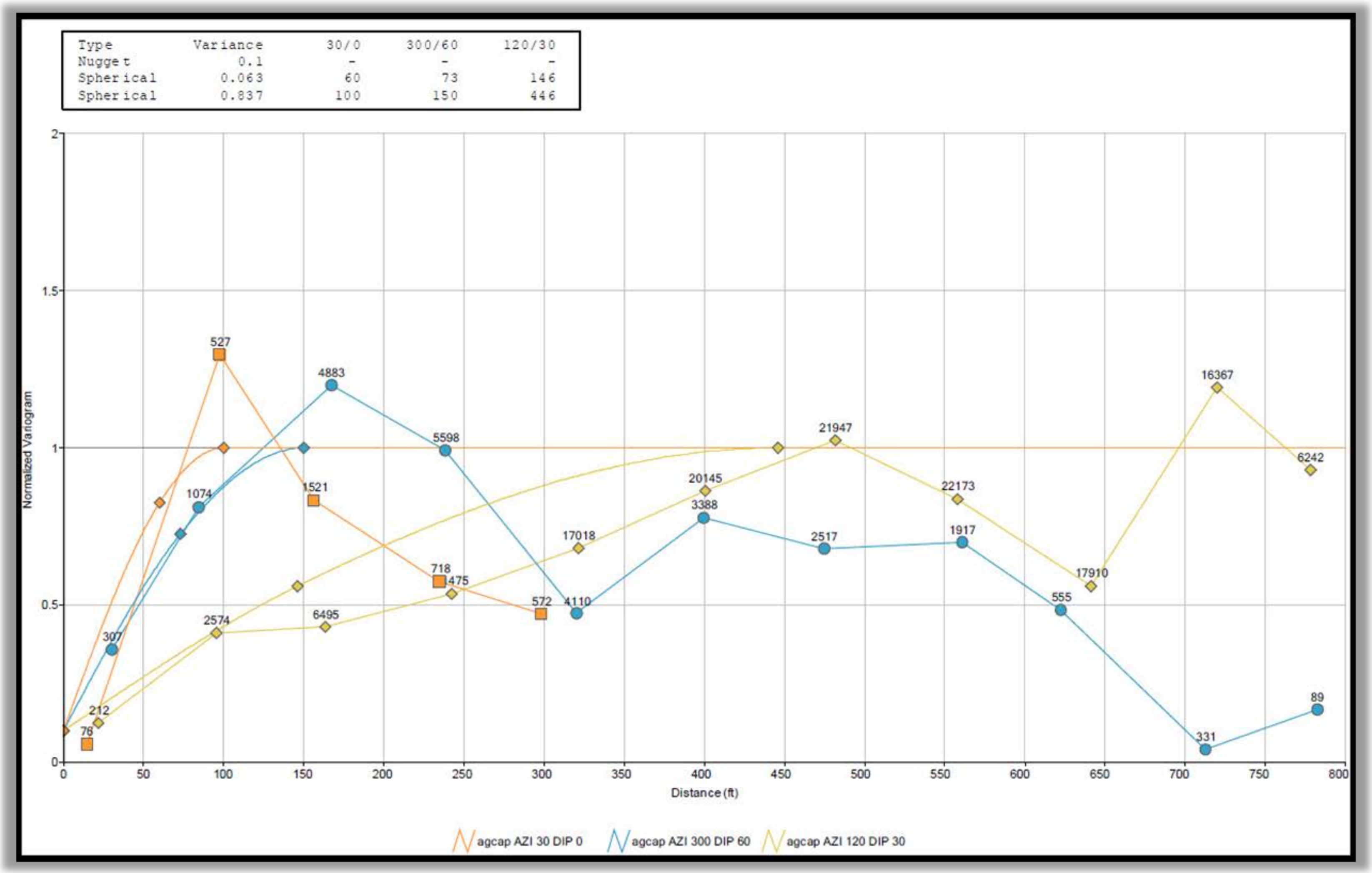


Figure 14.26 – Main Vein Gold Variogram

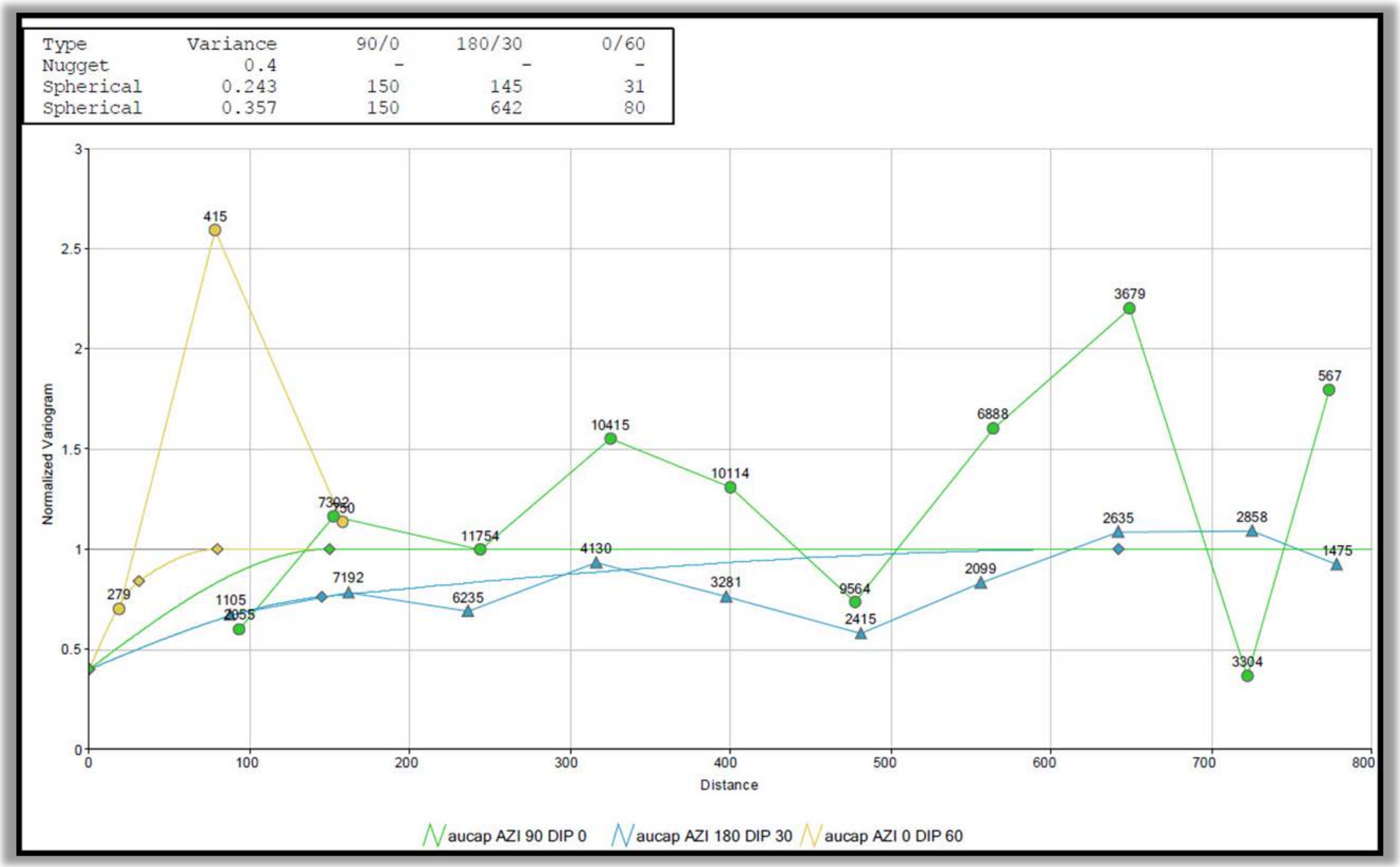


Figure 14.27 – Main Vein Silver Variogram

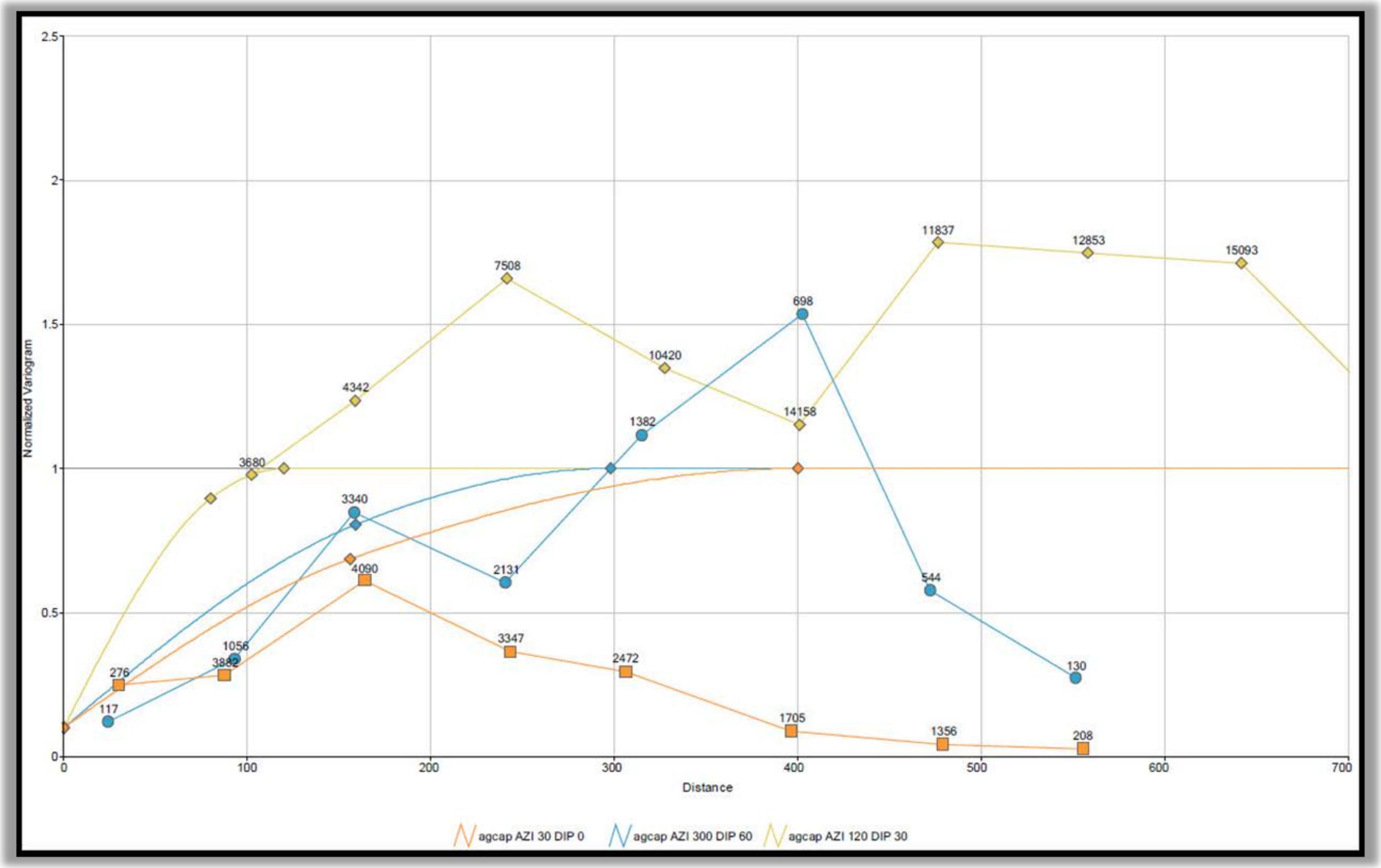


Figure 14.28 – Dyke Adit Vein Gold Variogram

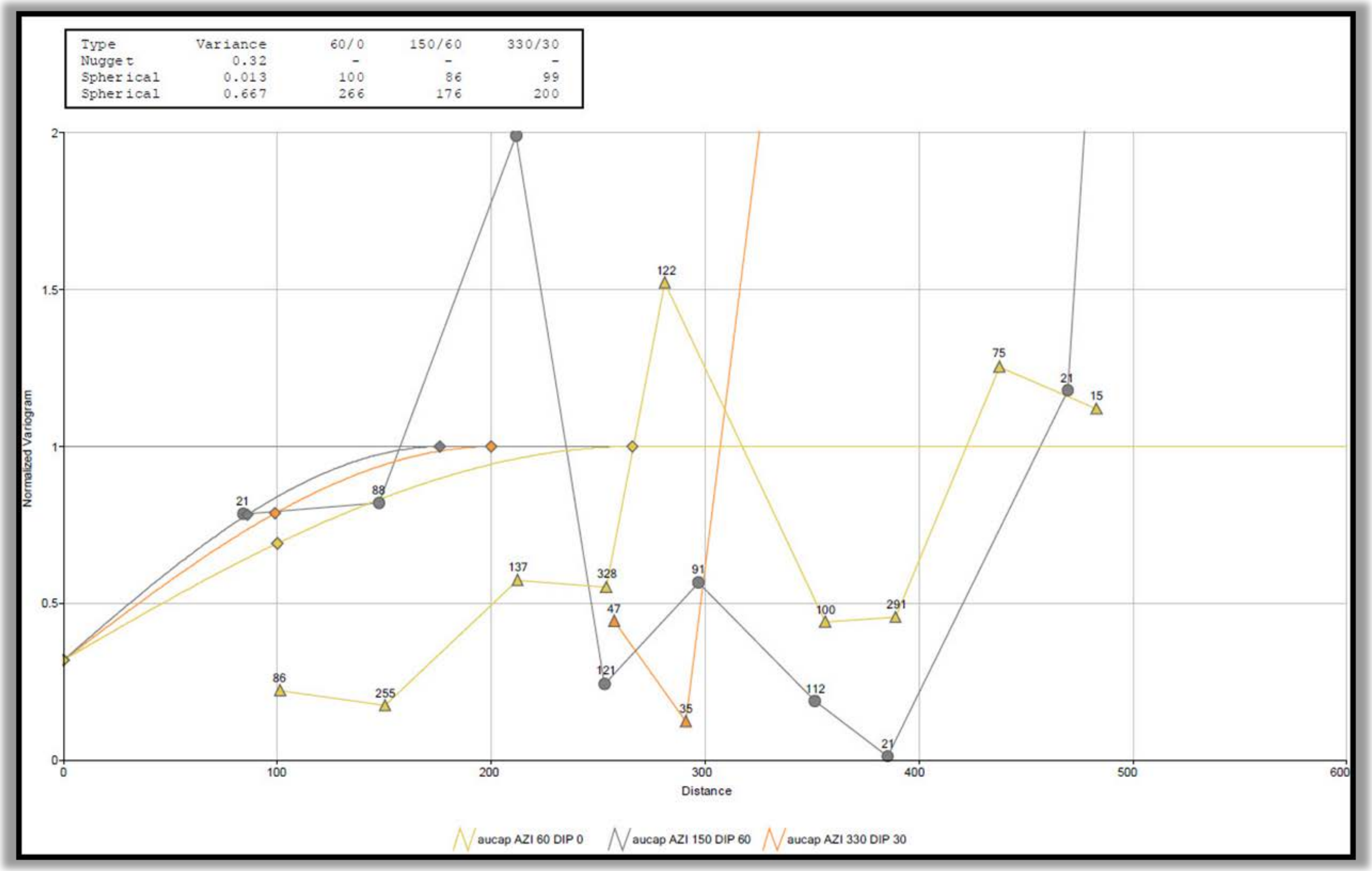


Figure 14.29 – Dyke Adit Vein Silver Variogram

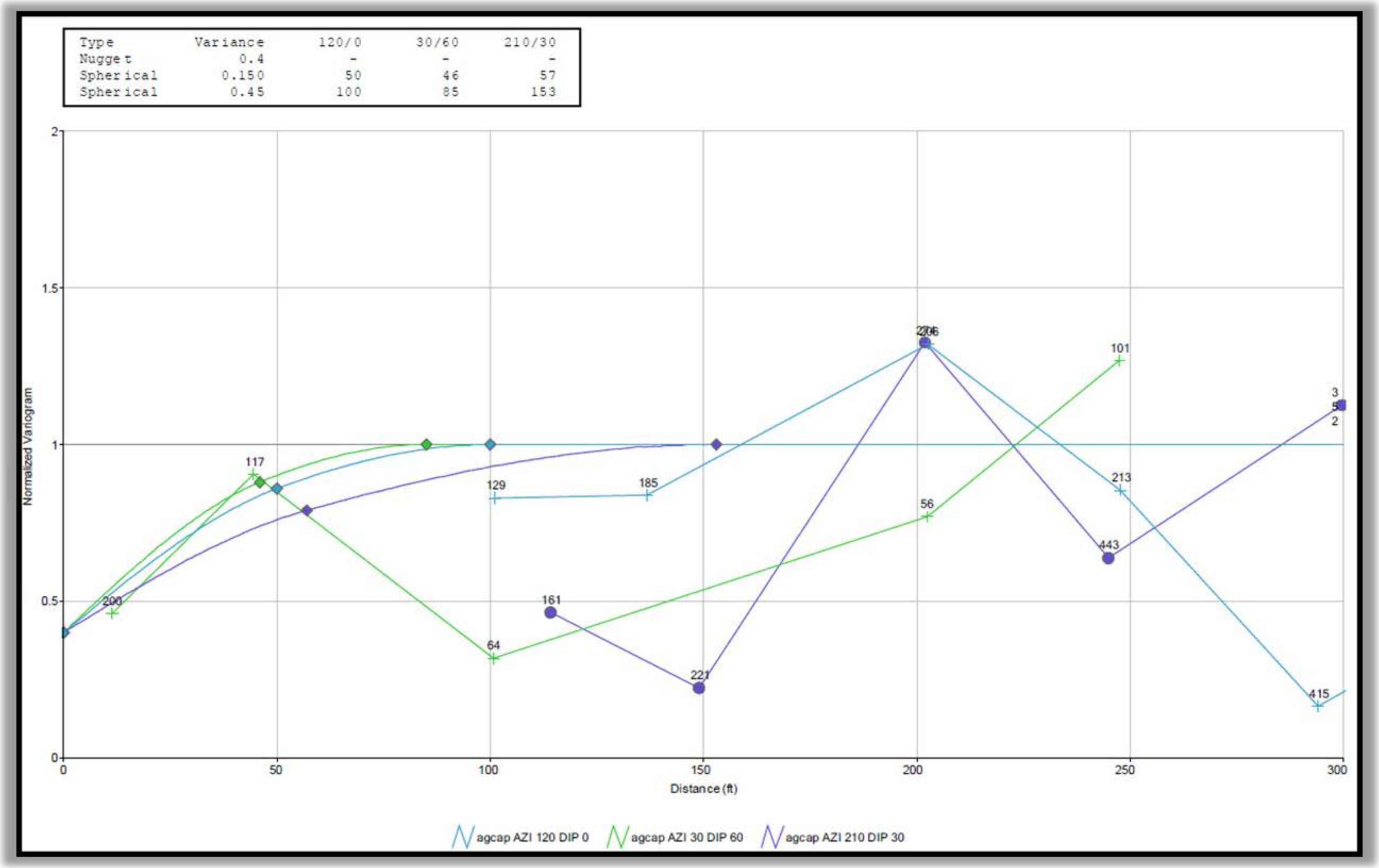


Figure 14.30 – East Hill Vein Gold Variogram

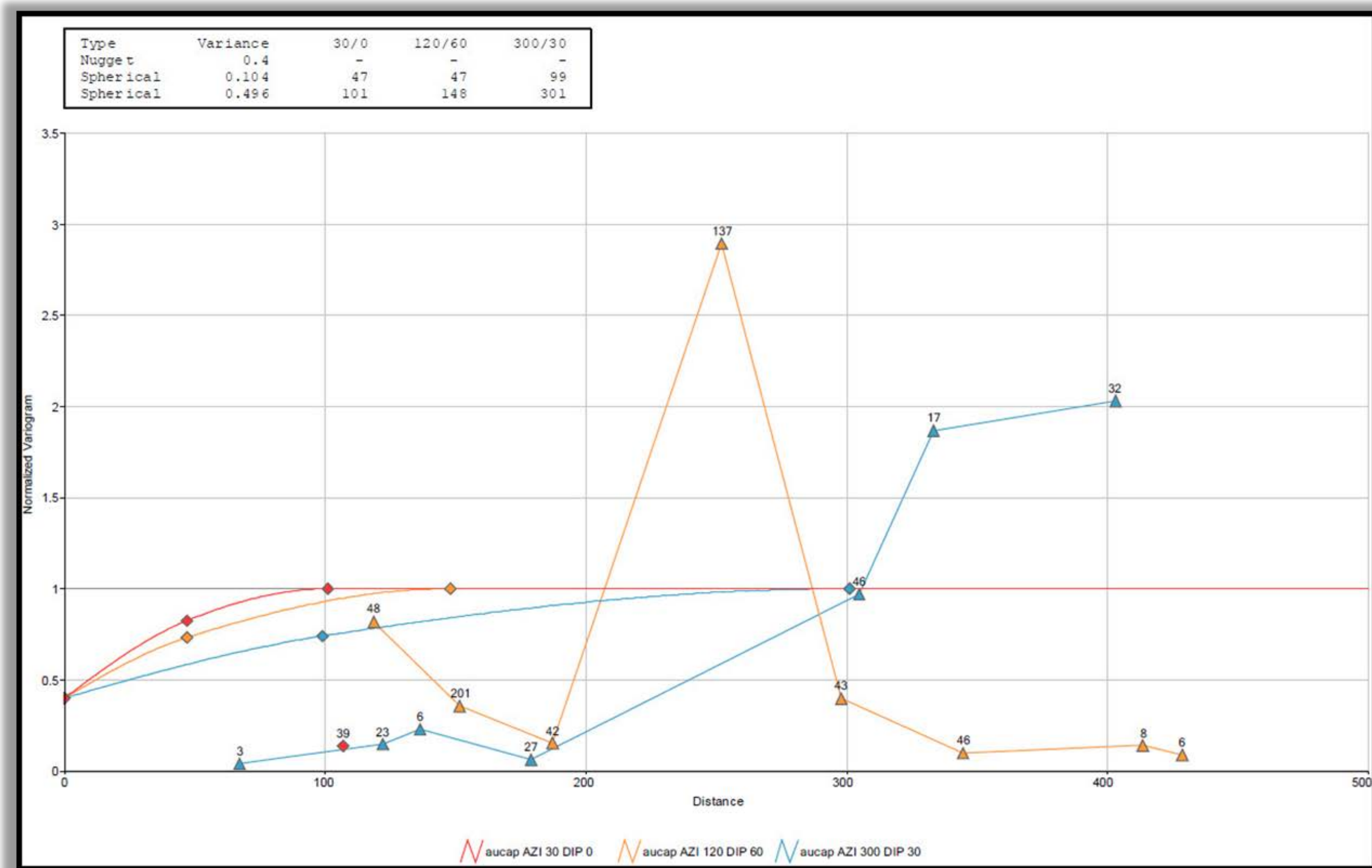


Figure 14.31 – East Hill Vein Silver Variogram

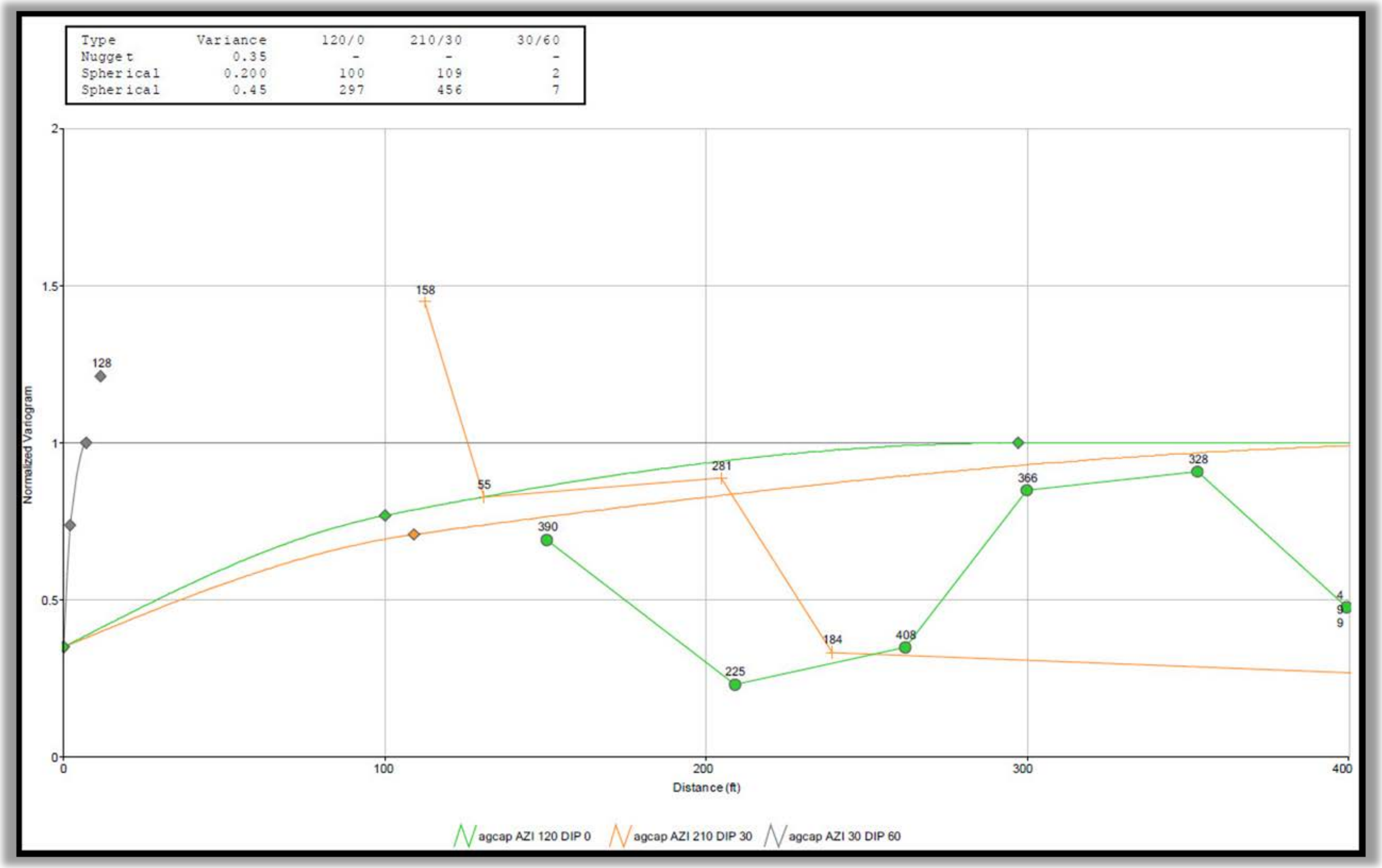


Figure 14.32 – Bear Creek HW Zone Gold Variogram

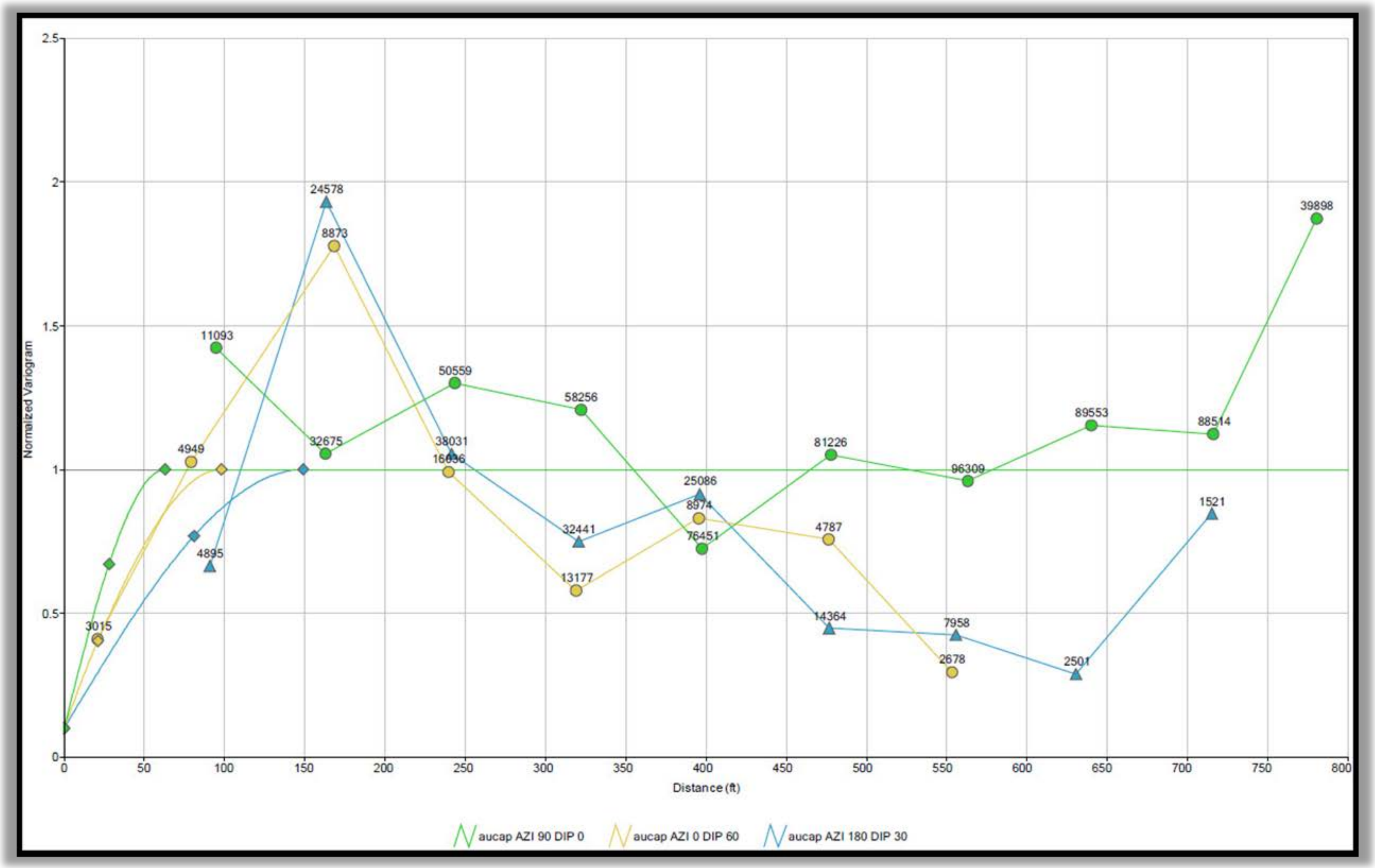


Figure 14.33 – Bear Creek HW Zone Silver Variogram

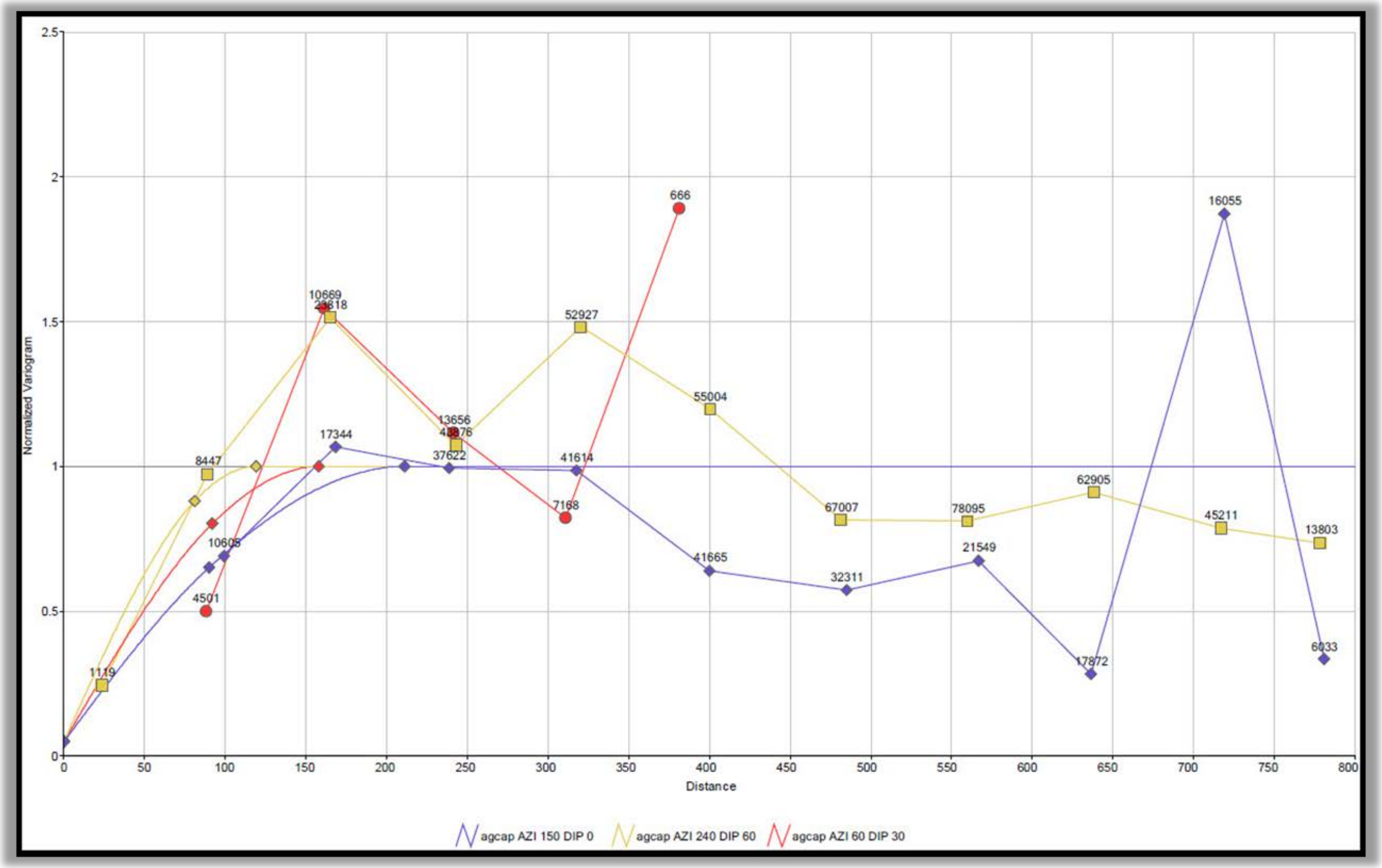


Figure 14.34 – Bear Creek FW Zone Gold Variogram

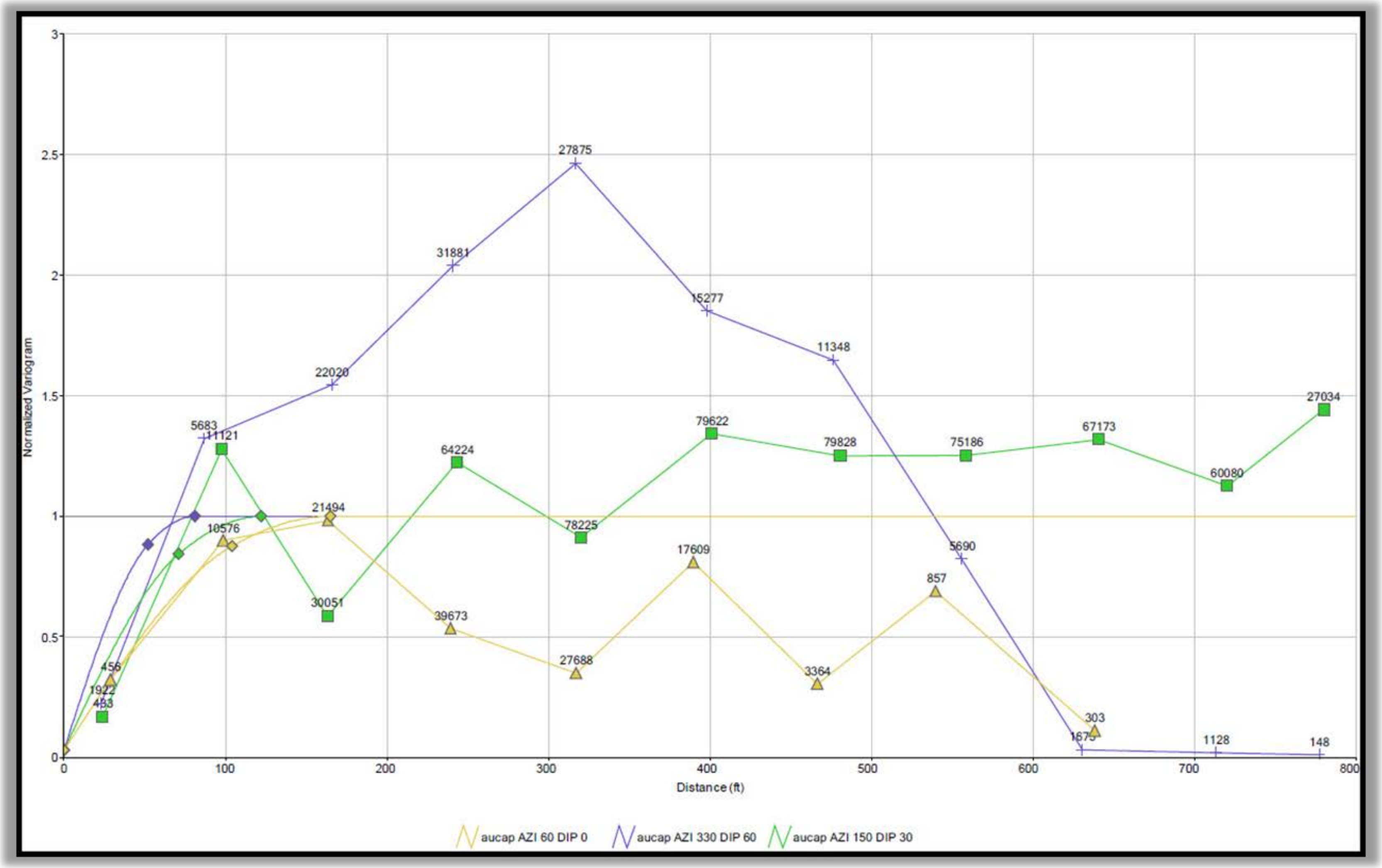


Figure 14.35 – Bear Creek FW Zone Silver Variogram

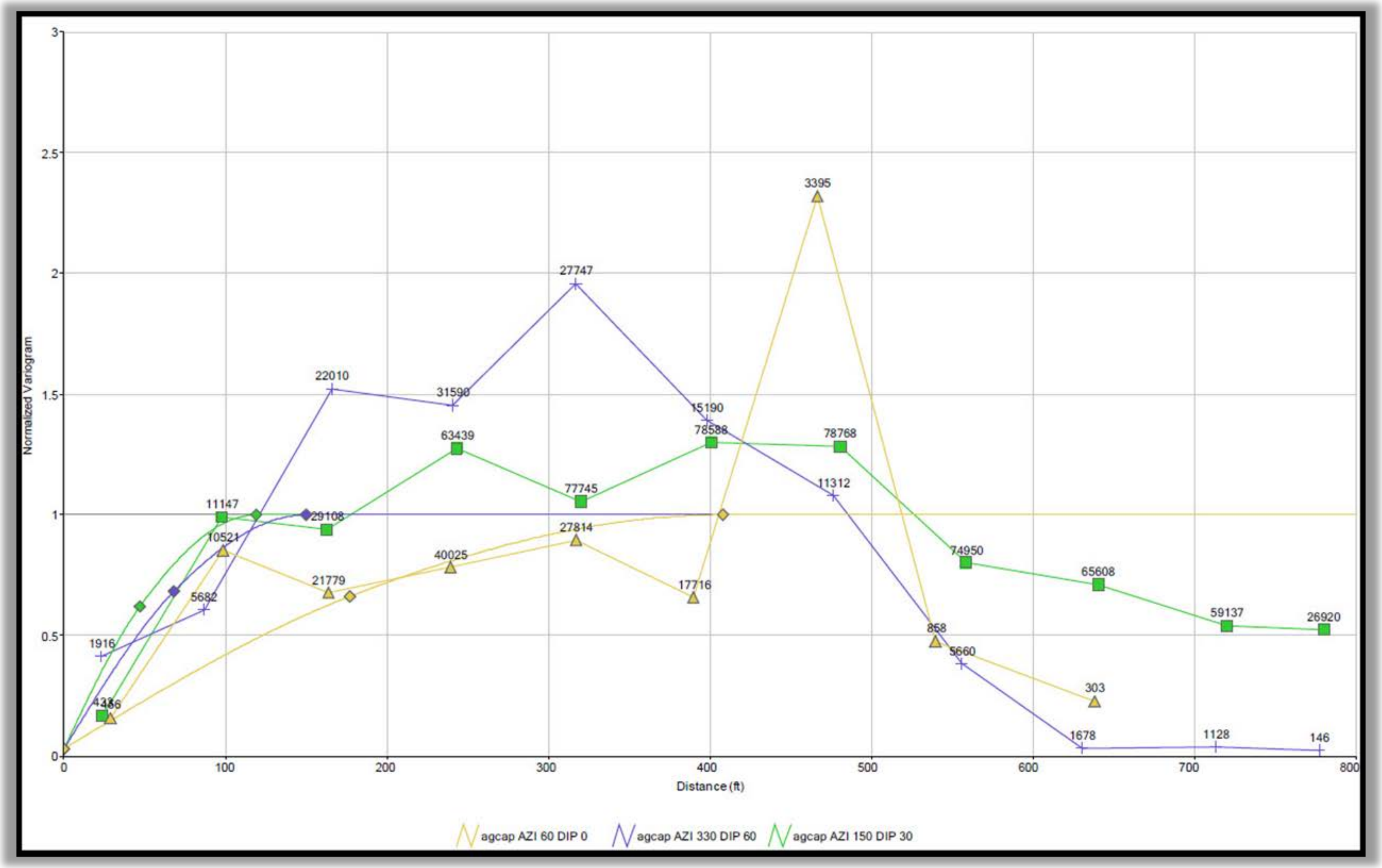


Figure 14.36 – Main Zone Gold Variogram

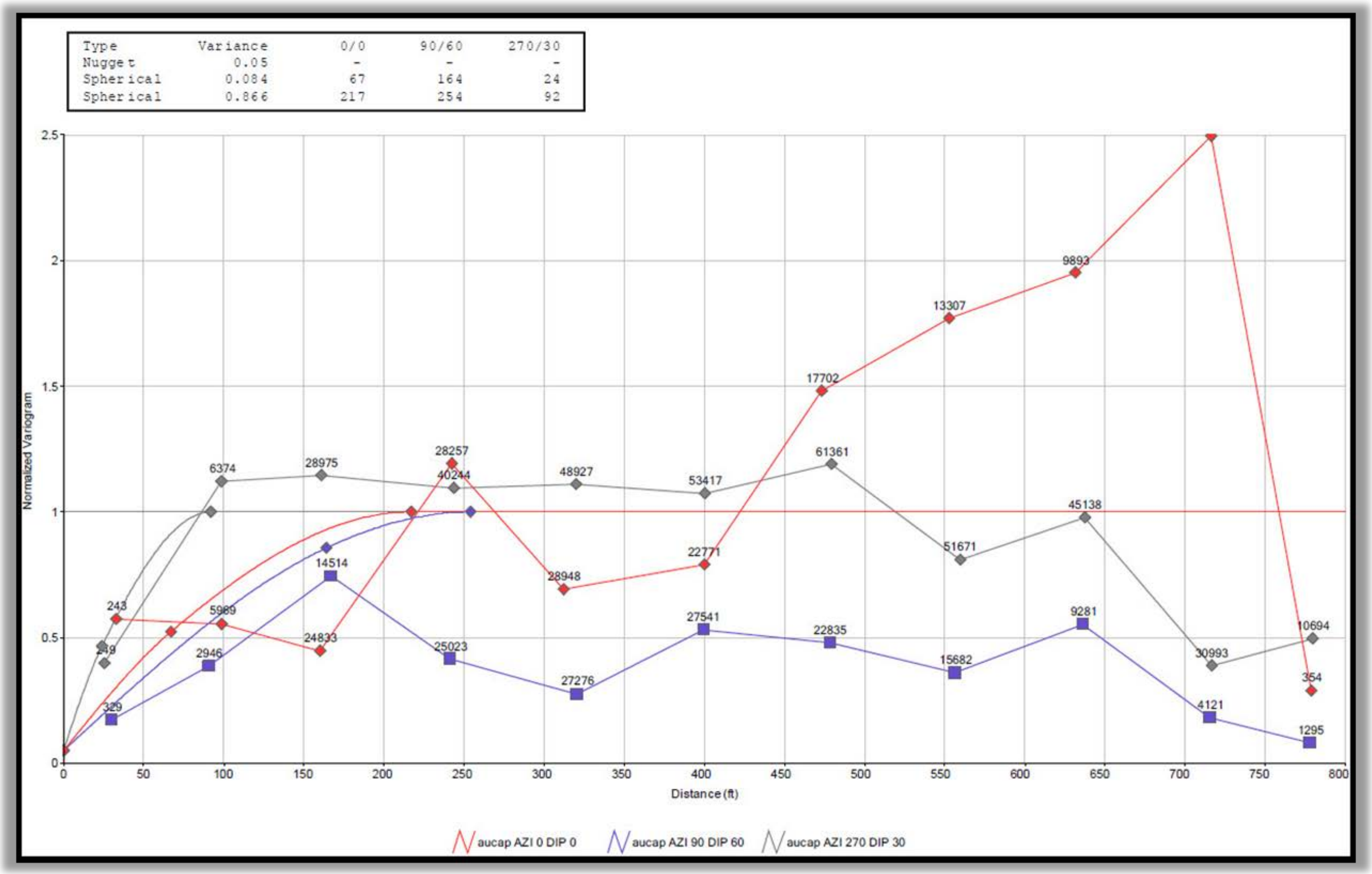


Figure 14.37 – Main Zone Silver Variogram

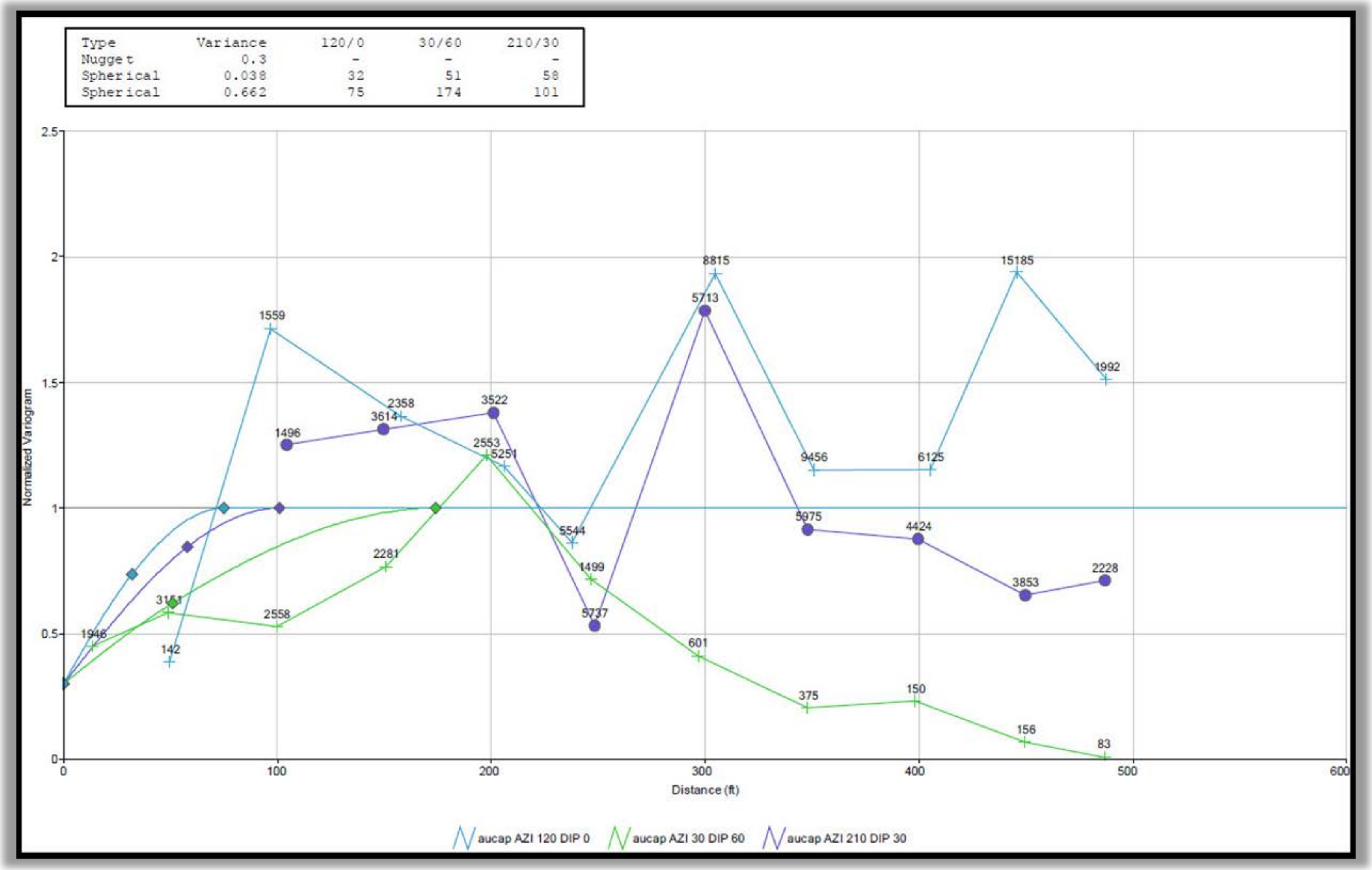


Figure 14.38 – Dyke Adit Zone Gold Variogram

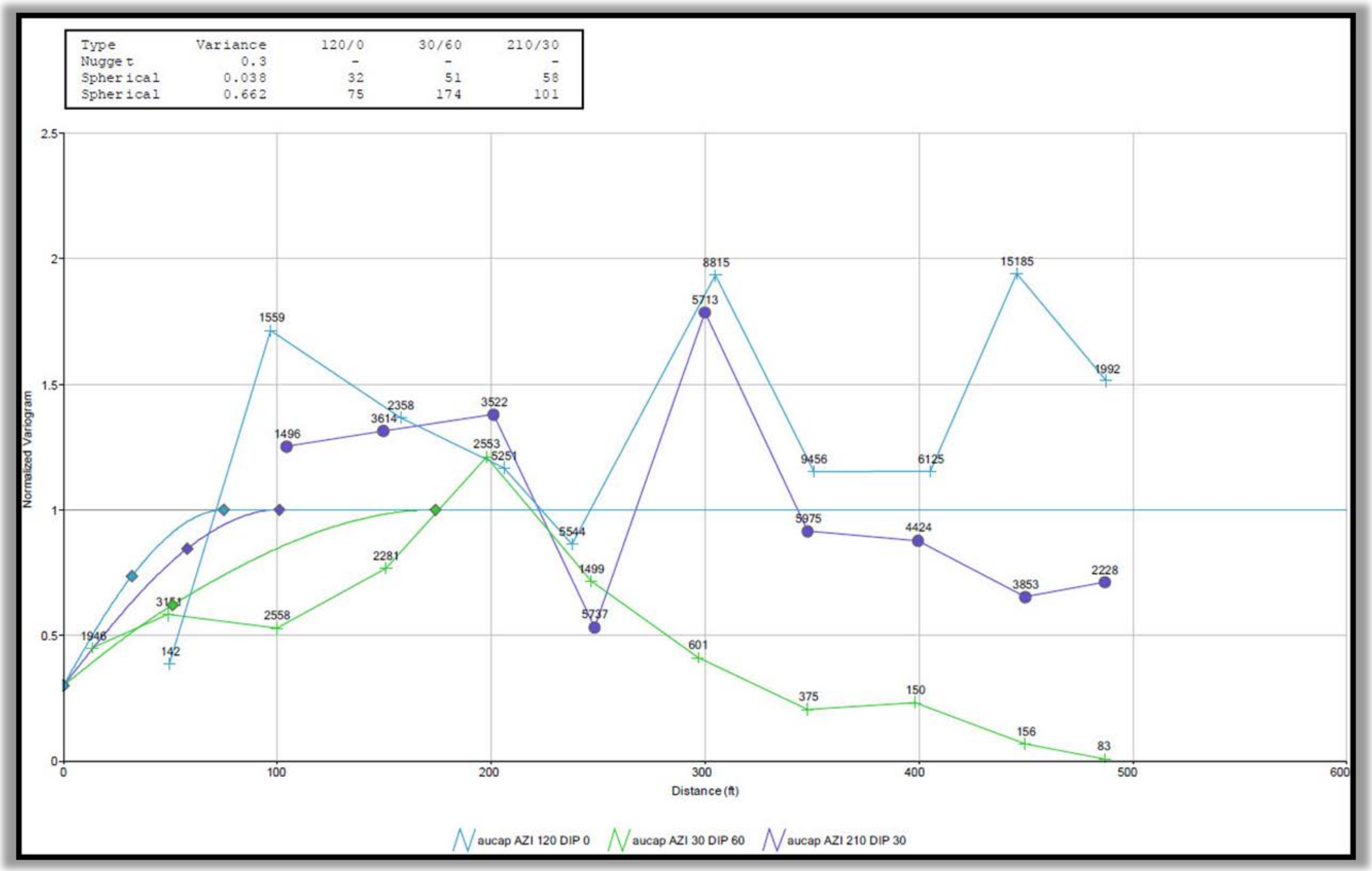


Figure 14.39 – Dyke Adit Zone Silver Variogram

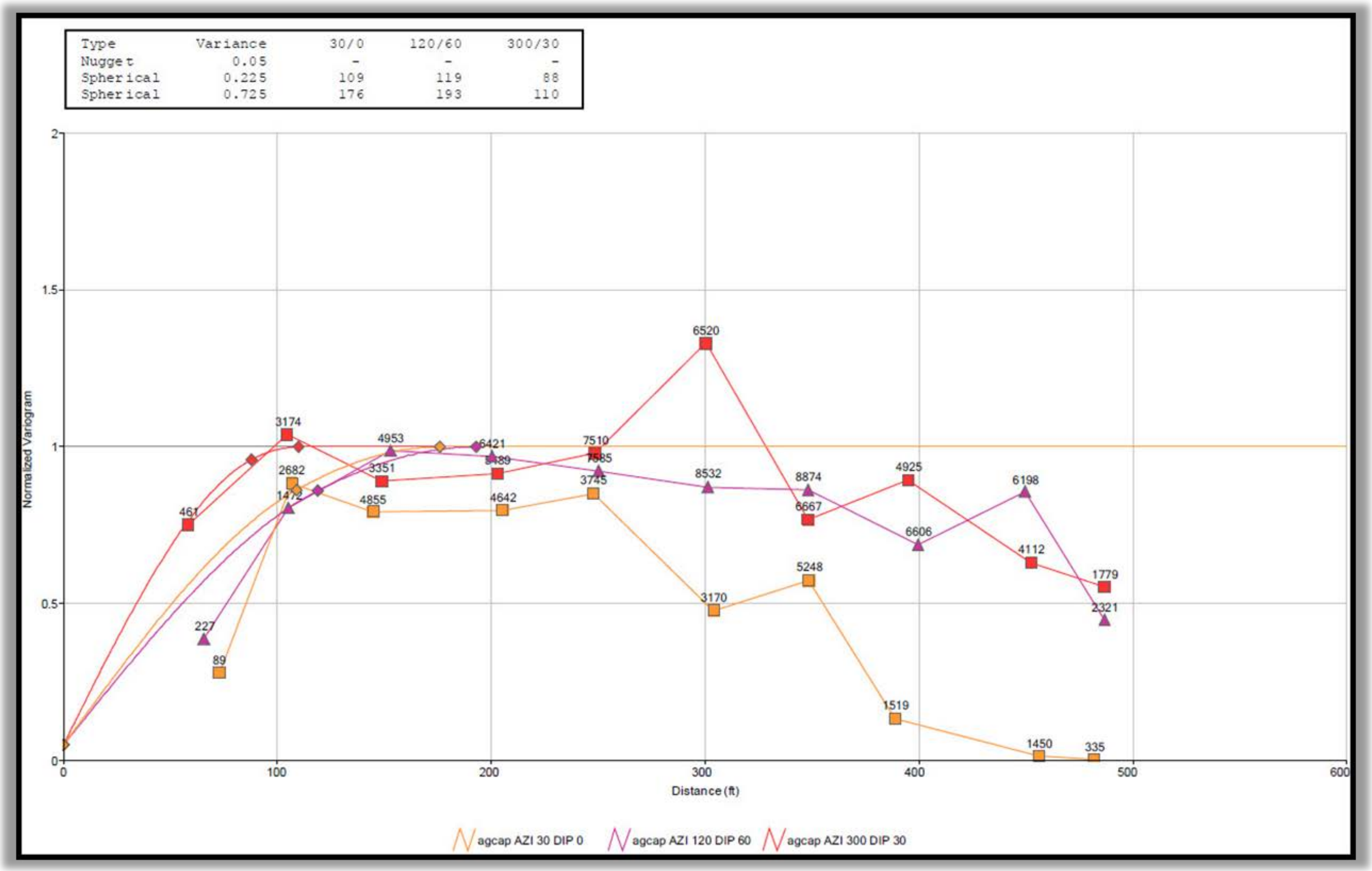
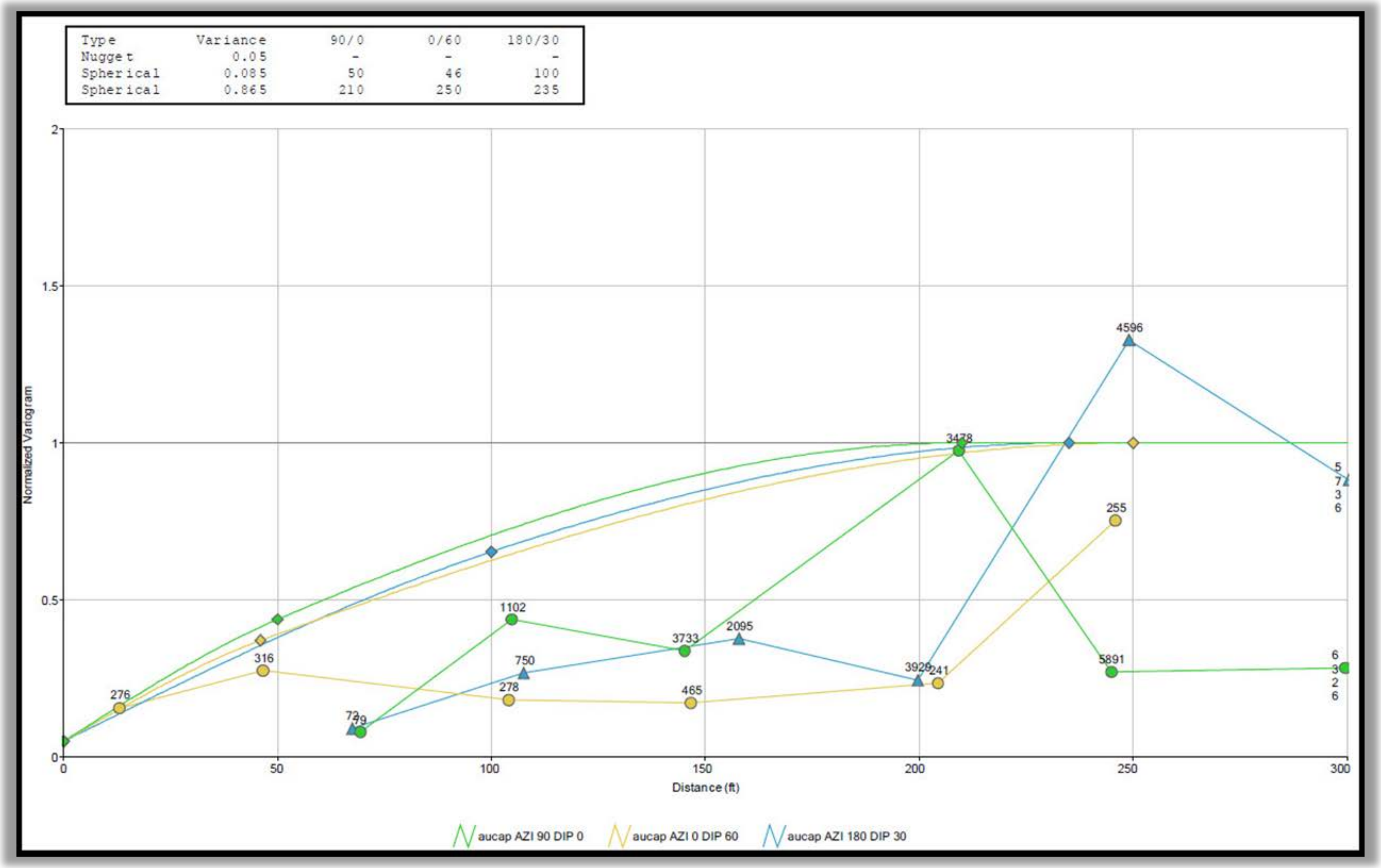
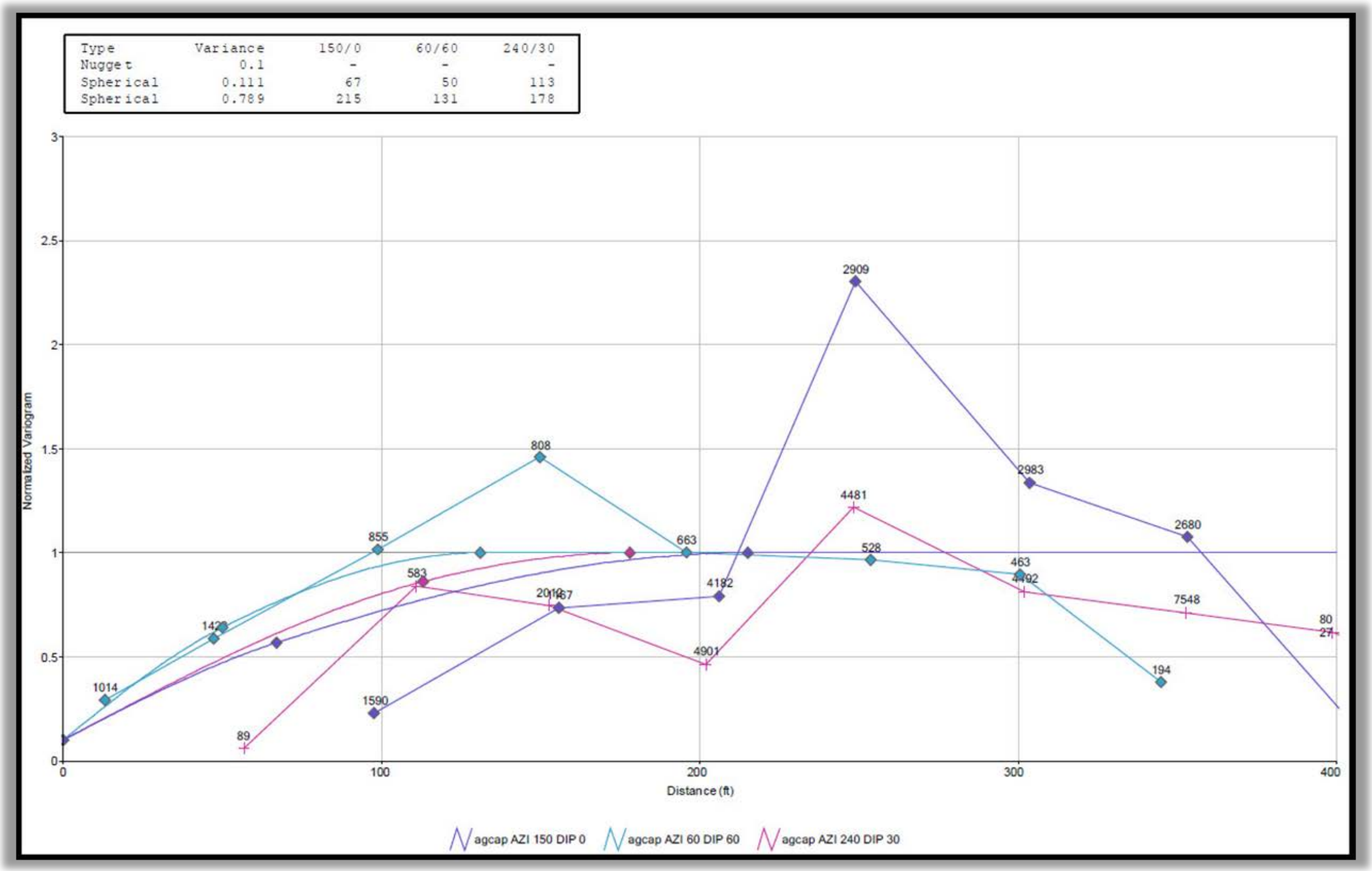


Figure 14.40 – East Hill Zone Gold Variogram



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 82 ft. section and 82 to 328 ft. 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 ft. by 7.5 ft. by 7.5 ft. 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 HW Vein	104,010,965.5	104,014,286.9	0.00
Bear Creek FW 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 HW Zone	385,778,386.4	366,367,073.0	5.03
Bear Creek FW 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 #	Imethod	Vrefnum
Dyke Adit Zone	auok	1	aucap	auok	17	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	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	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 #	Imethod	Vrefnum
East Hill Vein	auok	1	aucap	auok	9	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 FW Zone	auok	1	aucap	auok	13	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 FW Vein	auok	1	aucap	auok	3	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 #	Imethod	Vrefnum
Bear Creek HW Zone	auok	1	aucap	auok	15	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 HW Vein	auok	1	aucap	auok	5	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	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 #	Imethod	Vrefnum
Main Vein	auok	1	aucap	auok	1	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 1, 2013 has been tabulated in terms of a gold cut-off grade. Figures 14.43 to 14.45 and Tables 14.12 to 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

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

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

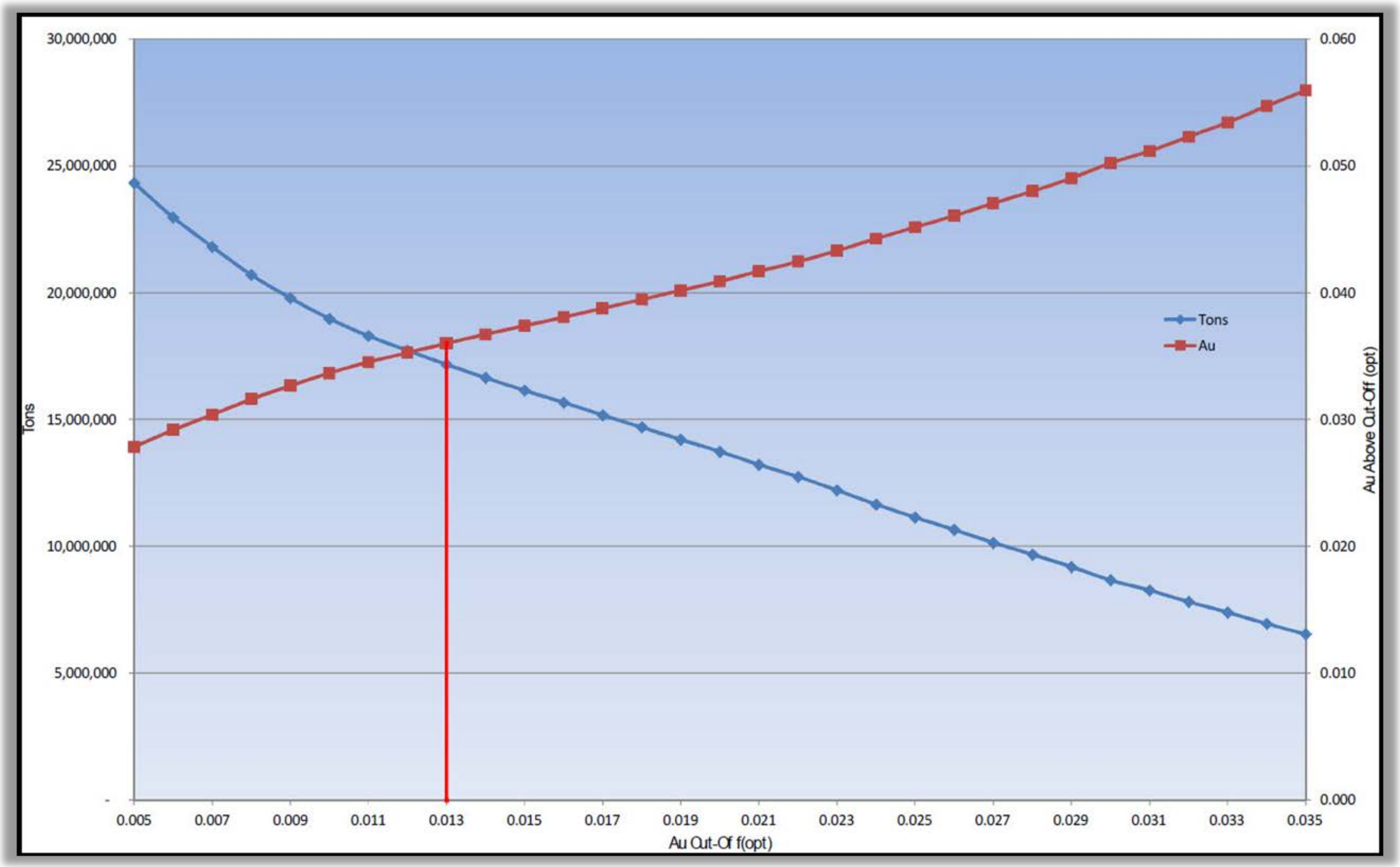


Figure 14.44 – Talapoosa Indicated Grade Tonnage Curve

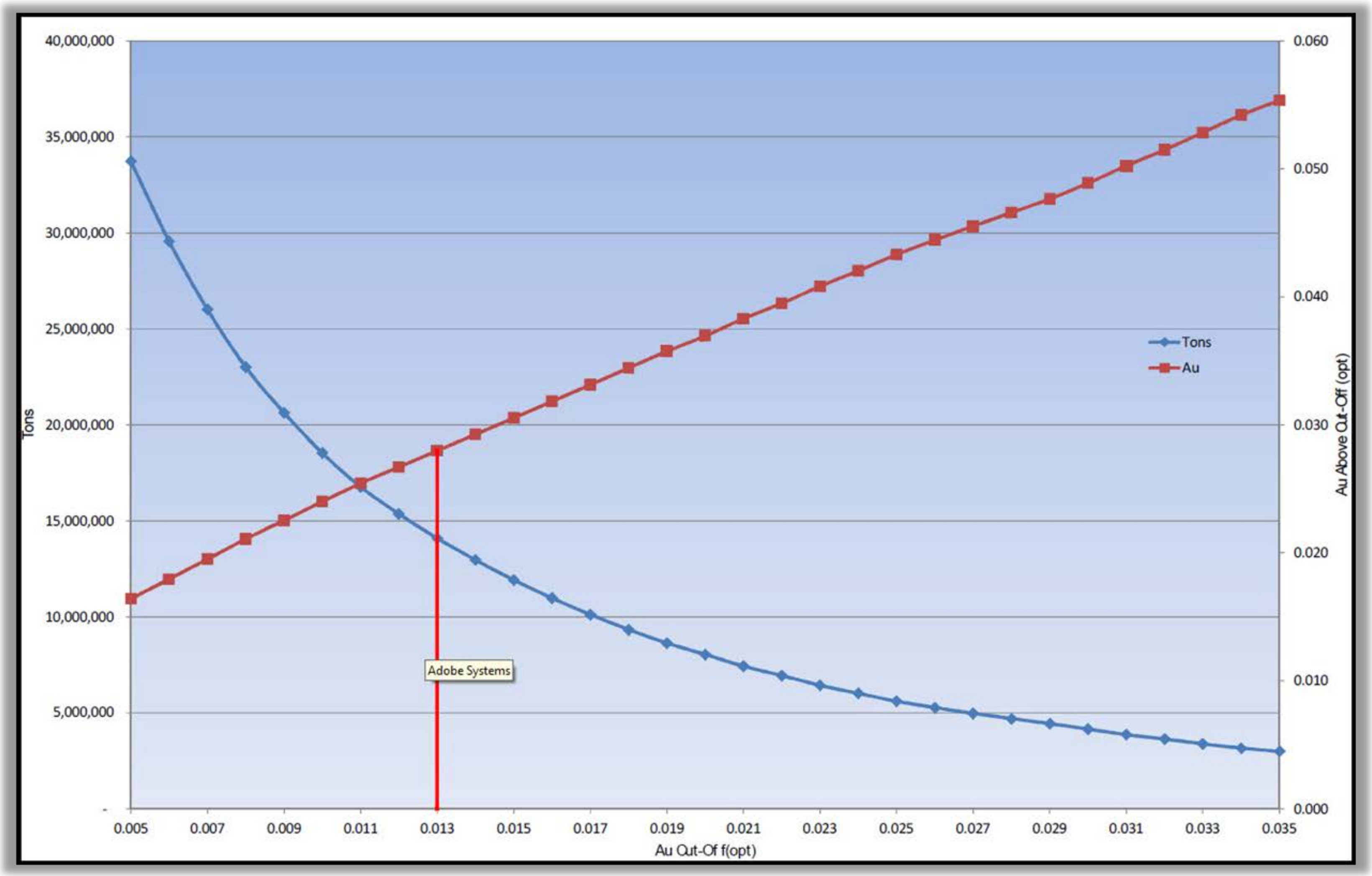
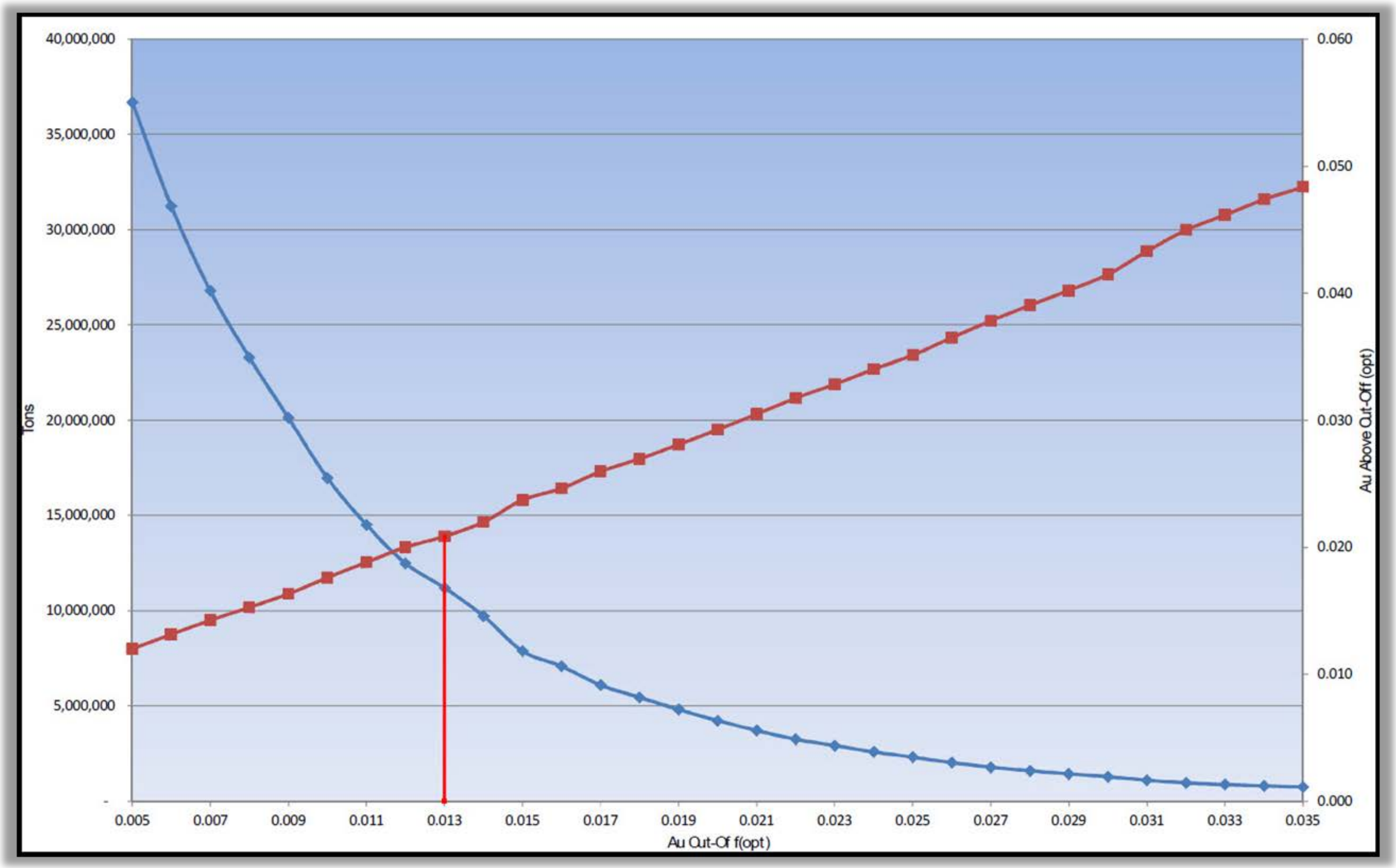


Figure 14.45 – Talapoosa Inferred Grade Tonnage Curve



Based on current mines operating in the region and a gold price of $1150/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.

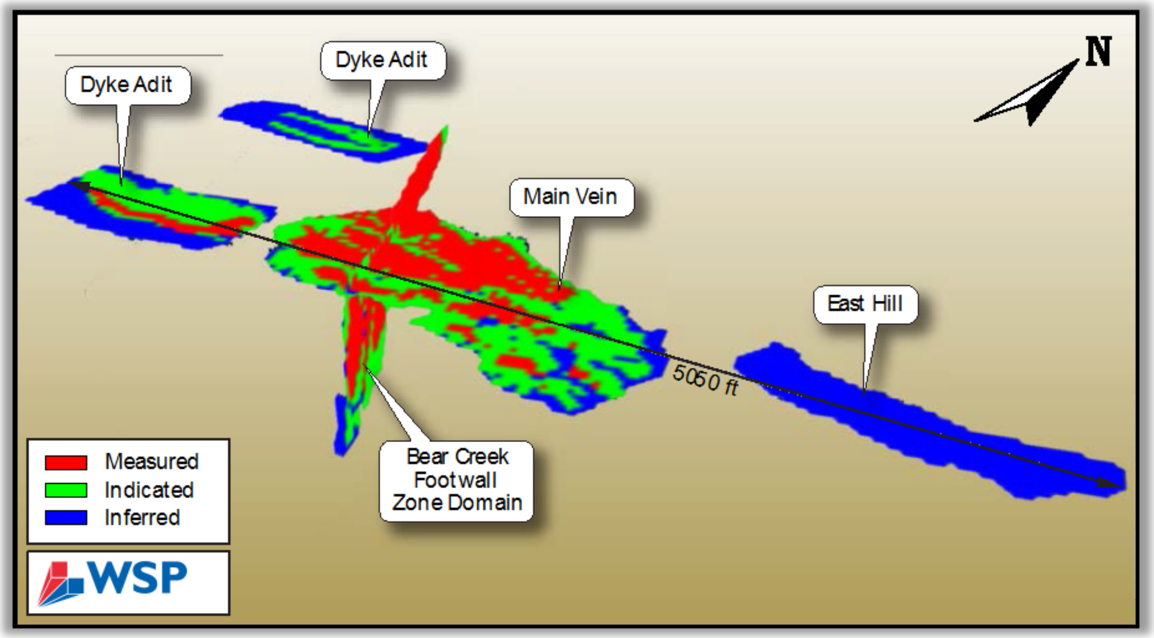
Table 14.15 – Talapoosa Mineral Resource Summary

	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 FW Zone	0.013	392,780	0.030	0.555	11,663	217,902
Bear Creek HW 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 FW Zone	0.013	436,400	0.023	0.401	10,164	174,998
Bear Creek HW 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 FW Zone	0.013	3,000	0.017	0.371	51	1,112
Bear Creek HW 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 FW Zone	0.013	5,147,790	0.033	0.496	169,891	2,555,726
Bear Creek HW 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 FW Zone	0.013	4,976,700	0.025	0.339	122,447	1,685,319
Bear Creek HW 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 FW Zone	0.013	149,000	0.029	0.221	4,353	32,988
Bear Creek HW 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:

- Visual comparison of colour-coded block model grades with composite grades on section and plan.
- Comparison of the global mean block grades for OK, ID^2, NN and composites.
- 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

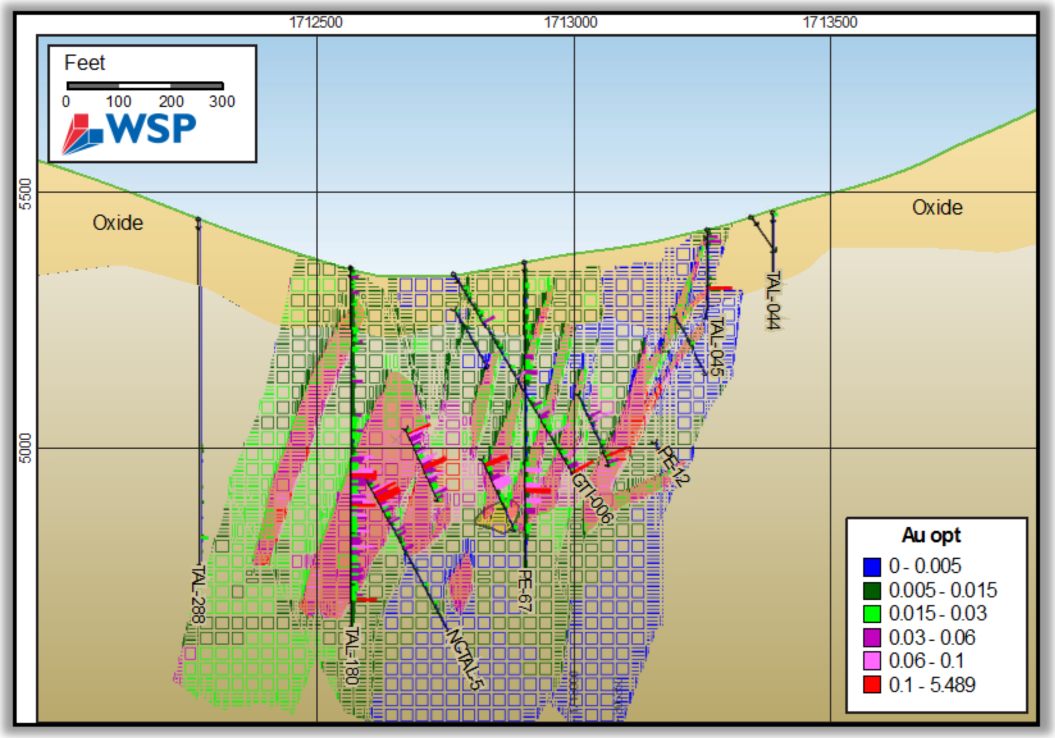
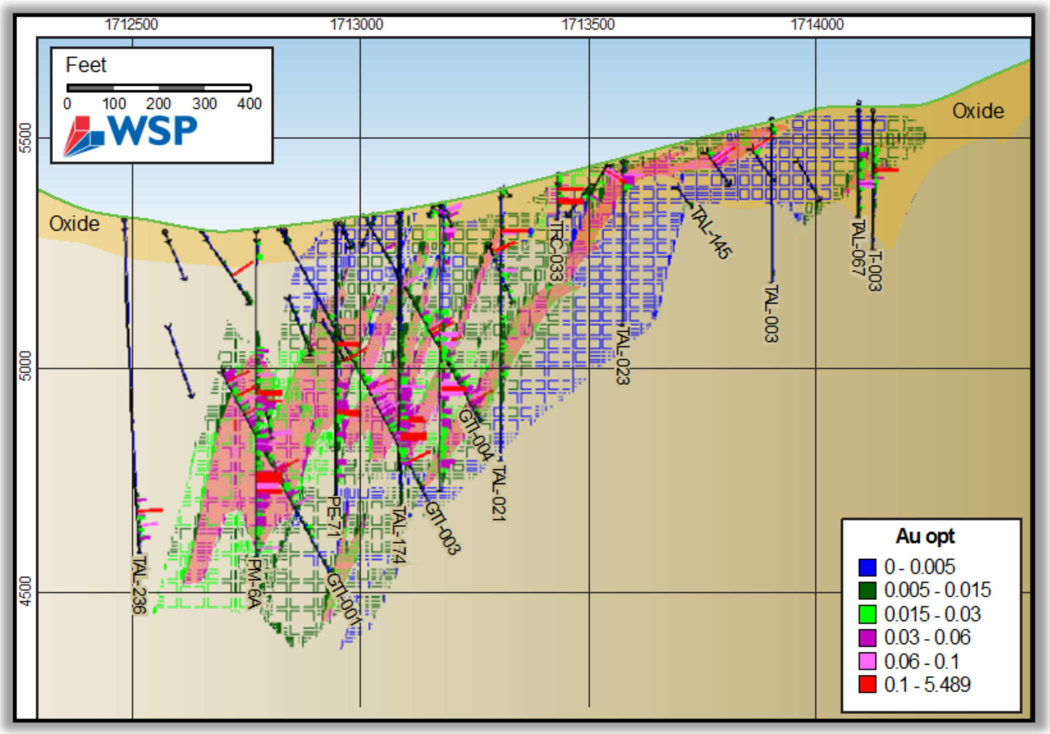


Figure 14.48 – Talapoosa Cross-Section 2



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

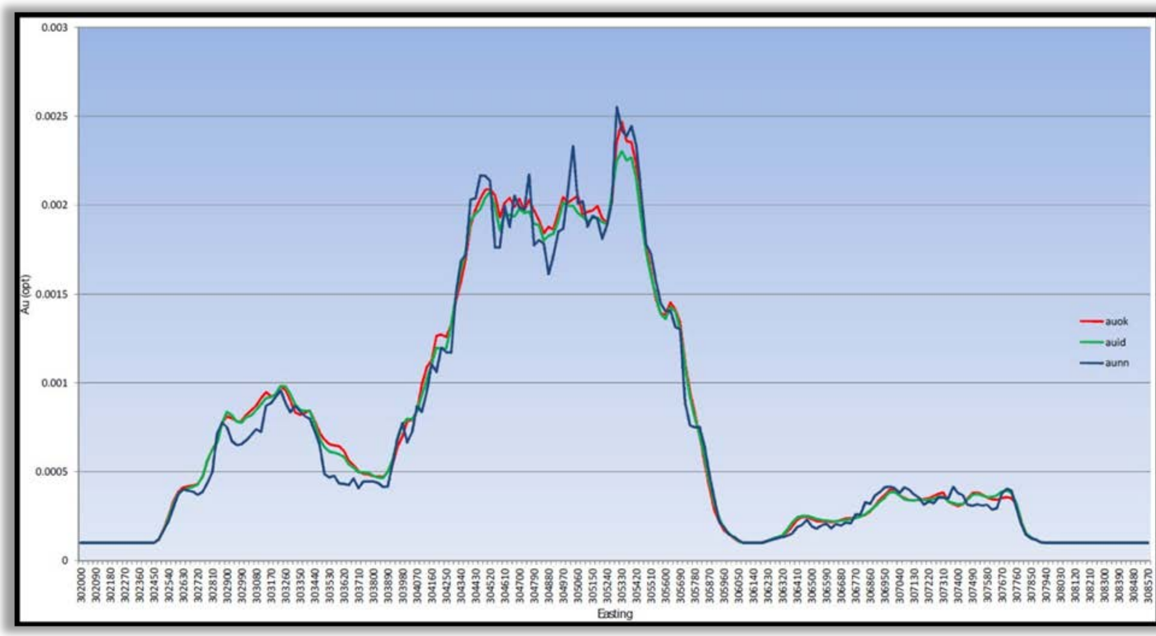


Figure 14.50 – Talapoosa Northing Plot

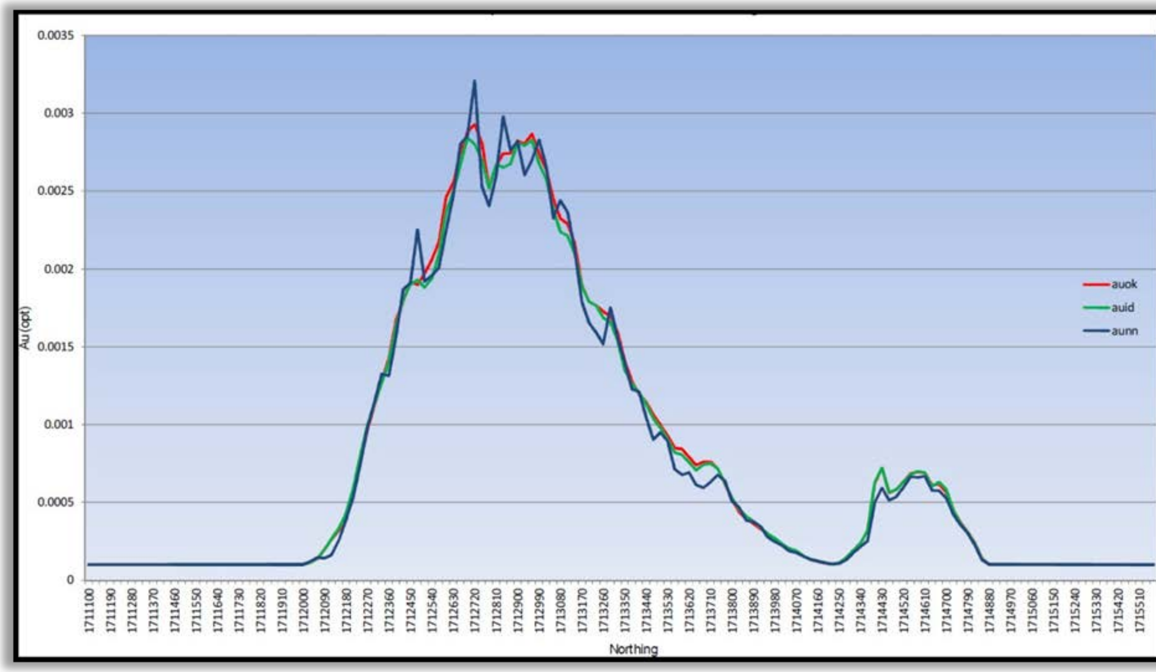


Figure 14.51 – Talapoosa Elevation Plot

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/ft^3 (2.50) Altered Host Rock: 0.072 t/ft^3 (2.32) Oxidized Host Rock: 0.067 t/ft^3 (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 HW, Bear Creek FW and Main
Block Size	25 ft. by 25 ft. by 25 ft.	30 ft. by 30 ft. by 30 ft. (27000 ft^3) with sub-celling
Estimation Method	OK with inverse distance cubed (ID^3) and NN validation	OK with ID^2 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 MINERALS RESERVE ESTIMATES

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-feasibility or Feasibility level as appropriate that include application of Modifying Factors.

A Mineral Reserve has not been estimated for the Project as part of this PEA.

16 MINING METHODS

16.1 SUMMARY

Approximately 42 Mt of minable PEA resource will be extracted by open pit mining from the Talapoosa deposit in just over 10 years. The mine production schedule calls for the production of 10,460 tpd of mineralized resource material to be placed on the heap in two twelve hour shifts, 365 days per year. The overall strip ratio is 1.47 (41.4 Mt mineralized material from within the PEA pit, 61.0 Mt waste rock); however, there are times during early mine life when the strip ratio will be higher.

16.2 MINING METHOD

Industry standard open pit mining methods will be used to extract the Talapoosa deposits. This method was selected considering the deposits' size, shape, orientation, and proximity to the surface. Drilling, blasting, loading, and hauling will be used to mine the resource material within the PEA pit so as to meet the mine production schedule.

The mine site will be prepared before commencement of operations. Vegetation will be cleared and grubbed, topsoil will be removed (where soils exists) and stockpiled / seeded for long-term storage. The main haul road will be developed and working benches will be established providing access and sufficient operating area for the mine equipment to efficiently operate.

Smaller drill rigs and pioneering equipment will work from a well-developed survey to install the first working benches, providing sufficient room for production mine equipment to operate in.

Blasthole drills will prepare production blastholes on 20-foot benches when working on both PEA resource material and waste. The holes will be loaded with ammonium nitrate – fuel oil (ANFO) blasting agent and shot using echelon-blasting techniques. Longer blastholes may be used when working in areas containing only waste rock.

Blasthole sampling and marking of rock materials will be used to control the grade of material sent to processing. Loader operators will also take advantage of discernable visual differences between mineralized material from within the PEA pit and waste material when applicable.

Resource material from within the PEA pit will be loaded into haul trucks and taken to the primary crusher dump pocket. Waste will be loaded into haul trucks, delivered to either the northeast or southwest waste dump, end dumped within a safe distance from the waste dumps' edge, and dozed into place.

Ancillary mobile equipment will support the mine fleet including maintenance of mine roads and infrastructure.

Ancillary facilities will be constructed to support warehousing and maintenance of mine equipment. Facilities will include a large maintenance structure with utilities and gang boxes for the storage of lubricants and tools; a fuel farm for storing both diesel and gasoline, a truck wash which uses recycled water, a fresh water source, electrical power, and offices for management and operations.

Concurrent reclamation will be practiced while building the waste dumps to minimize the amount of material to be moved at the end of mine life.

16.3 PIT LIMIT ANALYSIS

This PEA is preliminary in nature. In addition to the Measured and Indicated Resources, the mine plan presented in this section includes Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this PEA will be realized.

Economic mine limits were determined using Geovia's Whittle™ 4.5.5 software that uses the Lerchs-Grossmann (LG) algorithm. The LG algorithm progressively identifies economic blocks, taking into account waste stripping, that results in a highest possible total value mined within the open pit shell, subject to the specified pit slope constraints.

16.3.1 SCENARIOS

Two scenarios were considered during the course of the PEA study:

1. Oxide Only Scenario – This scenario considered mining only the Oxide portion of the resource.
2. Total Resource Scenario – This scenario considered mining the Oxidized, HW Type, and FW Type materials without any elevation or boundary constraint. This is the Base Case scenario.

The input parameters for the Oxide only scenario varied slightly from the base case scenario. A comparison of the results from the two scenarios is presented in Table 16.1.

Table 16.1– Comparison of Scenarios

		Oxide Only Scenario RF1 Pit Shell	Base Case Scenario RF1 Pit Shell
Throughput Rate Assumption	M st/y	2.125	3.818
Metal Price Au	$/oz	1,150	1,150
Ag	$/oz	16	16
Heap Leach	s.tons	5,713,000	47,759,000
Grade – Au	oz/ton	0.027	0.022
Grade – Ag	oz/ton	0.374	0.327
Contained Ounces - Au	oz	154,300	1,050,700
Contained Ounces – Ag	oz	2,136,700	15,617,000
Waste	s.tons	14,444,000	85,090,000
Total Rock	s.tons	20,157,000	132,850,000
Strip Ratio		2.5	1.8
Mine Life	years	2.7	12.5
Recovered Ounces – Au	oz	115,700	683,900
Recovered Ounces – Ag	oz	5,834,700	8,226,000

Note: Numbers may not add exactly due to rounding.

All further analysis presents only the Base Case scenario.

16.3.2 INPUT PARAMETERS

A 3D geological block model and other economic and operational variables were used as inputs into the LG program. These variables include overall pit slope angle, mining costs, processing costs, selling costs, metal prices, and other variables listed in Table 16.2. Although these parameters are not necessarily final, a reasonable degree of accuracy is required, since the analysis is an iterative process. The economic parameters used at the time of the pit optimization do not necessarily conform to those stated in the economic model.

Table 16.2 – Pit Optimization Parameters, Base Case

Parameter		Units	Base Case	
Process			Heap Leach	
Process Production Rate		M st/y	3.818	
Mining Dilution		%	4	
Mining Recovery		%	96	
Overall Slope Angle		Degrees	Varies by zone 36 – 47	
Mining Cost				
Contractor		$/ton mined	2.24	
Incremental haulage cost				
Elevations 5120 – 5000		$/ton mined	0.20	
Elevations <5000		$/ton mined	0.20	
Processing Cost (including additional costs for G&A, rehandle, reclamation, etc.)		$/ton processed	5.00	
Metallurgical Recovery		%	**Au**	**Ag**
Oxide Type			77	47
HW Type			65	60
FW Type			59	45
Metal Prices	Au	$/oz	1150	
	Ag	$/oz	16	
Selling Cost		% of Metal Price	4	
Discount Rate		%	8	
Resource Classifications Used in Optimization			Measured Indicated Inferred	

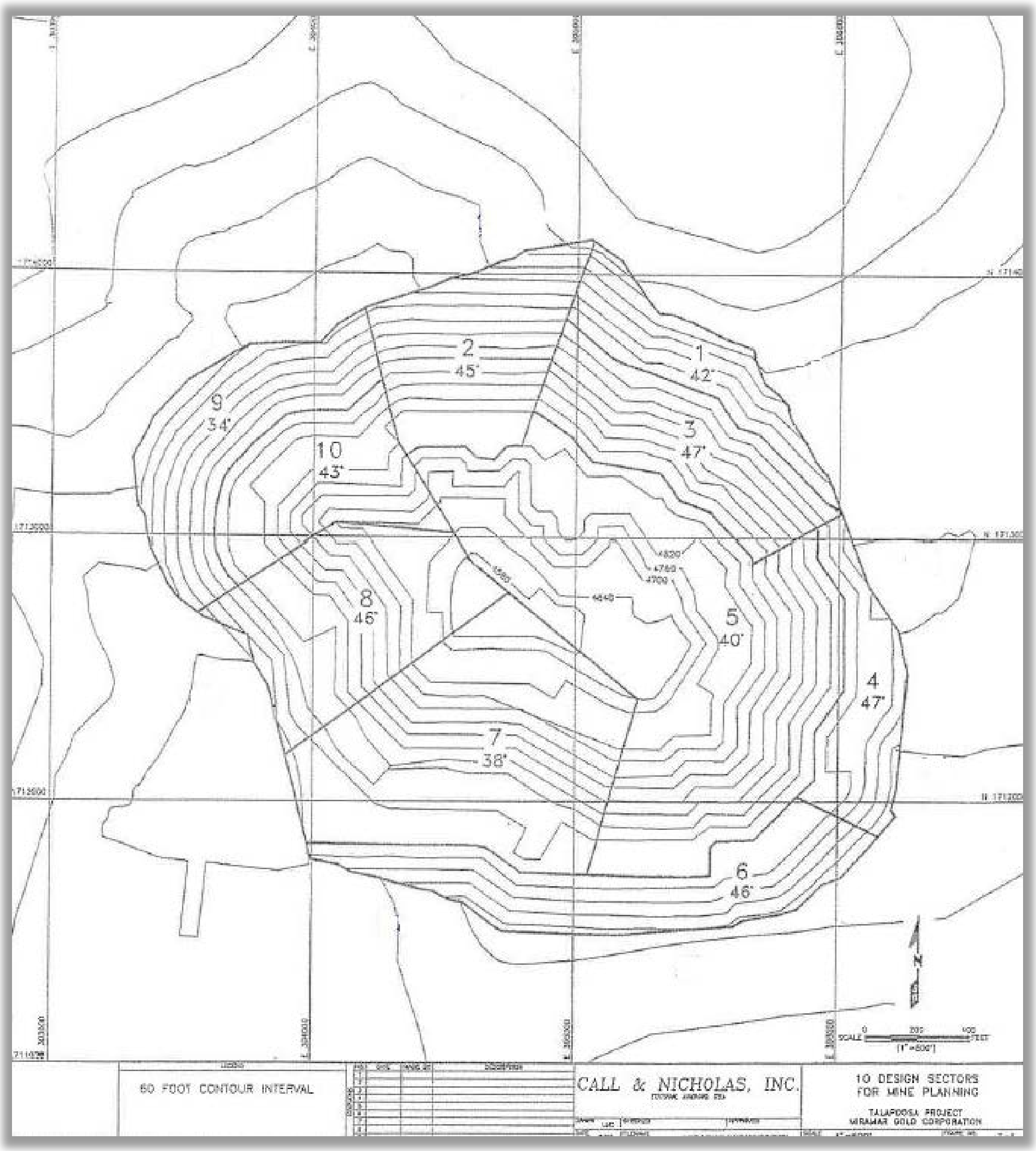
16.3.2.1 OVERALL SLOPE ANGLE

The Overall Slope Angles were based upon the report titled “Recommended Slope Angles for the Feasibility Study of the Talapoosa Pit”, July 1995 by D.E. Nicholas, P.E. and S.C. Bird.

The recommended slope angles presented in the above report were based on cell mapping, oriented core logging, and geotechnical test work conducted by Calls & Nicholas, Inc. during May and June 1995 and on the geologic interpretations of Miramar’s geology staff at that time.

The Talapoosa block model was partitioned into 10 sectors in plan view as shown in Figure 16.1. For the PEA purposes, the inter-ramp slope angles were reduced 0-2 degrees to account for haulage ramps. Remaining blocks were assigned to an eleventh sector with an overall slope angle of 42 degrees.

Figure 16.1 – Design Sectors and Slope Angles for Mine Planning (Calls & Nicholas, Inc., 1995)



Source: Call & Nichols, Inc. (1995)

16.3.2.2 CUT-OFF GRADE

Milling cut-off grade was used to classify the material contained within the open pit limits as either material for processing or material for waste. This break-even cut-off grade is calculated to cover processing costs, general and administrative costs, and selling costs using the economic and technical parameters listed in Table 16.2 – Pit Optimization Parameters, Base Case. Revenue is calculated from gold and silver content. Material contained within the pit shell limits and above the cut-off grade (i.e. covers the processing costs, general and administrative costs, and selling costs) was classified as heap leach feed, while the remaining material was classified as waste.

The following cut-off grades resulted from the evaluation:

- Oxide Type: 0.006 oz/t Au, 0.720 oz/t Ag
- HW Type: 0.007 oz/t Au, 0.564 oz/t Ag
- FW Type: 0.008 oz/t Au, 0.752 oz/t Ag

16.3.3 RESULTS

The optimization process results in a series of nested pit shells, each corresponding to a Revenue Factor (RF). The revenue factor scales only the metal prices but none of the costs. A revenue factor of 1 corresponds to a metal price of 1150 $/oz Au and 16 $/oz Ag. Table 16.3 summarizes the pit shell results for the deposit calculated at incrementally increasing revenue factors.

Table 16.3 – Nested Pit Shell Results, Talapoosa Deposit, Base Case

Revenue Factor	Total Rock (M st)	Mineralization (M st)	Waste (M st)	Strip Ratio	Grade Au (oz/st)	Grade Ag (oz/st)
0.50	4.0	1.9	2.0	1.1	0.037	0.490
0.52	4.5	2.1	2.4	1.2	0.036	0.489
0.54	6.7	2.4	4.3	1.8	0.036	0.528
0.56	7.5	2.7	4.8	1.8	0.035	0.507
0.58	61.5	20.2	41.3	2.1	0.027	0.442
0.60	66.8	22.1	44.7	2.0	0.027	0.436
0.62	68.8	23.1	45.7	2.0	0.027	0.429
0.64	72.2	24.9	47.4	1.9	0.026	0.418
0.66	77.1	26.6	50.6	1.9	0.026	0.411
0.68	82.6	28.2	54.3	1.9	0.026	0.406
0.70	86.0	29.7	56.4	1.9	0.025	0.401
0.72	89.7	31.3	58.4	1.9	0.025	0.392
0.74	91.7	32.4	59.3	1.8	0.025	0.386
0.76	93.8	33.5	60.3	1.8	0.024	0.380
0.78	94.5	34.2	60.3	1.8	0.024	0.376
0.80	97.9	35.5	62.5	1.8	0.024	0.371
0.81	99.8	36.3	63.5	1.8	0.024	0.368
0.82	100.2	36.7	63.6	1.7	0.024	0.366
0.83	102.4	37.3	65.1	1.7	0.023	0.363

Revenue Factor	Total Rock (M st)	Mineralization (M st)	Waste (M st)	Strip Ratio	Grade Au (oz/st)	Grade Ag (oz/st)
0.84	103.1	37.8	65.3	1.7	0.023	0.360
0.85	105.9	38.6	67.2	1.7	0.023	0.358
0.86	106.8	39.1	67.7	1.7	0.023	0.356
0.87	110.1	40.0	70.1	1.8	0.023	0.353
0.88	110.1	40.3	69.8	1.7	0.023	0.351
0.89	110.3	40.6	69.7	1.7	0.023	0.350
0.90	113.1	41.1	72.0	1.8	0.023	0.349
0.92	115.2	42.2	73.0	1.7	0.022	0.344
0.94	119.1	43.4	75.7	1.7	0.022	0.340
0.96	123.6	45.0	78.6	1.8	0.022	0.336
0.98	132.0	47.1	84.9	1.8	0.022	0.329
1.00	132.8	47.8	85.1	1.8	0.022	0.327

A basic schedule is applied to the pit shells to produce a “pit-by-pit” graph. An objective of the pit-by-pit graph is to illustrate the impact of scheduling on the pit shells and to identify “more optimal” pit shells to use for mine planning and scheduling.

In this case, three schedules are represented:

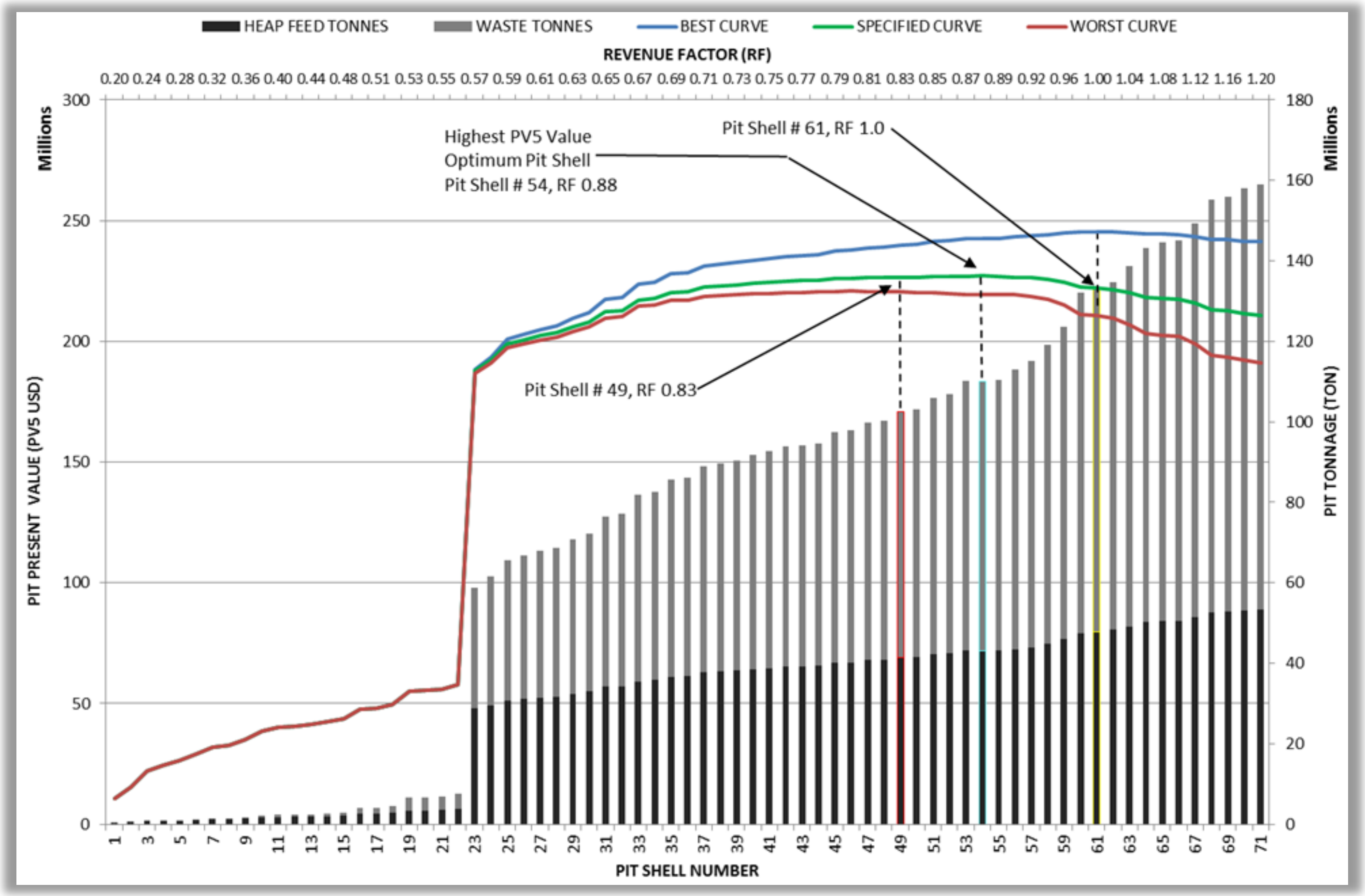
→ The Best Case schedule consists of mining out nested Pit Shell 1, the smallest pit, and then mining out each subsequent pit shell from the top down, before starting the next pit shell. This schedule is seldom feasible because the pushbacks are usually much too narrow to allow production mine equipment access. Its usefulness lies in setting an upper limit to the achievable Present Value (PV). If, as is sometimes the case, Worst Case and Best Case schedules differ by only a few percent then, for that pit, mining sequence is relatively unimportant from an economic point of view.

→ The Worst Case schedule consists of mining each bench completely before starting on the next bench. This schedule is usually feasible though not practical as the schedule mines waste material earlier than required. Its usefulness lies in setting a lower limit to the PV.

→ The Specified Case approximates a more realistic mining schedule, between the Best and Worst Cases, by defining a sequence of pit pushback outlines. Ideally, the selection of pushbacks should satisfy the mining constraints and produce a PV curve that is as close as possible to the Best Case curve.

Figure 16.2 illustrates the pit-by-pit graph.

Figure 16.2 – Pit Optimization Results, Pit-by-Pit Graph, Base Case



Note that the Present Value shown in the above figure is only as a guide in pit shell selection. The actual net present value of the Project is summarized in the Economic Analysis section of this report.

Based on the analysis of the optimization results, three of the economic pit shells are identified:

- Pit Shell #54, RF0.88, which corresponds to the highest PV5 value on the specified curve;
- Pit Shell #61, RF1.0, which corresponds to Revenue Factor 1 pit; and
- Pit Shell #49, RF0.83.

Table 16.4 summarizes the details of the pit shells selected.

Table 16.4 – Pit Shell Selection Comparison

		Max PV on Specified Curve Pit Shell	RF1	Pit Shell 49 RF0.83
Pit Shell #		54	61	49
RF Factor		0.85	1.0	0.83
Heap Leach	M tons	43.2	47.8	41.4
Au	oz/st	0.022	0.022	0.022
Ag	oz/st	0.336	0.327	0.339
Contained ounces - Au	k oz	949.5	1,050.7	911.0
Contained ounces – Ag	k oz	14,502	15,617.3	14,038.8
Waste	M tons	66.9	85.1	61.0
Total Rock	M tons	110.1	132.8	102.4
Strip Ratio		1.6	1.8	1.5
Mine Life	years	11.3	12.5	10.9
Recovered ounces – Au	k oz	618.8	683.9	593.7
Recovered ounces – Ag	k oz	7,618.7	8,226.1	7,363.2
Present Value, PV5	K $	227.0	222.0	226.5

Note that the Present Value shown in the above table is only a guide in pit shell selection. The actual Net Present Value (NPV) of the Project is summarized in the Economic Analysis section of this report.

The Project has an existing Plan of Operations (Plan) which was prepared in 1996 and filed with the Bureau of Land Management. The Plan permitted 42 million tons of mineralized material to be processed at the heap leach facility. Since the shape of the specified curve is flat around the Max PV point, there is little difference in value between the max PV pit shell and a pit shell with less than 42 million tons.

Timberline has selected to advance the mine design and scheduling using the RF0.83 pit shell that will deliver 41.4M tons of mineralized material to the heap.

16.4 ULTIMATE PIT DESIGN

Detailed mine designs, which incorporate haulage ramps and bench designs, have not been created at this stage. The ultimate pit shell for the Talapoosa deposit is shown in Figure 16.3 (Table 16.5).

The ultimate pit shell covers an area of approximately 141 acres. The elevation at the pit bottom of the Main Pit area is approximately 4,665 ft., while the pit bottom at Dyke Adit is approximately 5,295 ft. and at East Hill is approximately 5,235 ft.

Figure 16.3 – Talapoosa Ultimate Pit Shell

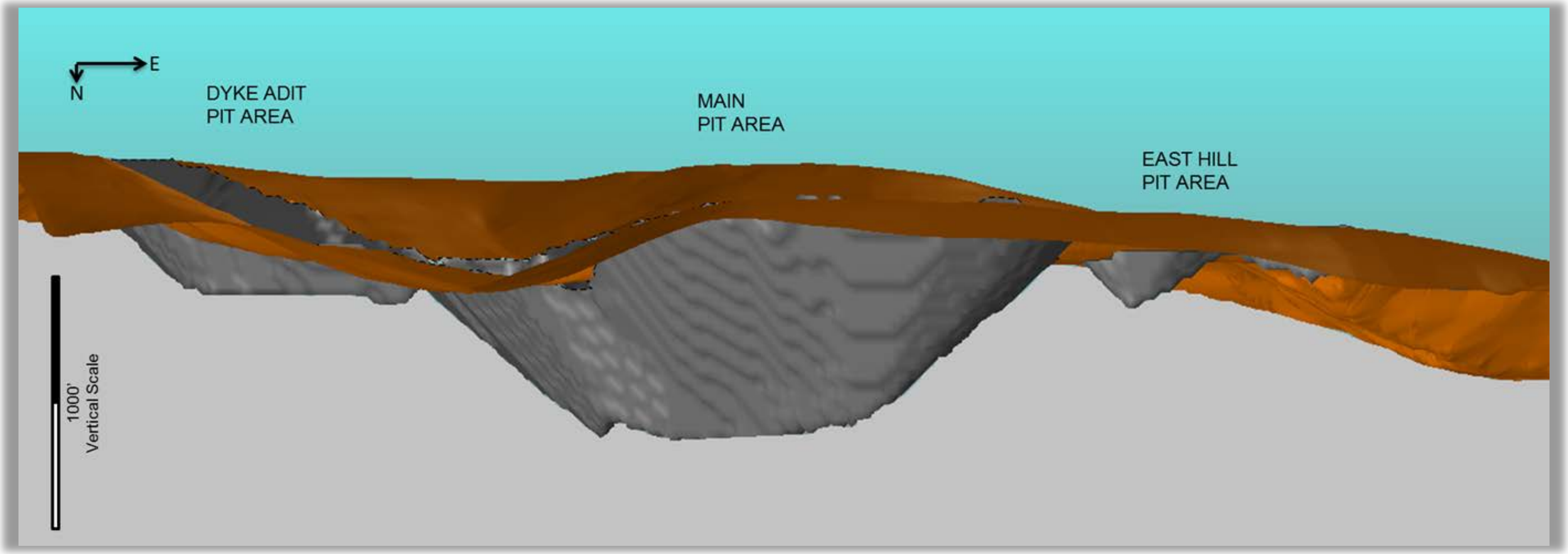


Table 16.5– Summary of Open Pit Resource

Category	Tonnage (k st)	Au Grade (oz/st)	Ag Grade (oz/st)
Total Material Mined	102,444		
Waste Rock Mined	61,023		
Heap Leach Feed by Zone, by Resource Category			
Bear Creek FW – Measured & Indicated			
Oxide	1,976	0.014	0.278
FW Type	11,502	0.023	0.373
Bear Creek FW – Inferred			
Oxide	14	0.009	0.249
FW Type	26	0.014	0.358
Bear Creek HW – Measured & Indicated			
Oxide	1,343	0.014	0.202
HW Type	9,587	0.028	0.441
Bear Creek HW – Inferred			
Oxide	528	0.012	0.192
HW Type	1,889	0.016	0.256
Main Zone – Measured & Indicated			
Oxide	4,228	0.018	0.289
HW Type	6,159	0.019	0.257
Main Zone – Inferred			
Oxide	72	0.015	0.257
HW Type	110	0.017	0.283
Dyke Adit – Measured & Indicated			
Oxide	1,972	0.025	0.491
HW Type	636	0.028	0.344
Dyke Adit – Inferred			
Oxide	151	0.007	0.042
HW Type	42	0.008	0.060
East Hill – Inferred			
Oxide	1,159	0.018	0.048
HW Type	27	0.009	0.060

Notes:

- Within Pit Shell #49, RF0.83.
- The following cut-off grades have been used in the evaluation:
  - Oxide: 0.006 oz/t Au, 0.720 oz/t Ag;
  - HW Type: 0.007 oz/t Au, 0.564 oz/t Ag;
  - FW Type: 0.008 oz/t Au, 0.752 oz/t Ag.
- Mining Loss & Dilution at 96% and 4% respectively.
- $1150/oz Au and $16/oz Ag metal prices.
- Numbers may not add exactly due to rounding.

16.5 LIFE OF MINE SCHEDULE

A mine production schedule was developed with the main objective of delivering 3.818 million tons of mineralized material per year to the heap leach facility. The mine schedule was developed using the Whittle™ software and the fixed lead option. With the fixed lead option, one specifies a number of benches by which the mining of each pushback is to lead the next one. For the purposes of the PEA schedule, a fixed lead of 5 was used.

The Life of Mine (LOM) Schedule is based on the same parameters as described in the Pit Limit Analysis.

Given this is a PEA level study, final pit designs for the starter pit or pushbacks were not completed. However, some optimization was performed to better understand the quantities of material that may actually be mined during production.

Phase I is the first pit that would be designed from the initial economic pit shells generated by the Whittle™ optimization run. Whittle™ Pit Shell 22 was used as the starter pit for scheduling purposes. The initial economic pit shells prioritize the high-grade resource mining of the resource body, and at the lowest amount of waste stripping. This will maximize cash flow and speed capital recovery during the initial years.

Pit Shell 23 was selected as another pushback. Due to the difference between these two pit shells, a manual shell was created to assist with the sequencing. There is opportunity for optimization of the shape and location of the manual shell in further studies.

Figure 16.4 illustrates the location of the starter pit (coloured red) in reference to the ultimate pit shell (coloured grey).

Table 16.6 shows the LOM schedule by material type.

Figure 16.4 – Starter Pit Shell (red) Compared to Ultimate Pit Shell (grey)

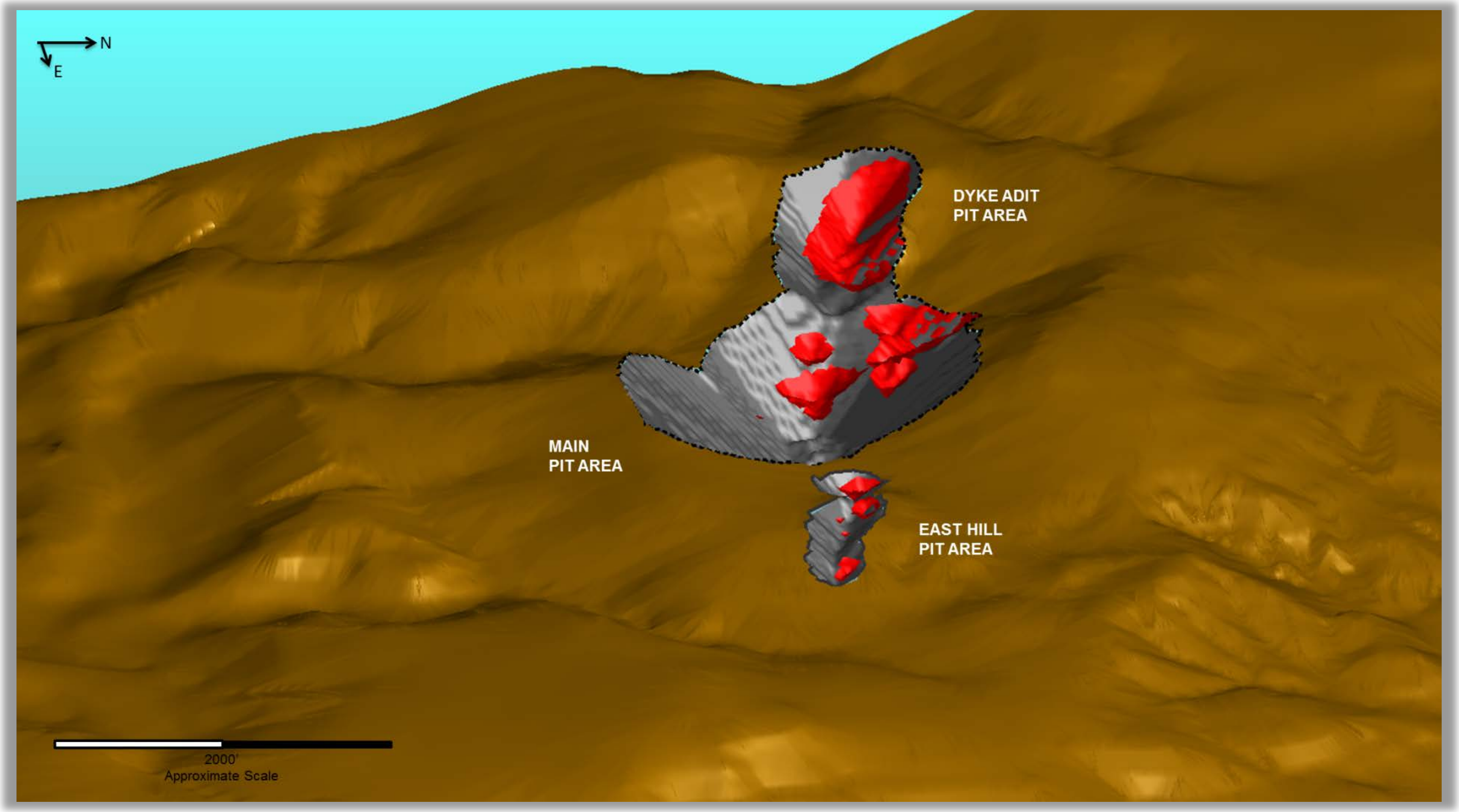


Table 16.6 – Life-of-Mine Schedule, by Zone, by Material Type

Material Type	Units	LOM	Yr. 1	Yr. 2	Yr. 3	Yr. 4	Yr. 5	Yr. 6	Yr. 7	Yr. 8	Yr. 9	Yr. 10	Yr. 11
Total Material Mined	M tons	**102.4**	8.0	12.5	10.6	10.4	14.9	9.9	12.6	8.8	6.3	4.9	3.4
Waste Rock Mined	M tons	**61.0**	4.2	8.7	6.8	6.6	11.1	6.1	8.8	5.0	2.5	1.1	0.2
Heap Leach Feed	M tons	**41.4**	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.2
Au	oz/ton	**0.022**	0.025	0.012	0.013	0.018	0.017	0.022	0.023	0.027	0.028	0.028	0.026
Ag	oz/ton	**0.339**	0.392	0.198	0.214	0.296	0.271	0.348	0.344	0.475	0.469	0.396	0.334
Heap Leach Feed by Zone, by Material Type													
Bear Creek HW		**13.3**	0.3	1.3	1.6	1.3	0.4	0.7	1.1	2.1	1.1	2.0	1.5
Oxide	M tons	**1.9**	0.3	1.1	0.2	0.1	0.0	0.0	0.0	0.1	0.0	0.0	0.0
Au	oz/ton	**0.014**	0.014	0.011	0.014	0.025	0.019	0.019	0.015	0.022	0.023	0.027	0.027
Ag	oz/ton	**0.199**	0.281	0.154	0.153	0.172	0.116	0.329	0.149	0.327	0.467	0.715	0.348
HW Type	M tons	**11.5**	0.0	0.2	1.4	1.2	0.4	0.7	1.1	2.0	1.1	1.9	1.5
Au	oz/ton	**0.026**	0.015	0.030	0.015	0.020	0.020	0.026	0.019	0.029	0.031	0.030	0.033
Ag	oz/ton	**0.411**	0.258	0.400	0.221	0.444	0.369	0.657	0.347	0.534	0.414	0.400	0.361
Bear Creek FW		**13.5**	0.1	1.2	1.3	1.3	0.9	1.4	1.4	1.3	2.0	1.2	1.3
Oxide	M tons	**2.0**	0.1	1.1	0.2	0.2	0.0	0.0	0.1	0.0	0.1	0.2	0.0
Au	oz/ton	**0.014**	0.020	0.012	0.011	0.018	0.000	0.005	0.009	0.025	0.027	0.019	0.019
Ag	oz/ton	**0.278**	0.278	0.214	0.187	0.320	0.000	0.229	0.271	0.744	0.799	0.445	0.299
FW Type	M tons	**11.5**	0.0	0.2	1.0	1.1	0.9	1.4	1.4	1.3	1.9	1.0	1.3
Au	oz/ton	**0.023**	0.008	0.012	0.014	0.018	0.020	0.022	0.025	0.028	0.028	0.026	0.020
Ag	oz/ton	**0.373**	0.346	0.269	0.274	0.275	0.459	0.310	0.358	0.438	0.508	0.364	0.314
Main Zone		**10.6**	2.2	1.3	0.8	1.0	1.4	0.8	0.8	0.5	0.7	0.7	0.4
Oxide	M tons	**4.3**	2.1	1.2	0.4	0.4	0.0	0.0	0.1	0.0	0.0	0.0	0.0
Au	oz/ton	**0.018**	0.026	0.010	0.011	0.012	0.019	0.015	0.015	0.018	0.007	0.013	0.000
Ag	oz/ton	**0.288**	0.375	0.185	0.185	0.250	0.111	0.206	0.323	0.225	0.058	0.282	0.000
HW Type	M tons	**6.3**	0.1	0.1	0.4	0.6	1.4	0.8	0.7	0.5	0.7	0.7	0.4
Au	oz/ton	**0.019**	0.022	0.010	0.011	0.014	0.015	0.020	0.022	0.021	0.022	0.025	0.020
Ag	oz/ton	**0.257**	0.165	0.172	0.147	0.162	0.154	0.198	0.322	0.350	0.406	0.405	0.300

Material Type	Units	LOM	Yr. 1	Yr. 2	Yr. 3	Yr. 4	Yr. 5	Yr. 6	Yr. 7	Yr. 8	Yr. 9	Yr. 10	Yr. 11
Dyke Adit		**2.8**	1.0	0.0	0.0	0.1	0.6	0.6	0.3				
Oxide	M tons	**2.1**	1.0	0.0	0.0	0.1	0.6	0.4	0.0				
Au	oz/ton	**0.024**	0.029	0.007	0.007	0.017	0.014	0.027	0.045				
Ag	oz/ton	**0.458**	0.551	0.037	0.030	0.099	0.352	0.507	0.711				
HW Type	M tons	**0.7**	0.0	0.0	0.0	0.0	0.1	0.3	0.3				
Au	oz/ton	**0.027**	0.017	0.008	0.007	0.084	0.058	0.018	0.027				
Ag	oz/ton	**0.379**	0.222	0.030	0.046	0.488	0.421	0.317	0.462				
East Hill		**1.2**	0.1	0.0	0.1	0.1	0.5	0.3	0.1				
Oxide	M tons	**1.2**	0.1	0.0	0.1	0.1	0.5	0.3	0.1				
Au	oz/ton	**0.018**	0.022	0.000	0.012	0.019	0.013	0.019	0.036				
Ag	oz/ton	**0.048**	0.030	0.000	0.076	0.079	0.059	0.034	0.010				
HW Type	M tons	**0.03**	0.018	0.000	0.002	0.003	0.004	0.000	0.000				
Au	oz/ton	**0.009**	0.010	0.000	0.007	0.009	0.007	0.007	0.000				
Ag	oz/ton	**0.060**	0.059	0.000	0.051	0.062	0.073	0.047	0.000				

Note: Numbers may not add exactly due to rounding.

16.6 WASTE ROCK DISPOSAL

Waste rock storage areas are located toward the northeast and southwest sides of the pit areas in close proximity to the mining areas so as to minimize waste haulage distances. These areas were sized to store a combined 90 Mt of waste rock. Mining from within the pit areas will deliver waste rock to the nearest available storage area.

To prepare these areas, topsoil is removed, stockpiled, and seeded for long-term storage and later use during reclamation. Waste rock is then end dumped from the haul trucks forming 40 to 80 foot lifts. Trucks dump near, but at a safe distance from the edge of the lift. Lifts will be constructed such that the final waste rock storage areas will have an overall 3H:1V slope. Waste rock placement and grading during production operations will concurrently work toward developing this final reclaimed slope.

The waste rock storage areas are unlined because test work on waste rock materials to date has demonstrated that the waste rock is substantially net neutralizing with respect to acid generation.

The proposed mine plan will generate 61.0 million tons of waste rock. Assuming a swell factor of 30%, a volume of 104 million cubic yards of waste storage is required.

16.7 PIT WATER HANDLING

The progressive development of the open pit will result in increasing water infiltration from precipitation and groundwater inflows. As the pit deepens and increases in footprint, it will be necessary to control water inflow through the construction of in-pit dewatering systems such as dewatering wells, drainage ditches, sumps, pipelines, and pumps.

Based on the historical report title “Evaluation of the Baseline Hydrology and Prediction of Hydrologic Conditions during Operation and Closure”, August 1996, by Water Management Consultants, the envisioned dewatering plan involved dewatering the pit using one or two pumping wells, in conjunction with localized drainage measures installed on an as-needed basis within the pit.

There is an existing production well, PW-1, its condition will need to be verified. It was anticipated that this well would lower the water level to roughly 4,900 ft. elevation. A second, deeper well would be required to dewater the pit at lower levels. The second well would target elevations of 4,550-4,650 ft. It was envisioned that the second well could be drilled from a catch bench within the pit area. The second well would be of similar diameter to PW1 at 8 in. diameter.

The Dyke Adit and East Hill areas are above the water table.

16.8 MINE EQUIPMENT SELECTION

Various combinations of mining equipment were considered. Ultimately, the conceptual plan considered the use of 27 ft. single pass blasthole drills, 16.5 cubic yard loaders, and 100 ton mine haul trucks. These fleet components, in combination, were selected as the most appropriate production equipment considering the mine plans and current understanding of the deposit areas.

Production drilling by an Atlas Copco (AC) DM-45 (or equivalent) on 20 ft. benches will produce 15 ft. x 15 ft. square blast patterns with 6.75 in. holes. Pre-split drilling with a FlexiROC D55 (or equivalent) adjacent to the final pit walls will minimize overbreak and maintain pit slope stability. This equipment will also be used to drill dewatering holes later in the mine life.

A reputable surface mine contractor that specifically uses both the mining methods described and the select fleet was contacted and asked to prepare a conceptual proposal to perform the mining and various ancillary services at the Property. The contractor visited the site and was given a generalized site plan, a topographic map of the area, and a conceptual pit plan showing the locations of waste dumps and crushing facilities.

The contractor currently uses a combination of AC DM-45 production drills, CAT 992 – 16.5 cubic yard loaders, and CAT 777 – 100 ton mine haul trucks at multiple active and ongoing operations in Nevada. Additional equipment on site will include CAT D10 dozers, CAT 14G graders, a CAT 773 water truck, a CAT 330 excavator, an IT28 material tele-handling equipment, a mechanics truck, a fuel / lube service truck, light plants, and small generators.

Capital and operating cost estimates (CAPEX and OPEX) to mine the deposit were subsequently prepared in conjunction with the contractor and presented to Timberline management.

16.9 PIT OPERATION PERSONNEL

A conceptual estimate of manpower to mine the Talapoosa deposit is presented in Table 16.7.

Table 16.7 – Mine Manpower Estimate

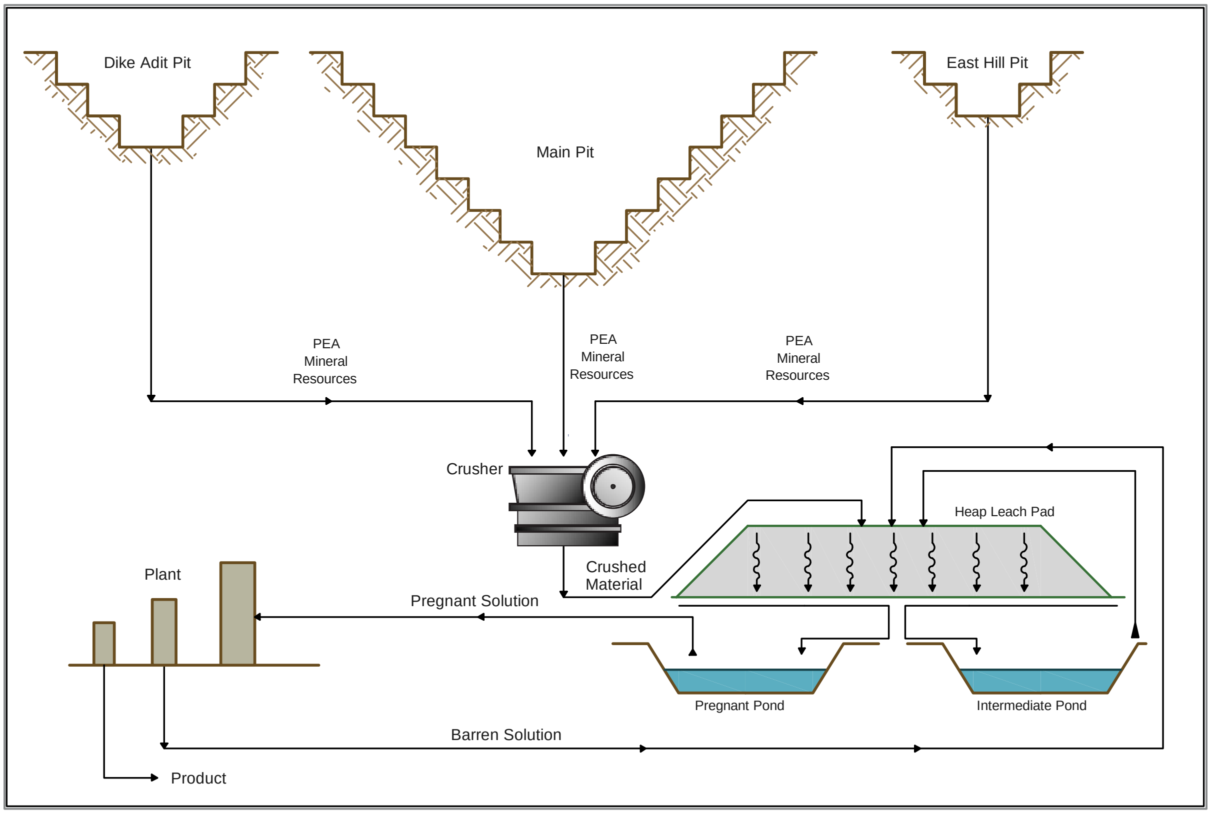
Crew	1 (Day)	2 (Night)	3 (Day)	4 (Night)
12 hr/shift – 7 day per week				
# of Shifts per Day	1	1	1	1
Sub Contract Blasting Crew – Blaster, Laborers	2	0	2	0
Foreman/Shifter	1	1	1	1
Drilling Crew – Driller	1	1	1	1
Cat 992 Loader Operator	1	1	1	1
Cat 330 Excavator Operator / Utility	1	0	1	0
Cat D10 Dozer Operator	1	1	1	1
Cat 14G Grader / Water Truck Operator	1	1	1	1
Cat 777 100 ton Haul Truck Operators	4	4	4	4
Mechanic	1	0	1	0
Service Person	1	1	1	1
Total Hourly Employees	12	10	12	10
Total Hourly Subcontractor Employees	2	0	2	0
Total Salaried Employees	3	1	3	1
Total Manpower	17	11	17	11

17 RECOVERY METHODS

17.1 OVERVIEW

A conceptual scheme to process resource material within the PEA pit shell at Talapoosa was developed for the PEA. The scheme is based on results from earlier metallurgical tests performed on mineralized material. A conceptual flowsheet of the heap leach process is presented in Figure 17.1.

Figure 17.1 – Overall Process Diagram



Gold and silver from the Talapoosa deposits will be recovered using industry standard heap leach cyanidation and gold precipitation techniques.

Resource material within the PEA pit shell as delivered from the mine will be crushed, agglomerated, and stacked on a heap. Cyanide leach solution will be distributed over the stacked heap and percolate through the material. Leach solution containing the gold and silver (pregnant solution or “preg-solution”) will collect at the base of the heap in the overliner-piping network and flow into the preg-pond.

Conceptual design criteria for this flowsheet are presented in Table 17.1.

Table 17.1 – Preliminary Process Design Criteria

Material Bulk Density	100	lbs/ft^3
Heap Lift Height	20	ft.
Primary Leach Cycle Duration	60	days
Primary Leach Solution Application Rate	0.004	gpm/ft^2
Intermediate Leach Cycle Duration	60	days
Intermediate Leach Solution Application Rate	0.004	gpm/ft^2
Primary Area Under Leach	925,000	ft^2
Intermediate Area Under Leach	925,000	ft^2

Preg-solution will be pumped to plate and frame clarifying filters in the plant. Clarified preg-solution is deaerated and combined with zinc powder and lead nitrate immediately before being pumped into plate and frame precipitation filters. Gold and silver, both being substantially more nobel metals than zinc, will precipitate out of solution while the zinc powder will go into solution. Lead nitrate is a minor (often not needed) catalyst to the process providing an electrochemical conduit for the Merrill Crowe reaction to occur.

Gold and silver bearing precipitates are discharged into in-plant shuttle containers when removed from the plate and frame filters. The precipitate and the container are moved into a mercury recovery retort where mercury vapour, if contained in the precipitate, is volatilized and collected through chilled decantation.

Mercury free precipitate is placed into ceramic crucibles with various fluxes and charged into a smelting furnace. The precipitate melts in the crucible wherein doré settles at the bottom and slag forms at the top. Once firing is complete the slag is poured off the top of the melt and the doré is poured into bars or buttons.

Upon cooling and solidification the doré is knocked loose from its' mold at which time it is weighed, stamped, photographed, and secured in a vault.

There is no discharge of process solutions from the facilities. Barren process solution remaining after precipitation of the gold and silver is adjusted in pH and cyanide content, ultimately being recirculated to the heap.

17.2 PRIMARY CRUSHING

A large jaw crusher was selected as the primary crusher to match production needs and rock hardness.

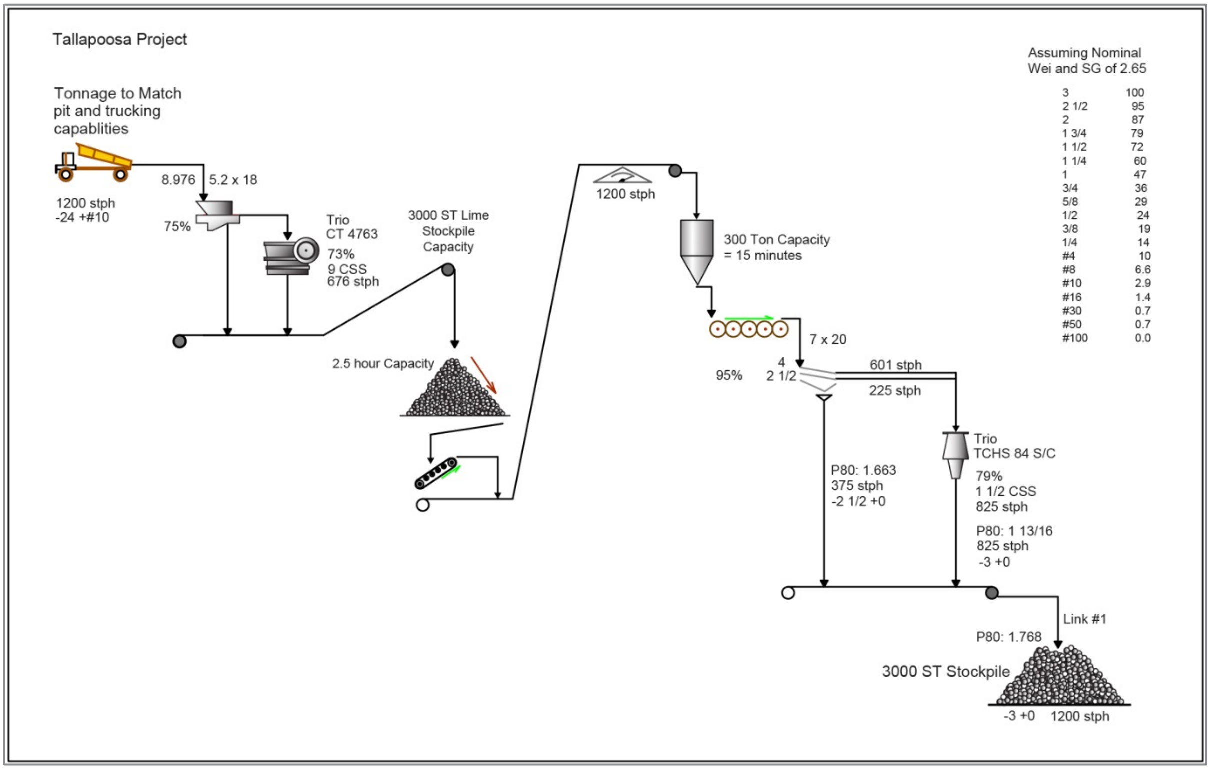
Mineralized material mined within the PEA pit shell will be dumped into a pocket above a vibrating grizzly screen. Oversize will move by gravity and shaking action over the grizzly bars to feed the jaw.

Coarse material, having been crushed by the jaw, will discharge and combine on the same conveyor belt with undersize material that passed through the grizzly bars. Crushed material will be stacked in a 3,000-ton live capacity primary crushed stockpile. Figure 17.2 depicts the first two crushing stages of the comminution process.

17.3 SECONDARY CRUSHING

Primary crushed material will be extracted from under the stockpile by an apron feeder housed in a steel reclaim tunnel and will be transferred to a conveyor that feeds a vibrating secondary screen above the secondary heavy-duty standard cone crusher. Coarse material that does not pass the screen will discharge into a day-bin above the cone crusher. The coarse material will be crushed, combined with fines that passed the secondary screen and stacked in a 3,000-ton live capacity secondary crushed stockpile.

Figure 17.2 – Primary and Secondary Crushing Diagram



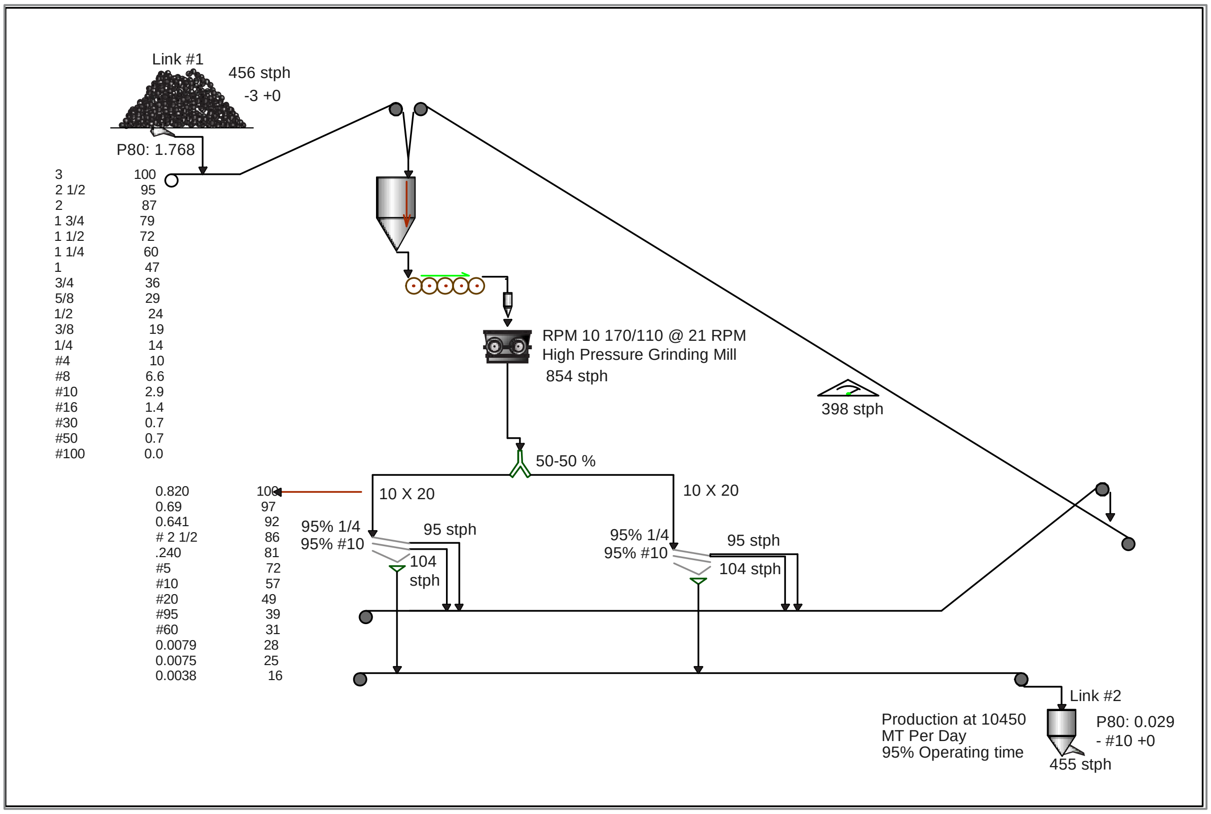
17.3.1 HPGR CRUSHING

Product from secondary crushing and screening will be extracted from under the stockpile by a vibrating pan feeder housed in a steel reclaim tunnel and transferred to a conveyor that feeds a day bin. A vibrating pan feeder delivers material to the high pressure grinding roll (HPGR).

HPGR discharge is screened on twin – double deck screens to remove the fine product. Coarse material that does not pass the screen will discharge back into the day-bin above the HPGR to be combined with fresh feed material.

A nominal 10-mesh product will be collected on a conveyor belt and stacked in a 3,000-ton live capacity HPGR crushed stockpile. Figure 17.3 graphically depicts the fine crushing HPGR circuit.

Figure 17.3 – HPGR Circuit



17.4 AGGLOMERATION

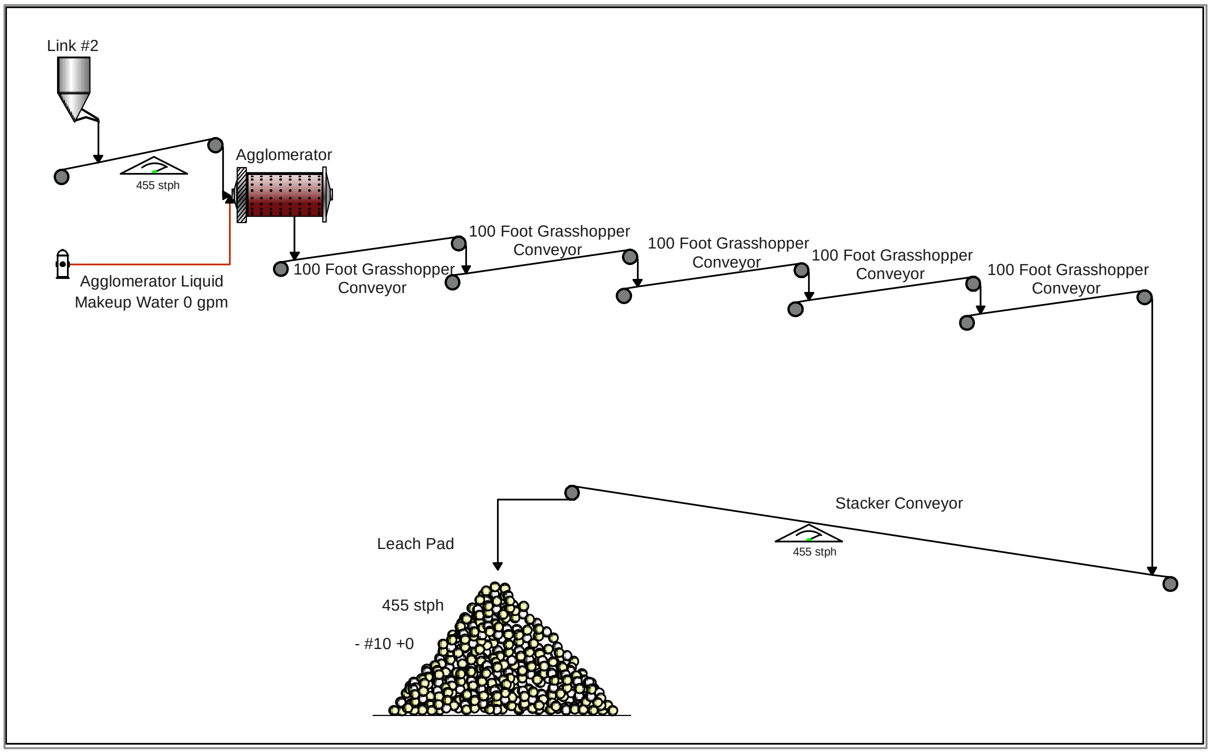
HPGR crushed material will be extracted from under the stockpile by a vibrating pan feeder housed in a steel reclaim tunnel and transferred to a conveyor that feeds a day bin. A vibrating feeder delivers material to a weight-indexed belt on which the fines produced by the HPGR are combined with lime and cement.

Water is added to the feed end of the agglomeration drum, wetting the combined materials and increasing their moisture content.

17.5 HEAP STACKING

The agglomeration drum will discharge onto a conventional transfer conveyor and subsequent mobile "grasshopper" conveyors that carry the agglomerates to a radial stacker with a 20 ft. stinger. The agglomerates will be staked to a 20 ft. heap height. Figure 17.4 presents a graphic depiction of the agglomeration and stacking circuit.

Figure 17.4 – Agglomeration and Stacking Diagram



17.6 LEACHING AND SOLUTION IRRIGATION

Barren solution, after adjusting for pH and cyanide concentration will be pumped at 3,700 gpm from a barren pond, located near the plant, to the once leached material on the heap. This "once leached" material will receive secondary leaching at an average application rate of 0.004 gpm/ft^2. The intermediate leach solution (ILS) generated from secondary leaching will percolate through the mineralized material taken from the PEA pit shell, be collected in an overliner piping collection system, and be discharged by gravity flow into the intermediate solution pond. From this pond, it will be pumped back onto the heap for primary leaching of "fresh" resource material within the PEA pit shell at the same flowrate and application rate.

Solution from the primary leach cycle will be collected in the pregnant solution pond. This pregnant solution will be pumped to the Merrill Crowe plant for processing. After processing, the now barren solution will return to the barren pond, be adjusted for pH and cyanide concentration, and be recycled back to the heap. Both the primary and secondary leach cycles will be 60 days in duration.

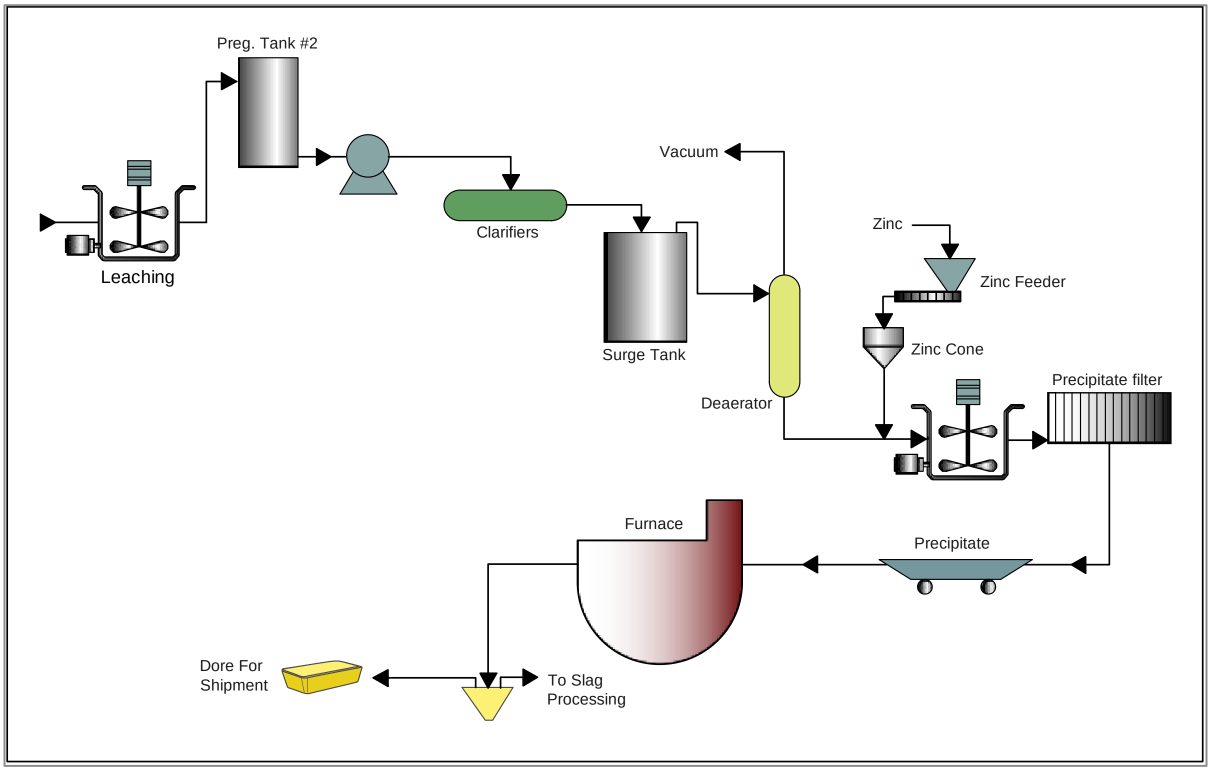
17.7 SOLUTION COLLECTION PONDS

Preg-solution collected in the overliner piping system discharges to the pregnant solution pond. Solution ponds (pregnant and intermediate) and storm event ponds will be located toward the south end of the heap leach pad. The process ponds will be double lined with leak detection and be capable of storing operational process solutions as well as an 8-hour drain down of the heap should power be lost. The event pond will have the capacity to contain a 25-year 24-hour storm event along with a power outage over 8 hours. The storm pond will be single lined.

17.8 GOLD AND SILVER RECOVERY PLANT

Gold and silver will be recovered in a Merrill Crowe process plant. A conceptual flowsheet of the Merrill Crowe process plant is presented in Figure 17.5.

Figure 17.5 – Merrill Crowe Process Plant Flowsheet



Pregnant solution will be clarified and stored in a tank. Clarified solution will be pumped to a de-aeration tower wherein residual oxygen in the pregnant solution is removed by vacuum.

Zinc powder is added to precipitate the gold and silver. Lead nitrate may be added to assist with the gold and silver precipitation reaction.

Gold and silver precipitates are removed in a plate and frame precipitate filter. Diatomaceous earth and/or other filter aids are used to coat the precipitate filter to “catch” fine gold and silver precipitates.

Raw gold and silver precipitate is transferred via a wheeled tub to a mercury vapor condensation retort wherein any mercury that may be present in the precipitates is removed and collected.

Mercury-free precipitate is transferred to a large crucible and charged into a smelting furnace with the appropriate fluxes. The smelting process liquefies the precipitates with doré forming in the bottom of the crucible and slag forming on top.

The slag is poured off and cooled into glass. The doré is then poured into molds, forming buttons or bars. The doré is cooled, sampled, stamped, photographed, and stored in a vault for holding until being shipped to a master refiner.

17.9 PROCESS WATER BALANCE

A process water balance was developed to determine makeup water requirements, the solution flow rates within the system, and the size the pumps, pipes, pond volumes, and other pad and pond components.

Based on the production of 10,460 tpd mineralized resource material from the pit, the solution flowrate from the heap to the process plant will be 3,700 gpm. Pumps, pipes, and pond volumes will be capable of efficiently storing or transporting this volume of process solution. The solution ponds are conceptually designed to contain all of the runoff from a 25-year 24-hour storm event in addition to the process water. A summary of the amount of precipitation that would be anticipated to fall for a specific storm event is presented in Table 17.2.

Table 17.2 – Return Period and Intensity

Storm Event	Rainfall (in)
10-year 24-hour	1.6
25-year 24-hour	2.0
100-year 24-hour	2.4

The climate at the site is described as High Desert. The average precipitation is 13.4 in. per year. Snowfall (calculated water equivalent) can account for up to one-third of the total precipitation for the year. Snowfall of over 20 in. has occurred near the site, but storms of these magnitudes are rare. The project site is easily accessible year round with proper maintenance of the roads.

Evaporation at the site is high. The average annual evaporation is 50 in. Because the evaporation is significantly higher than the precipitation at the site, the water balance for the site is negative. Extra water will need to be added into the system to account for evaporation and to increase the moisture content of the mineralized material during leaching. Approximately 2,000,000 ft^2 of mineralized material within the PEA pit shell will be under leach at any given time. Based on this area and the as-mined moisture content of the mineralized material, makeup water of approximately 500 gpm will need to be developed.

After large precipitation events, excess water stored in the ponds will be used as fresh water makeup. The fresh water from the wells will be reduced or stopped until the amount of water in the ponds has been lowered to standard operational levels.

17.10 CYANIDE DETOXIFICATION

Detoxification of residual cyanide in the heap and the process solutions will commence through natural degradation. The bulk of the cyanide will degrade naturally over time with more stringent chemical detoxification processes possibly being applied if needed before facilities closure.

Cyanide degradation proceeds through continued but intermittent circulation of process solution through the heap. Merrill Crowe operations are suspended and the associated unit operations are bypassed.

Natural degradation processes include:

- Volatilization;
- Hydrolysis in soils;

- Microbial degradation;
- Anaerobic biodegradation;
- Complexation.

Natural degradation may be promoted by solution aeration and the addition of common agricultural chemicals to promote microbial growth.

18 PROJECT INFRASTRUCTURE

Infrastructure for the project will need to be constructed. Current temporary infrastructure includes roads and drill pads that have been constructed for exploration purposes.

18.1 EXISTING REGIONAL INFRASTRUCTURE

Alternate route State Highway 95 runs approximately 2.5 miles to the east of the site. The main entrance to the site will be from this highway.

Existing power lines start toward the south end of the Property, traversing north across the Property. New lines to support the current electrical loads, in addition to the Projects' mine, process, and infrastructure requirements, will have a similar alignment as the existing power lines.

Existing power lines are located south of the Property. Lines will be extended to the north to provide power for the facilities.

International commercial airline service is available in Reno approximately 1 hour away. A general aviation landing strip is located in Carson City and a regional airport is located in Silver Springs with a military grade 7,200 ft. strip. A rail line is located to the east of the site.

18.2 PROJECT SITE LAYOUT

The conceptual layout of the project has been developed to maximize efficiency of mine operations and processing while minimizing the overall disturbance of the facilities. Figure 18.1 shows the conceptual layout. Perimeter fencing will be installed around the pads, ponds, mine, and plant facilities to discourage trespassing and keep domesticated livestock off of the mine property.

Access to the site will be from an improved gravel road connecting to Alternative State Highway 95. This access road will be approximately 2.5 miles long and wide enough and at sufficient grades to allow for safe and efficient travel between the mine and the State Highway. The main access road will terminate on site at the office and administration building.

An office and administration building will be located to the north and east of the crusher and plant facilities. A pad will be graded around the East Hill Pit for the crusher, truck shops, administration, and plant facilities. This area will be graveled and be designed for efficient movement of material and traffic. This site is located to the east of the main pit and to the west of the heap leach pad facility.

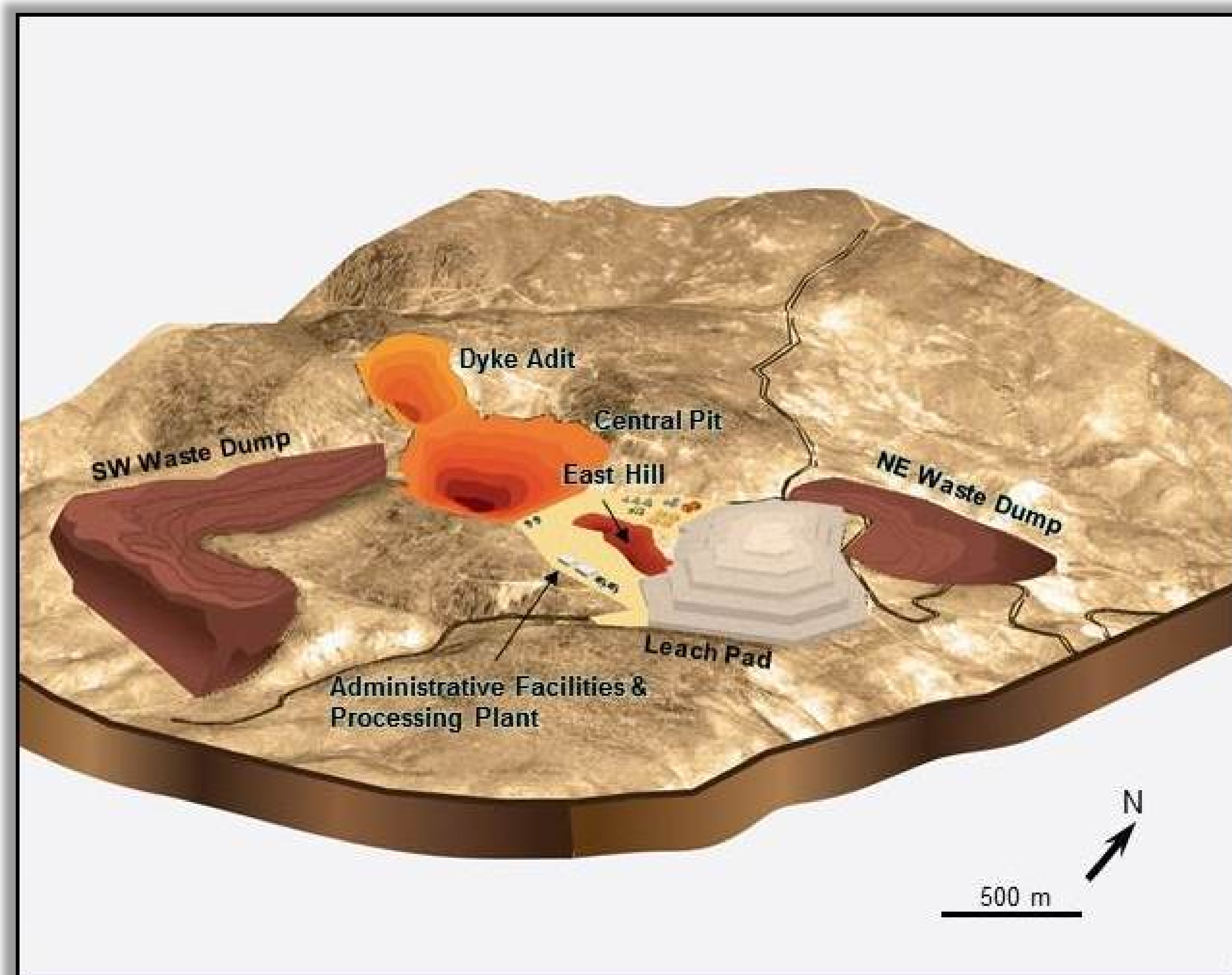
The main pit will be to the west of the crushing and plant site. Haul roads will be constructed out of the pit to efficiently deliver resource material within the PEA pit shell to the crushers and waste to the waste rock storage facilities.

Waste rock storage facilities are located to the northeast and southwest of the plant site. Overburden from mining of the pit will be stored in these locations. Sediment ponds will be constructed at the toes of these facilities to capture sediment that may runoff during storm events.

The heap leach pad facility is located directly to the east of the plant site to efficiently move the material to the pad and get the solutions back to the plant. Resource material from the PEA pit shell will be stacked on the heap using conveyors that will bring the mineralized material from the crushers to a stacker located on the heap. The heap leach facility will be lined with a geosynthetic liner in order to capture the solution and protect the environment. Intermediate, pregnant, and storm ponds will be built to the south of the heap to capture solutions.

An access road will be constructed from the plant site to the south. This road will provide access to the heap leach ponds, the fresh water storage pond, the southwest waste rock storage facility sediment pond, and the fresh water well. The fresh water pipeline and the overhead power for the site will also be located in this corridor. This road could also provide secondary access should the main access road be out of service.

Figure 18.1 – Overall Site Layout



18.3 MINE SITE DEVELOPMENT

Mine site development will commence with an aerial topographic survey to establish a baseline and to enable earthwork volumes to be calculated. Physical activities will begin with clearing, grubbing and topsoil stockpiling from the access roads, main haulage roads (to early development stage), and general site areas including mine facilities, heap leach pads and ponds, process plant, maintenance facilities, warehouse, and offices.

The Project currently considers a contract miner to assist in developing the site in addition to performing production mine operations. Work will include assistance getting water and power to site. The mine contractor may also be considered to assist with the earthworks component of constructing the leach pad and solution ponds.

18.4 PROJECT INFRASTRUCTURE

There is currently no permanent infrastructure at the site. New infrastructure will be developed for this project.

The following infrastructure will be constructed:

- Water – There are several water sources located near the south portion of the Property that may be developed for use by the Project. A water line will be constructed to convey all water to the site including that used for process water, haul road dust control, facilities water, and fire water systems. Bottled water will be provided for the workforce.
- Power – Power will be brought onto the site from an existing high-tension power line located near the south of the Property. About 2.5 miles of overhead power lines and associated transformers will need to be constructed.
- Communications – Telephone and communications lines will be constructed along the access road. Standard mine radio communications will be established.
- Security – A fence will be constructed around the Property to keep trespassers and livestock off of the Property. A security station with weight scale will also be constructed on the main access road.
- Access Roads – Access roads will be established between the site facilities. Access roads will be designed to allow travel by the largest piece of equipment that will be required to construct and maintain the facility.
- Administration and Support Facilities – Administration and support facilities will be constructed on site. These facilities will be in support of the mining and processing operations.
- Sewage treatment infrastructure including septic tanks and leach fields will be developed.
- Assay laboratory.
- First aid and industrial hygiene room.
- Communication and IT systems.

18.5 MINE SITE FACILITIES

Mine site facilities will be provided to support mine operations through proper maintenance and support of the mining fleet and ancillary equipment. Mine site facility components include the following:

- Mine maintenance facility;
- Fuel farm;
- Truck wash;
- Blasting cap magazine, powder magazine, ANFO bin (by Vendor);
- Warehousing of parts;
- Equipment and tool storage;
- Oil and lubricant storage;
- Offices and dries.

The conceptual mine maintenance facility will have multiple Conex boxes arranged to support a fabric-structure building. The Conex boxes will provide secure areas for essentially all uses. A structure such as this will be relatively inexpensive to construct and would meet all mine maintenance requirements while providing office space and storage.

Relatively standardized fuel farms can now be purchased directly from vendors with all components needed to store and dispense fuel. This would be placed in such a location to provide easy access for fuel delivery trucks and the mine operations and maintenance vehicles.

A truck wash would also be constructed at a convenient location, possibly near the maintenance facility, for use by the fleet.

18.6 MINE PIT ACCESS AND HAUL ROADS

Primary haul roads will be located to facilitate haul truck traffic between the Dyke Adit pit, East Hill pit, Main zone pit, the Southwest waste dump, Northeast waste dump, and primary crusher dump block.

Mineralized material in the Dyke Adit pit will be mined first due to its higher grades and sequenced loss of access upon commencement of mining the Main zone. The Main zone and portions of the East Hill mine areas will be mined next to maximize early mineralized material grade to the heap and to provide additional room around the process area.

18.7 ACCOMMODATIONS

There are more than sufficient accommodations in the immediate area to support all project operations considering the Projects' relatively near proximity to the surrounding communities of Reno, Carson City, Fernley, Yerington, and Silver Springs.

19 MARKET STUDIES AND CONTRACTS

No market studies have been conducted and no contracts have been entered into.

The study assumes that a gold-silver doré will be produced on site at Talapoosa. A long-established, dynamic, worldwide market exists for the buying and selling of gold and silver. It is reasonable to assume that the product from the Talapoosa project is saleable.

A selling price of $1,150/oz for gold and $16/oz for silver has been used to develop this PEA. At the end of February 2015, the three-year trailing average, as tabulated from public data from the website www.kitco.com, was $1423/oz for gold and $23.83/oz for silver. The spot price at that time was $1214/oz for gold and $16.53/oz for silver. The selling prices used in this PEA are approximately equal, though lower, to current prices and considerably lower than average prices for the past three years.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The Project has current permits in place for mining and exploration activities. A mining Plan (NVN070011) and NRP (0102) was approved for open pit and heap leaching operations in 1996. The NRP is active for the life of the Project. The BLM's Final EIS for the Project identifies and analyzes the routes for the power line and water line, as well as the location of the water well outside the Project boundary. However, the approved Plan was not modified to include these components.

Exploration activities have been authorized by NDEP and BLM through various permit actions since 1988 with one action currently open in the name of American Gold Capital US Inc. with a $152,568.00 bond for remaining reclamation obligations on the Project.

Other state permits and authorizations will be required for the Project as described in the following sections. An application for a WPCP was originally submitted to the BMRR on September 22, 1995, and the permit was issued on October 31, 1996. This permit has a term of five years and expired on October 31, 2001. A new WPCP will be required.

Air quality operating permits have not been issued for the Project.

The water rights that were acquired for the Project by TMI are no longer valid. Therefore, water rights will need to be obtained for the Project.

20.2 PERMIT ACQUISITION APPROACH

The approach to obtaining the necessary permits to construct, operate, reclaim, and close the Project is based on the development scenario where the planned operations would occur in a manner that is consistent with the Plan. This development scenario would occur within the footprint and have the same mining components that were approved in the Plan and NRP, by the BLM and BMRR, respectively.

To operate the Project as outlined in the approved Plan and the NRP, a WPCP, an air quality operating permit, and a mercury operating permit to construct would need to be obtained from the NDEP. Water rights would also need to be obtained from NDWR. Additionally, the ministerial permits, plans, and notifications outlined in Section 20.4 would need to be addressed. Currently, it appears that 4,000 feet of the power line from the new substation at the existing east-west power line to the Project area and the portion of the water line following the same route are located on public land and not included in the Plan, nor was there a ROWs issued for these facilities. The remaining portion of the water line outside of the Project area will be located on non-public land and principally under the jurisdiction of Lyon County.

For the ROWs under the jurisdiction of the BLM, the BLM will need to comply with NEPA. Since these two facilities were disclosed and analyzed in the EIS for the mining operation, the BLM could comply with NEPA through the Determination of NEPA Adequacy and Land Use Conformance process. Under this process the BLM would use the EIS to support their decision. However, there is the potential that the BLM would determine that the analysis in the EIS is too old and a new NEPA document, likely an environmental assessment (EA), would need to be prepared to support the decision on the ROWs. The assessment in this PEA assumes that an EA will be required.

The WPCP will likely require additional rock characterization. The permit acquisition timeline with the State of Nevada will be approximately six to eight months for the WPCP, plus six months of the characterization work assuming samples are available, and six to eight months for the air quality permits. The permit application preparation timeframes will principally be driven by the engineering design work for the WPCP and the air quality permit. An expected timeframe for this work will be three to four months, which could generally be concurrent with the characterization work.

As part of the existing NRP and approved Plan, a complete reclamation cost estimate (RCE) must be completed and the work coordinated with the BLM and BMRR. A decision must be issued by the BLM before operations can commence. This type of decision is not a Federal Action and is therefore not subject to a NEPA review.

The permit acquisition process for this Project will commence with introduction meetings with the applicable agencies and elected officials. This is particularly important at the local level with Silver Springs and Lyon County. These initial meetings should include the BLM, BMRR, Nevada Bureau of Air Pollution Control (BAPC), and NDWR. At the county level, the first meeting should be with the County Commissioner for the Silver Springs area, then meeting with the Lyon County Planning Department, and the community of Silver Springs.

20.3 MAJOR ENVIRONMENTAL PERMITS AND PROCESSES

In order to construct, operate, reclaim, and close mining operations at the Project, Timberline will be required to obtain a number of environmental and other permits, as well as other decisions, from the BLM, the NDEP, and Lyon County. The principal permits necessary for the mine development are:

- The Bonding Decision for the approved Plan and NRP from the BLM and BMRR, respectively;
- ROWs with the BLM for portions of the power line and water line;
- The WPCP with the BMRR;
- The Air Quality Operating Permit with the BAPC;
- The Mercury Operating Permit with the BAPC;
- Water rights from the NDWR;
- The special use permit from Lyon County.

In order to obtain these permits, applications need to be submitted to each agency. In the case of the approved Plan and the NRP, a single RCE report will be submitted to both agencies for the Bonding Decision.

The following sections provide additional detailed information on the principal permits necessary to develop the Project.

20.3.1 BLM AND BMRR BONDING

In order to commence work under the approved Plan and NRP, Timberline will be required to place a bond with the BLM that is also acceptable with the BMRR. The process to place the bond first requires the submittal of a RCE report to the BLM and BMRR. The RCE report utilizes the Standardized Reclamation Cost Estimator to develop the cost for the reclamation should the agencies need to reclaim the Project. The detailed Project site design information developed for the WPCP will form the basis for the RCE. Once the RCE report is submitted to the BLM and BMRR and they have reviewed and accepted the estimates, the BLM will issue a Bonding Decision and the BMRR will approve the estimate. At this point the bond can be placed with the BLM and operations can begin. This process will likely require four months to complete, once the design information for the WPCP is completed.

20.3.2 BLM ROWS

The Project will require two ROWs, each held by a separate entity. The water line ROW will be applied for and issued to Timberline. The power line ROW will be applied for and issued to NV Energy. Each ROW application will include an SF-299 form and a Plan of Development (POD) that details how the ROW will be utilized. The applications require approximately a month to complete once the design specifics are determined. The NV Energy time frame for the development of the design specifics will likely be many months.

As discussed above, this PEA assumes that the BLM will require an EA to complete the processing of the ROW applications. The EA process is conducted in accordance with NEPA regulations (40 CFR 1500 et. seq.), BLM guidelines for implementing the NEPA in BLM Handbook H-1790-1 (updated January 2008), and BLM Washington Office Bulletin 94-310. The intent of the EA is to assess the direct, indirect, residual, and cumulative effects of the Project and to determine the significance of those effects. Scoping is conducted by the BLM and includes a determination of the resources to be analyzed in the EA, as well as the degree of analysis for each resource. The scope of the cumulative analysis is also addressed under the scoping process. Following scoping and baseline information collection, the EA is either prepared by the BLM or prepared by a third-party contractor for the BLM. Once the BLM determines that the EA is complete, a Preliminary EA is submitted to the public for review. Comments received from the public would be incorporated into a Final EA or included in the decision record and Finding of No Significant Impacts. It is anticipated that the EA process will take six months to complete if a third party prepares the document.

20.3.3 WATER POLLUTION CONTROL PERMIT

A WPCP must be procured from the BMRR to construct, operate, and close a mining facility. The contents of the application are prescribed in the Nevada Administrative Code Section 445A.394 through 445A.399. The WPCP application will be prepared for the Project and will address the following Project components:

- Open pit;
- Waste rock storage;
- Heap leaching with associated process water ponds;
- Merrill Crowe processing;
- Refining;
- A water supply pipeline, associated water delivery pipelines, and power;

- A power substation and distribution system;
- Access and haul roads;
- Ancillary facilities that include the following: storm water diversions; sediment control basins; reagent and fuel storage; fresh water storage; monitoring wells; a meteorological station; and solid and hazardous waste management facilities to manage wastes.

The WPCP application will include an engineering design for waste rock storage areas and heap leach facilities, waste rock characterization reports, hydrogeological summary reports, engineering design for process components including methods for the control of storm water runoff, and containment reports detailing specifications for containment of process fluids. The WPCP Application will also contain the appropriate WPCP plans, including a process fluid management plan, a monitoring plan, an emergency response plan, a temporary closure plan, and a tentative plan for permanent closure of the mine.

The time frames for the preparation of the WPCP application are driven by the time necessary to complete the engineering design for the Project and the associated Engineering Design Report, as required under NAC 445A.397. This process is likely to take six months. The BMRR will take six to eight months to process the application and issue the permit.

20.3.4 AIR QUALITY OPERATING PERMIT

The Project will require a Class I Operating Permit to Construct (OPTC), because the mining plan for this property includes components that have the potential to emit mercury. The application for this permit is made using BAPC forms, and includes a description of the facility, a detailed emissions inventory, and air quality modeling. The application also includes specific equipment locations, plot plans, and process flow diagrams.

The BAPC will issue an initial completeness determination within 30 days of receiving the permit application for the Class I OPTC, and any deficiencies in the application will be addressed at that point. The BAPC will then perform a technical review of the application and when complete, issues a draft permit. The operator reviews this draft permit and if deemed acceptable for operations, a final permit is issued. The permit issuance process is between six and nine months.

20.3.5 MERCURY OPERATING PERMIT

The Project also requires a Mercury Operating Permit to Construct (MOPTC). The application for this permit is made using BAPC forms, and includes a description of each facility, a detailed emissions inventory, and a maximum achievable control technology (MACT) assessment. The MACT assessment is based on the Project’s detailed engineering design of the mercury emission control system. The application also includes specific equipment locations, plot plans, and process flow diagrams.

The BAPC will issue an initial completeness determination within 30 days of receiving the permit application, and any deficiencies in the application will be addressed at that point. The BAPC will then perform a technical review of the application and when complete, issue a draft permit. This permit is reviewed by the operator and if deemed acceptable for operations, a final permit is issued. The permit issuance process is between six and nine months.

20.3.6 NDWR WATER RIGHTS

The Project is located within hydrologic basin number 102, Churchill Valley that the NDWR has identified as a designated basin with preferred use and irrigation denied. The Nevada State Engineer issued an order on April 22, 2010, that applications for new water rights in the basin would be denied, except in a few, very limited cases. The basin currently has approximately 10,000 acre-feet of issued underground water rights. Most of these rights are for quasi-municipal and irrigation. Therefore, the Project will need to purchase or lease water rights and then change the place and manner of use. Timing to have water for the Project will depend on the negotiations with a willing seller and the transfer process with NDWR. Once the water rights are purchased or leased, the transfer process can be a one to four month process. However, if the transfer is appealed, the time frame could be extended to nine to 12 months.

20.3.7 LYON COUNTY SPECIAL USE PERMIT

A special use permit (SUP) from Lyon County is required for the Project, as well as the use of public county roads for the water line to the Project area. The process to obtain a SUP includes the submittal of a SUP application, community meetings, a hearing with the Planning Commission, and a hearing with the County Commission. The timing for the issuance of a SUP is approximately six months from application preparation to SUP approval.

20.4 OTHER MINISTERIAL PERMITS

In addition to the major environmental permits outlined above, Table 20.1 lists other notifications or ministerial permits that would likely be necessary to operate the Project.

Table 20.1 – Ministerial Permits, Plans, and Notifications

Notification/Permit	Agency	Timeframe	Comments
Mine Registry	Nevada Division of Minerals	30 days after mine operations begin	
Mine Opening Notification	State Inspector of Mines	Before mine operations begin	
Solid Waste Landfill	Nevada Bureau of Waste Management	180 days prior to landfill operations	
Hazardous Waste Management Permit	Nevada Bureau of Waste Management	Prior to the management or recycling of hazardous waste	
General Storm Water Permit	Nevada Bureau of Water Pollution Control	Prior to construction activities	
Hazardous Materials Permit	State Fire Marshall	30 days after the start of operations	
Fire and Life Safety	State Fire Marshall	Prior to construction	
Explosives Permit	Bureau of Alcohol, Tobacco, Firearms, and Explosives	Prior to purchasing explosives	Mining contractor may be responsible for permit
Notification of Commencement of Operation	Mine Safety and Health Administration	Prior to start-up	
Radio License	Federal Communications Commission	Prior to radio use	

21 CAPITAL AND OPERATING COSTS

21.1 INTRODUCTION

Capital and operating costs have been estimated for the proposed Project. These costs were developed to support a projected cash flow for the operation, which assesses the Project's economic viability. Capital cost estimates are based on the PEA scenario developed and address the engineering, procurement, construction and start-up of the mine and processing facilities, as well as ongoing sustaining capital costs. Operating cost estimates include the cost of mining, processing, waste management, reclamation, and related general and administrative (G&A) services.

The capital and operating cost estimates were developed for a conventional open pit mine, heap leach process facility using Merrill Crowe recovery, and supporting infrastructure for an operation capable of treating 3.82 million tons of material per annum.

All costs are estimated in United States dollars (US$) as of Q1 2015 and, unless otherwise stated, are referred to as "$".

21.2 COST ESTIMATE ACCURACY

Potential variance of actual costs compared to cost estimates developed in this analysis (Cost Estimate Accuracy) is dependent upon the level of engineering, the estimating methodology, and the degree to which project implementation activities have been estimated.

The Cost Estimate Accuracy of the Project cost estimate is deemed to be within a range of -20% / +35% of the overall project costs, as of Q1 2015.

21.3 EXCLUSIONS

The following cost items were not included in this estimate:

- Costs associated with Scope changes.
- Escalation beyond 2015 Q1.
- Financing costs.
- Cost associated with Schedule delays such as those caused by:
  - scope changes;
  - unidentified ground conditions;
  - labor disputes.
- Accommodations for local labor.
- Environmental permitting activities.
- Permits.
- Sunk costs.
- Project development costs (e.g. engineering test work costs, exploration costs, feasibility study costs) have been excluded.

21.4 CAPITAL COSTS

21.4.1 CAPITAL COSTS SUMMARY

The estimate covers the direct costs of purchasing mining facilities, constructing the heap leach pads, ponds, and process facility and development and construction of infrastructure components of the Project. Indirect costs associated with the design, construction and commissioning of the new facilities, Owner's costs, and contingencies have also been estimated, based on percentages of the Direct Capital Cost Estimate.

The total initial capital cost (CAPEX) to bring the proposed project into production, is estimated at $51.2 million. This initial capital cost is inclusive of $7.8 million indirect costs and $5.7 million contingency. With an additional total sustaining capital cost of $2.7 million, including $2.0 million reclamation bond and $0.4 million contingency, the total LOM CAPEX is $51.9 million, including the return of the reclamation bond.

A summary of the initial and sustaining capital requirements are shown in Table 21.1.

Table 21.1 –Summary of Project Capital Cost Estimate

Description	Initial (Year 0 – 1) M $	Sustaining (Year 2 – 11) M $	Total M $
Direct Costs	*37.7*	*0.3*	*38.0*
Mining	2.5	0.8	3.3
Processing	29.6	0	29.6
Site	3.5	1.5	5.0
Reclamation (bond)	2.0	-2.0	0
Indirect Costs	7.8	0	7.8
Contingency	5.7	0.4	6.1
Total Capital Costs with Contingencies	**51.2**	**0.7**	**51.9**

Note: Numbers may not add exactly due to rounding.

21.4.2 MINING

The mining capital costs have been estimated based on a contractor mining scenario. Table 21.2 summarizes the mining capital requirements. The costs shown are presented without any contingency allowance.

Table 21.2 – Summary of Mining Capital Cost Estimate

Category	Total LOM K $
Contractor Mobilization & Demobilization	580
Topographic Survey Update	15
Pre-production and Access Road Development	1,000
Topsoil Removal and Stockpiling	480
Equipment Maintenance Facility, Fuel Storage Facility	650
Water Stand and Delivery System, Electrical Hookups	100
Dewatering	500
Total Mining	**3,325**

Note: Numbers may not add exactly due to rounding.

21.4.3 MINERAL PROCESSING

Mineral Processing capital expenditures include crushing, heap leach pad and ponds, material conveyance, and mineral processing plant.

The basis of estimate for individual items is:

- Crushing – Estimates from major equipment vendor.
- Heap Leach Pad, Ponds, and Pumps – Estimates from similar facilities that are currently being constructed in the western US.
- Conveyors and Stackers – Estimated based on unit costs from vendor.
- Process Plant and Equipment – Estimated based on vendor costs for a similar size plant in the western US.

Table 21.3 summarizes the process capital requirements. The costs shown are presented without any contingency allowance.

Table 21.3 – Summary of Process Capital Requirements

Category	Total LOM K $
Primary Crushing, including equipment, dump pocket, foundations	11,878
Heap Leach Pad	9,960
Ponds	695
Pumps	650
Conveyors and Equipment	2,205
Process Plant Building and Equipment (with Laboratory)	4,200
Total Processing	**29,588**

Note: Numbers may not add exactly due to rounding.

21.4.4 SITE

The capital requirements for the site are based on the site layout as provided in Section 18. Major earthworks including the access roads, plant site were estimated based on quantities estimated from AutoCAD Civil 3D. Utilities were estimated based on lengths taken from the site plan.

Unit costs for the various items were based on current construction of similar facilities currently ongoing in the western US and engineers' best estimates from previous work recently completed of similar size and scope.

Assumptions regarding the cost estimate are as follows:

- Non-potable fresh water is available from a well located south of the Property.
- The main access to the site will be from Alt. 95 as shown on the site plan.
- Power will be connected to an existing source located south of the Property.

Table 21.4 summarizes the site capital requirements. The costs shown are presented without any contingency allowance.

Table 21.4 – Summary of Site Capital Requirements

Category	Total LOM K $
Plant Site Grading and Parking Areas	370
Power, Substations	354
Access Road	256
Utilities (sewage, storm water, non-potable, fresh water supply)	1,300
Buildings	400
Vehicles	2,250
Security	110
Total Site	**5,040**

Note: Numbers may not add exactly due to rounding.

21.4.5 RECLAMATION COST

Reclamation costs have been estimated based on a $/t processed assumption and are reported under the operating costs. The requirements for the Reclamation Bond were not evaluated in detail for the study. The bond would likely be secured with some combination of cash, insurance or similar financial instrument at some annual cost. Allowance for a cash payment (and return of payment at the completion of reclamation) has been made. However, costs of financing the bond have not been included.

21.4.6 INDIRECT CAPITAL AND CONTINGENCY COST ESTIMATES

Indirect capital costs are expected to be low as a result of the following positive factors:

- There is a large amount of historical data, engineering, and test work available.
- The site is in close proximity to a major city (Reno, NV).
- The site has an existing approved BLM Plan and NRP was previously permitted.

Indirect costs included Engineering, Procurement and Construction Management (EPCM) and Owner's Costs. They were estimated at approximately 20% of Direct Capital costs.

Contingency is defined as additional capital costs allowed for over and above the base estimate, to account for unexpected items and unforeseen activities and requirements not anticipated in the cost estimate. Contingencies were factored from the total direct costs as follows:

- 10% contingency on Mining Capital that reflects the budgetary bid from a Mining Contractor;
- 10% contingency on Crusher Capital that reflects the budgetary bid from Vendor;
- 20% contingency on remaining Capital items that reflects the more conceptual nature of the estimates.

The indirect capital and contingency combine to form 27% of the Total Capital Costs.

21.5 OPERATING COSTS

21.5.1 SUMMARY

Operating costs for the entire LOM period is estimated to $440 million. Operating costs are summarized in Table 21.5 for the entire LOM period.

The operating costs shown in this section are presented without any contingency allowance.

Table 21.5 – Summary of Operating Cost Estimate (LOM)

Description	LOM K $	Average Cost (LOM) $/ton Feed	% of Total Costs
Mining	237,802	5.74	51
Processing	155,414	3.75	33
General & Administrative	35,207	0.85	7
Reclamation	11,183	0.27	2
Total Operating Costs	**439,607**	**10.61**	**94**
Royalties and Refining Charges	13,037	0.31	3
Nevada Net Proceeds Tax	17,455	0.42	4
Total Cash Costs (Operating + Royalties and Refining + Net Proceeds Tax)	**470,099**	**11.35**	**100**

Note: Numbers may not add exactly due to rounding.

21.5.2 MINING

Operating costs assumes a Contractor-operated fleet. Table 21.6 summarizes the Mine Operating Cost. These costs have been estimated based on a quote from a reputable surface mine contractor that specifically uses both the mining methods described and the select fleet. The contractor completed a site tour and was given a generalized site plan, a topographic map of the area, and a conceptual pit plan showing the locations of waste dumps and crushing facilities. Capital and operating cost estimates (CAPEX and OPEX) to mine the deposit were subsequently prepared by the contractor and presented to Timberline management.

Table 21.6 – Summary of Mining Operating Cost Estimate

Category	Total LOM Cost Estimate K $	Total LOM Cost Estimate $/ton Mined	% of Total Costs
Drilling	25,610	0.25	11
Blasting	10,245	0.10	4
Loading	28,684	0.28	12
Hauling	81,619	0.80	34
Mine Support	50,198	0.49	21
Mine General	41,446	0.40	17
Total	**237,802**	**2.32**	**100**

Note: Numbers may not add exactly due to rounding.

MINE GENERAL

The following assumptions have been made with respect to the Mine General operating cost estimate:

→ Includes fuel costs, ANFO costs, and allowance for pit dewatering in the later years of the mine life. For this study, fuel costs have been considered at a fixed rate of 3.00$/gal.

LABOR

The labor component to the Mine Operating cost is limited to the Mining Contractor personnel only. The Owner's Mine Technical team are considered within the G&A section.

21.5.3 MINERAL PROCESSING

Operating costs for mineral processing were estimated based on engineers' experience at other sites of similar size and function. The following operating costs include labor, consumables, maintenance, and power.

Table 21.7 summarizes the Process Operating Costs.

Table 21.7 – Summary of Process Operating Costs

Category	Total LOM Cost Estimate K $	Total LOM Cost Estimate $/ton Feed	% of Total Costs
Primary Crushing	61,768	1.49	40
Pumps	8,360	0.20	5
Reagents	29,603	0.71	19
Conveyors	6,653	0.16	4
Drip Lines	2,071	0.05	1
Labor	35,322	0.85	23
Vehicles – Fuel and Maintenance	1,283	0.03	1
Rehandle at Crusher	10,355	0.25	7
Total	**155,415**	**3.75**	**100**

Note: Numbers may not add exactly due to rounding.

21.5.4 GENERAL AND ADMINISTRATIVE

General and administration costs have been estimated by MPDI and WSP based on assumed personnel requirements and typical requirements for Nevada mining operations. The estimate was built up based on personnel salaries, supplies, light vehicle costs, and other service costs. The general services include general management (not included within mining and processing), accounting, human resources, purchasing, health and safety, environment, security, and the Owner's mine technical team. Table 21.8 summarizes the cost estimate.

Table 21.8 – Summary of General and Administrative Cost Estimate

Item	Total LOM Cost Estimate K $	Total LOM Cost Estimate $/ton Feed
Salaries – Administration	11,960	0.29
Salaries – Owner's Mine Technical Team	23,423	0.30
Expenses	10,824	0.26
Total	**35,207**	**0.85**

Note: Numbers may not add exactly due to rounding.

21.5.5 RECLAMATION COST ESTIMATE

The closure, decommissioning, and reclamation of the Project were not estimated in detail at this stage. However, a provision of $11 million was made in the economic analysis for those activities.

21.5.6 OTHER COST ESTIMATE

The Nevada Net Proceeds Tax was estimated and included as part of the selling costs. The item was estimated at a rate of 5% of the revenue after royalties, refining charges, and operating costs, provided operating income was over $4 million per year.

22 ECONOMIC ANALYSIS

22.1 SUMMARY

The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too speculative geologically to have mining and economic considerations applied to them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The PEA economic results are only intended as a preliminary indicator on potential project economics.

A pre-tax and after-tax cash flow evaluation has been generated from the LOM production schedule and capital and operating cost estimates described in previous sections. The results of the evaluation are summarized in Tables 22.1 and 22.2.

The evaluation reflects a Contract Mining scenario for the project. The Contract Mining scenario reduces the capital costs of the Project as the open-pit mining contractor provides its equipment to the Project. The mining operating costs, however, increase, reflecting the contractor equipment ownership costs and profit.

Table 22.1 – Summary of Project Economic Performance

	Commodity Price Assumption		
	Downside Case	**Base Case**	**Upside Case**
Au Price	$1,000/oz	$1,150/oz	$1,300/oz
Ag Price	$14.50/oz	$16.00/oz	$17.50/oz
Pre-Tax:			
NPV @ 5%	$114 million	$184 million	$254 million
NPV @ 8%	$85 million	$145 million	$205 million
NPV @ 10%	$70 million	$124 million	$178 million
IRR	30.4%	48.4%	68.4%
Payback Period (from start of production)	5.4 years	0.9 years	0.8 years
After-Tax:			
NPV @ 5%	$84 million	$136 million	$188 million
NPV @ 8%	$61 million	$106 million	$150 million
NPV @ 10%	$49 million	$90 million	$130 million
IRR	25.4%	38.8%	52.6%
Payback Period (from start of production)	5.5 years	3.1 years	1.0 years

Table 22.2 – Summary of Project Evaluation, Base Case

Description	Units	LOM
Tonnage Heap Leach Feed	M st	41.4
Feed Grade – Au	oz/ton	0.022
Feed Grade – Ag	oz/ton	0.339
Gold Recovery (average)	%	66.0
Silver Recovery (average)	%	52.5
Tonnage Waste Rock	M st	61.0
Production Period	Years	10.8
Stripping Ratio	W/O	1.5
Gold Production	k oz	593
Silver Production	k oz	7,365
Average Annual Gold Production (LOM)	k oz/a	55
Average Annual Silver Production (LOM)	k oz/a	679
Gross Revenue	M $	799.7
Net Revenues [a]	M $	769.2
Total Operating Costs (Mining + Processing + G&A + Reclamation)	M $	439.6
Total Cash Costs (Operating + Refining Charges , Royalties, Net Proceeds Tax)	M $	470.1
Total Capital Costs with contingencies	M $	51.9
Initial Capital Costs	M $	51.2
Sustaining Capital Costs [b]	M $	0.7
Total Cash Costs + Total Capital Costs	M $	522.0
Corporate Tax	M $	69.0
Total Cash Costs + Total Capital Costs + Corporate Tax	M $	590.9

a) Gross Revenue – Selling Costs (including Refining charges, Royalties, Net Proceeds Tax)
b) Includes return of Reclamation Bond

Note: Numbers may not add exactly due to rounding.

22.2 ASSUMPTIONS

All economic metrics presented in Table 22.1 are expressed in Q1 2015 terms.

Key assumptions used in the analysis include:

- Cash flow analysis conducted on the assumption of 100% equity investment and excludes any element of impact of financing arrangements.
- Flat long-term metal prices of $1150/oz for gold and $16/oz for silver.
- Escalation and inflation have been excluded.
- Corporate / Head Office costs have been excluded.

- Salvage value has not been included.
- Selling costs include:
  - 1% royalty;
  - refining, transportation charges were allocated at $8.50 per ounce of recovered Au;
  - 5% Net Proceeds Tax.
- Reclamation costs have been estimated based on a $/st processed assumption. The requirements for the Reclamation Bond were not evaluated in detail for the study. The bond would likely be secured with some combination of cash, insurance or similar financial instrument at some annual cost. Allowance for a cash payment (and return of payment at the completion of reclamation) has been made. However, costs of financing the bond have not been included.
- No pre-stripping expenditures are estimated as the initial mineralized material can be accessed directly.
- Project Development Costs (e.g. engineering test work costs, exploration costs, feasibility study costs) have been excluded.

Capital expenditures, as shown in Section 21, were assumed to be incurred over a one-year period (Year -1), which is reflected in the discounted cash flow calculations.

Annual gross revenue is determined by applying estimated metal prices to the annual recovered metal estimated for each operating year.

In addition to the 5% Nevada net proceeds tax, a US corporate tax rate of 35% was used for the calculation of after-tax cash flow. The 35% tax rate was applied after consideration of the 7-year modified accelerated cost recovery system (MACRS) depreciation for mine, plant and infrastructure equipment and 40-year straight-line depreciation for fixed facilities and structures.

The economic analysis assesses the project on both a pre-tax and after-tax basis. It must be noted that there are many potential complex factors that affect the taxation of a mining project. The taxes, depletion, and depreciation calculations in the PEA economic analysis are simplified and only intended to give a general indication of the potential tax implications; like the rest of the PEA economics, they are only preliminary.

22.3 CASH FLOW ANALYSIS

Table 22.3 provides a summary of annual production and pre-tax and after-tax cash flows while Figure 22.1 provides a graph of the LOM cash flow.

The evaluation indicates:

- A 48% pre-tax internal rate of return and a $184 million pre-tax NPV at 5% discount rate over a 10.8 year mine life.
- A 39% after-tax internal rate of return and a $136 million after-tax NPV at a discount rate of 5%.
- The payback period is expected to occur within 0.9 and 3.1 years from start of production for pre-tax and after-tax respectively.
- Approximately 85% of the gross revenue is derived from gold production, with the remaining 15% from silver production.

Table 22.3 – Evaluation of Base Case Scenario

		LOM	Year											
	Units	**Total**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
METAL PRICING														
Au	$/oz	1,150		1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150
Ag	$/oz	16		16	16	16	16	16	16	16	16	16	16	16
PRODUCTION														
Heap Leach Feed	k t	**41,420**	0	3,818	3,818	3,818	3,818	3,818	3,818	3,818	3,818	3,818	3,818	3,240
Waste Rock	k t	**61,023**	0	4,166	8,719	6,819	6,603	11,111	6,076	8,815	4,962	2,471	1,094	187
Total Mined	k t	**102,444**	0	7,984	12,537	10,637	10,421	14,929	9,894	12,633	8,780	6,289	4,912	3,427
Strip Ratio		**1.47**	0	1.09	2.28	1.79	1.73	2.91	1.59	2.31	1.30	0.65	0.29	0.06
Au Oz Recovered	k oz	**593**	0	74	34	34	44	43	56	55	66	66	68	54
Ag Oz Recovered	k oz	**7,365**	0	707	367	427	607	528	709	679	999	917	837	589
REVENUE														
Gross	M $	**800**	0.0	95.9	44.8	45.7	60.4	58.4	75.5	74.2	91.4	91.1	91.2	71.1
Selling Costs	M $	**30**	0.0	4.5	0.7	0.9	1.9	1.3	2.9	2.5	3.9	4.2	4.4	3.4
Net	**M $**	**769**	**0.0**	**91.4**	**44.0**	**44.8**	**58.5**	**57.1**	**72.5**	**71.7**	**87.5**	**86.9**	**86.9**	**67.8**
OPERATING COSTS														
Mining	M $	**238**	0.0	18.1	28.5	24.2	23.7	33.8	22.8	29.6	21.2	15.3	12.1	8.5
Processing	M $	**155**	0.0	14.3	14.3	14.3	14.3	14.3	14.3	14.3	14.3	14.3	14.3	12.2
G&A	M $	**35**	0.0	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	2.8
Reclamation	M $	**11**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.8	2.8	2.8	2.8
Total OPEX	M $	**440**	**0.0**	**35.7**	**46.1**	**41.8**	**41.3**	**51.3**	**40.3**	**47.2**	**41.5**	**35.7**	**32.4**	**26.3**
CAPITAL COSTS														
Mining	M $	**3**	2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.5	0.0	0.0	0.3
Processing	M $	**30**	29.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Site	M $	**5**	3.5	0.0	0.0	0.0	0.8	0.0	0.0	0.0	0.8	0.0	0.0	0.0
Reclamation (bond)	M $	**0**	2.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	(2.0)
Indirect	M $	**8**	7.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Contingency	M $	**6**	5.7	0.0	0.0	0.0	0.2	0.0	0.0	0.0	0.2	0.0	0.0	0.0
Total CAPEX	M $	**52**	**51.2**	**0.0**	**0.0**	**0.0**	**0.9**	**0.0**	**0.0**	**0.0**	**1.5**	**0.0**	**0.0**	**-1.7**
Pre-Tax Cash Flow	**M $**	**278**	**-51.2**	**55.7**	**-2.0**	**3.1**	**16.3**	**5.8**	**32.2**	**24.5**	**44.6**	**51.2**	**54.4**	**43.2**
US Tax Expense /(Refund)	M $	**69**	0.0	12.7	(4.2)	(1.4)	2.3	0.4	6.2	3.6	10.8	13.3	14.4	10.9
After-Tax Cash Flow	**M $**	**209**	**-51.2**	**43.0**	**2.1**	**4.5**	**14.1**	**5.4**	**26.0**	**20.9**	**33.7**	**37.9**	**40.0**	**32.3**

Note: Numbers may not add exactly due to rounding.

Figure 22.1 – Base Case Scenario Cash Flow (after tax)

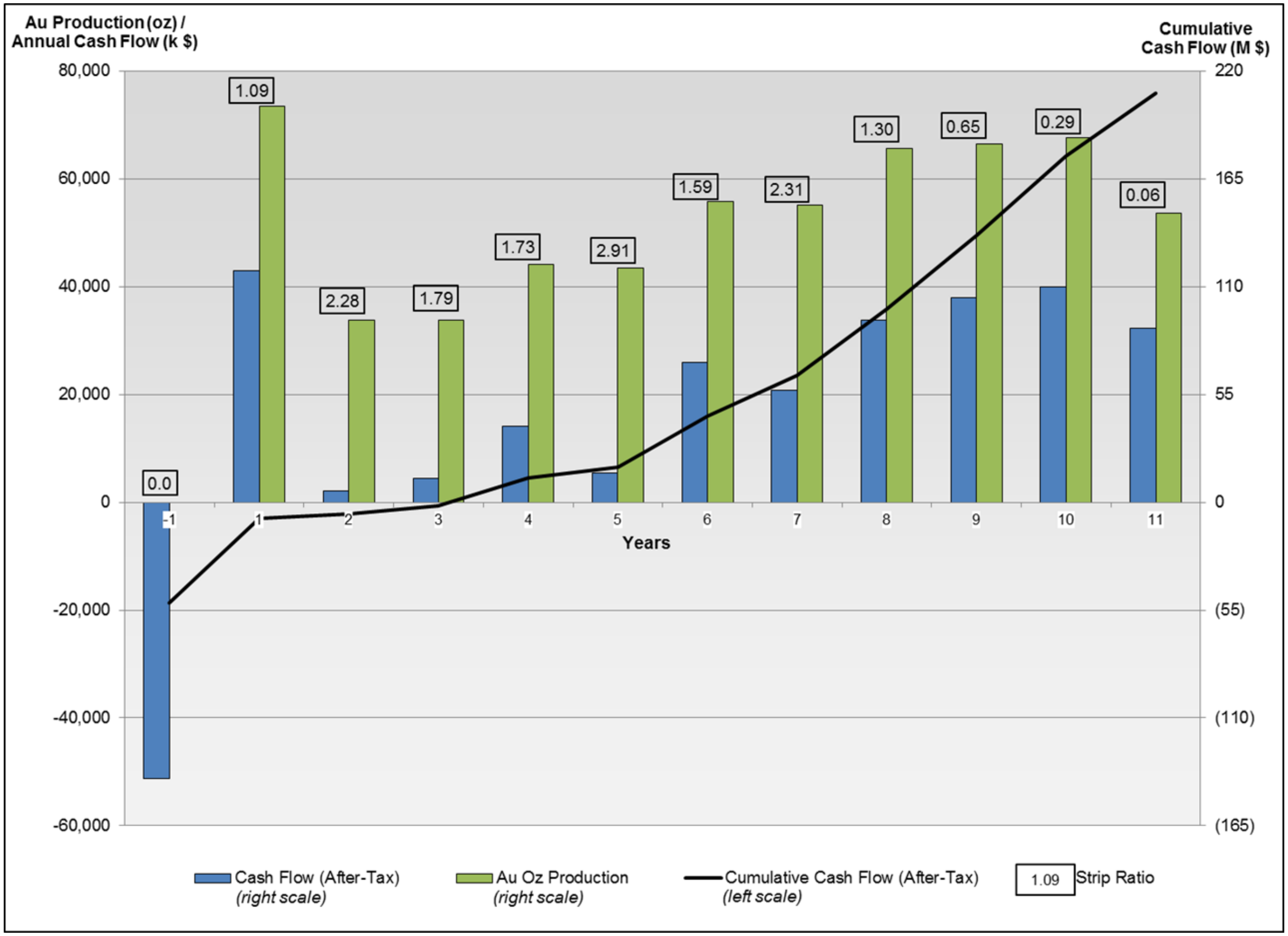


Table 22.4 provides a summary of the LOM costs estimated for the project. The final column of the table expresses the costs per ounce of gold recovered (net Ag) whereby the revenue from the silver production is included as a credit.

Table 22.4 – Summary of LOM Costs

Description	LOM Total Cost (M $)	$/ton Feed	$/oz Au Recovered	$/oz Au Recovered (net Ag) [(d)]
Total Operating Costs [(a)] (Mining + Processing + GA + Reclamation)	439.6	10.61	741	543
Total Cash Costs [(a)] (Total Operating + Refining Charges , Royalties, Net Proceeds Tax)	470.1	11.35	793	594
Total Cash Costs + Sustaining Capital Costs [(a), (b)]	472.8	11.41	797	599
'All-in' Cost [(a) (c)] (Total Operating + Refining Charges, Royalties, Net Proceeds Tax + Initial & Sustaining Capital)	522.0	12.60	880	682
Total Costs [(c)] (Total Cash Costs + Sustaining Capital Costs + Initial Capital Cost + Corporate Tax Costs)	590.9	14.27	997	798

Notes:
(a) Corporate income tax is not included.
(b) Excluding return of Reclamation Bond on Reclamation Costs.
(c) Includes $2M reclamation bond, $6M in contingency, and return of reclamation bond at end of project.
(d) Costs in this column are shown with silver as a credit, at the base case $16/oz silver price.

22.4 SENSITIVITY ANALYSIS

Key economic risks were examined by running cash flow sensitivities to:

- Metal prices;
- Au Recovery;
- Operating costs;
- Capital costs.

Each variable is examined one-at-a-time. After-tax NPV sensitivity over the base case has been calculated for -20% to +20% variations to the key economic parameters. Gold Recovery sensitivity was calculated on -10% to +10%. The sensitivities are shown in Figures 22.2 and 22.3, as well as in Tables 22.5 to 22.9.

Project economics are most sensitive to Gold Price and Gold Recovery. A 10% decrease (increase) in gold prices results in an approximate $35 million decrease (increase) in the after-tax NPV at a 5% discount rate (respectively). Though the sensitivity analysis of gold and silver price was run, silver is a smaller component and has less impact on the value of the Project. The Project is also sensitive to operating costs, and to a lesser extent capital costs. A 10% decrease (increase) in operating cost results in an approximate $24 million increase (decrease) in the after-tax NPV at a 5% discount rate, while a 10% decrease (increase) in capital cost results in an approximate $5 million increase (decrease) in the after-tax NPV at a 5% discount rate (respectively).

Figure 22.2 - Sensitivity Analysis of After-Tax NPV at 5% Discount Rate

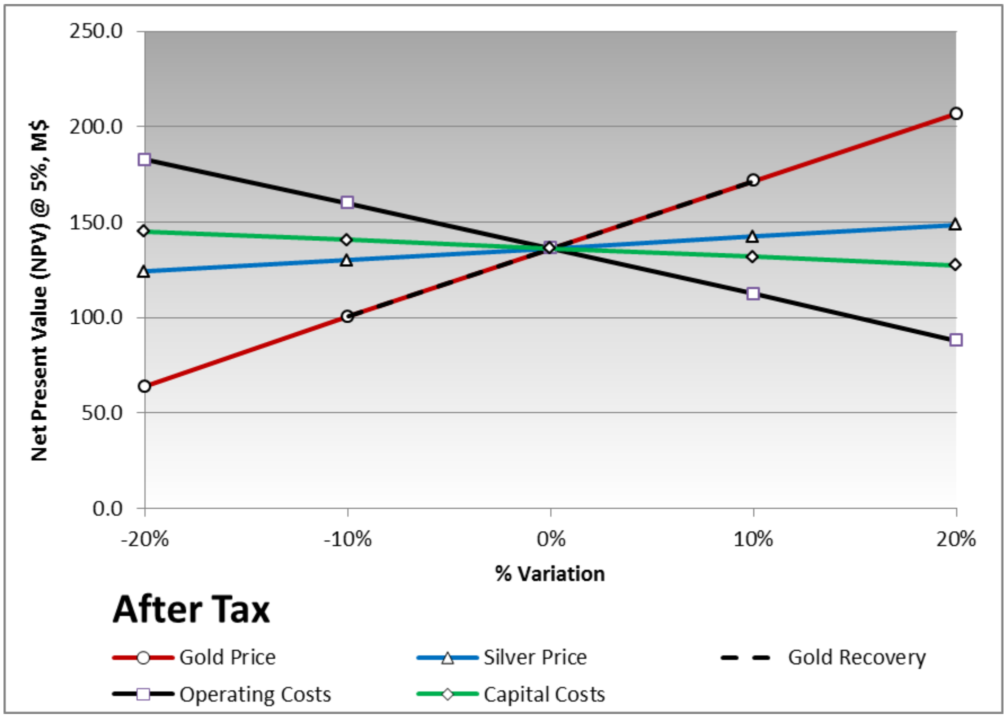


Figure 22.3 - Sensitivity Analysis of After-Tax IRR

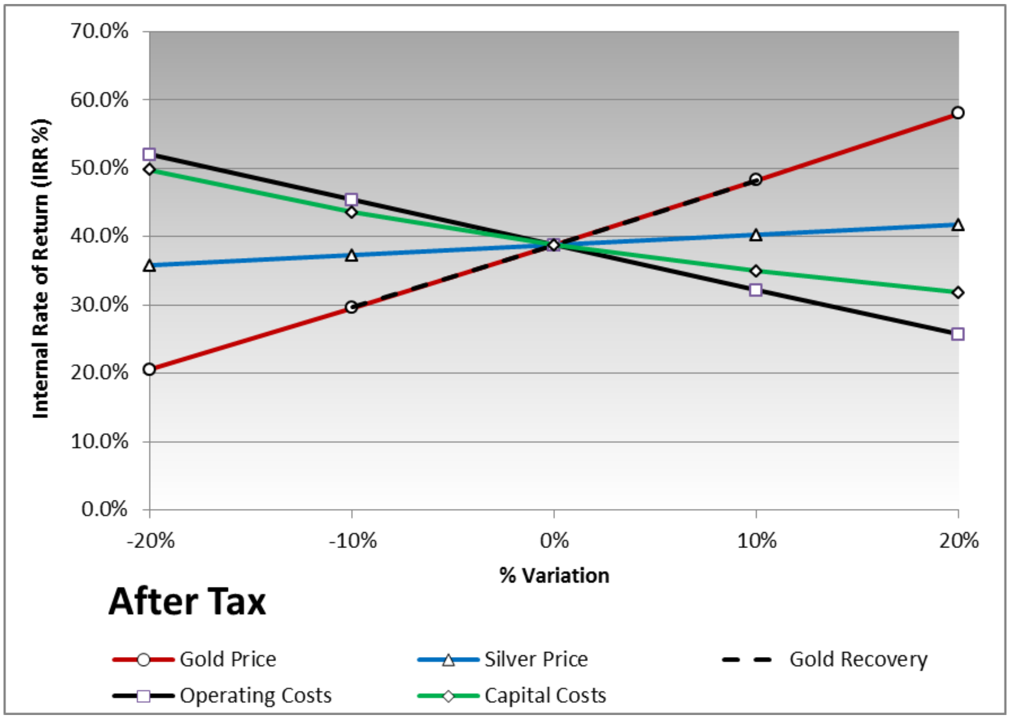


Table 22.5 - Sensitivity Analysis, Gold Metal Prices

Description		Unit	Net Present Value (M $)				
% Variation		%	-20%	-10%	**0%**	10%	20%
Metal Price	Au	$/oz	920	1,035	**1,150**	1,265	1,380
	Ag	$/oz	16	16	**16**	16	16
Pre-Tax							
Discount Rate	0%	M $	148.7	213.4	**277.7**	342.0	406.1
	5%	M $	87.8	136.0	**183.9**	231.7	279.4
	10%	M $	49.8	87.1	**124.2**	161.1	197.9
Internal Rate of Return (IRR)		%	24.2%	35.8%	**48.4%**	62.1%	76.5%
Payback Period (from start of production)		years	6.2	3.9	**0.9**	0.8	0.7
After-Tax							
Discount Rate	0%	M $	112.1	160.7	**208.8**	256.1	303.1
	5%	M $	64.0	100.4	**136.3**	171.7	206.9
	10%	M $	33.8	62.1	**89.9**	117.4	144.7
Internal Rate of Return (IRR)		%	20.6%	29.6%	**38.8%**	48.2%	58.0%
Payback Period (from start of production)		years	6.4	5.1	**3.1**	1.5	0.9

Table 22.6 - Sensitivity Analysis, Silver Metal Prices

Description		Unit	Net Present Value (M $)				
% Variation		%	-20%	-10%	**0%**	10%	20%
Metal Price	Au	$/oz	1150	1150	**1150**	1150	1150
	Ag	$/oz	12.8	14.4	**16.0**	17.6	19.2
Pre-Tax							
Discount Rate	0%	M $	255.5	266.6	**277.7**	288.8	299.9
	5%	M $	167.5	175.7	**183.9**	192.1	200.3
	10%	M $	111.6	117.9	**124.2**	130.4	136.7
Internal Rate of Return (IRR)		%	44.4%	46.4%	**48.4%**	50.5%	52.5%
Payback Period (from start of production)		years	2.5	0.9	**0.9**	0.9	0.9
After-Tax							
Discount Rate	0%	M $	192.2	200.5	**208.8**	217.0	225.2
	5%	M $	124.1	130.2	**136.3**	142.5	148.5
	10%	M $	80.5	85.2	**89.9**	94.7	99.3
Internal Rate of Return (IRR)		%	35.8%	37.3%	**38.8%**	40.2%	41.7%
Payback Period (from start of production)		years	3.4	3.3	**3.1**	3.0	2.7

Table 22.7 - Sensitivity Analysis, Gold Recovery

Description		Unit	Net Present Value (M $)				
% Variation		%	-10%	-5%	**0%**	5%	10%
Average Au Recovery		%	59%	63%	**66%**	69%	73%
Pre-Tax							
Discount Rate	0%	M $	213.9	245.8	**277.7**	309.6	341.5
	5%	M $	136.3	160.1	**183.9**	207.6	231.3
	10%	M $	87.4	105.8	**124.2**	142.5	160.8
Internal Rate of Return (IRR)		%	35.9%	42.0%	**48.4%**	55.1%	62.0%
Payback Period (from start of production)		years	3.9	3.2	**0.9**	0.9	0.8
After-Tax							
Discount Rate	0%	M $	161.1	184.9	**208.8**	232.3	255.7
	5%	M $	100.7	118.5	**136.3**	154.0	171.5
	10%	M $	62.3	76.1	**89.9**	103.6	117.2
Internal Rate of Return (IRR)		%	29.6%	34.1%	**38.8%**	43.4%	48.2%
Payback Period (from start of production)		years	5.1	3.6	**3.1**	2.3	1.5

Table 22.8 - Sensitivity Analysis, Operating Costs

Description		Unit	Net Present Value (M $)				
% Variation		%	-20%	-10%	**0%**	10%	20%
Total Operating Costs		M $	351.7	395.6	**439.6**	483.6	527.5
		$/t	8.5	9.6	**10.6**	11.7	12.7
Pre-Tax							
Discount Rate	0%	M $	361.3	319.6	**277.7**	235.7	193.3
	5%	M $	247.8	215.9	**183.9**	151.7	119.2
	10%	M $	174.6	149.4	**124.2**	98.7	73.1
Internal Rate of Return (IRR)		%	67.0%	57.7%	**48.4%**	39.4%	30.7%
Payback Period (from start of production)		years	0.8	0.9	**0.9**	3.5	5.6
After-Tax							
Discount Rate	0%	M $	268.8	239.0	**208.8**	177.7	146.1
	5%	M $	182.7	159.7	**136.3**	112.3	88.0
	10%	M $	126.9	108.6	**89.9**	70.9	51.5
Internal Rate of Return (IRR)		%	51.9%	45.4%	**38.8%**	32.1%	25.7%
Payback Period (from start of production)		years	1.4	2.0	**3.1**	4.1	5.7

Table 22.9 - Sensitivity Analysis, Capital Costs

Description		Unit	Net Present Value (M $)				
% Variation		%	-20%	-10%	**0%**	10%	20%
Total Capital Cost Estimate		M $	41.5	46.7	**51.9**	57.1	62.3
Pre-Tax							
Discount Rate	0%	M $	288.1	282.9	**277.7**	272.5	267.3
	5%	M $	194.3	189.1	**183.9**	178.7	173.5
	10%	M $	134.5	129.3	**124.2**	119.0	113.8
Internal Rate of Return (IRR)		%	64.3%	55.2%	**48.4%**	43.2%	39.0%
Payback Period (from start of production)		years	0.7	0.8	**0.9**	2.9	3.3
After-Tax							
Discount Rate	0%	M $	217.3	213.0	**208.8**	204.5	200.2
	5%	M $	145.2	140.8	**136.3**	131.9	127.5
	10%	M $	99.0	94.5	**89.9**	85.4	80.9
Internal Rate of Return (IRR)		%	49.7%	43.5%	**38.8%**	34.9%	31.8%
Payback Period (from start of production)		years	1.0	2.3	**3.1**	3.4	3.8

22.5 CONCLUSIONS

Based on the study results, the conclusions are as follows:

- This PEA demonstrates the potential economic viability of the Talapoosa project. The project economics are positive using contract mining.
- The base case scenario shows a rapid payback period.
- Pre-production capital expenditures are relatively low as the mine development and surface infrastructure required to commence heap leach production are not overly extensive. Regional communities provide much of the support services for employees and the mine.
- The project economics are most sensitive to variations in gold price and average gold recovery. The project's breakeven ($0 after-tax NPV @ 5%) gold price and average gold recovery is $721/oz and 41%, respectively.
- 85% of the gross revenue is derived from the gold production while the remaining 15% is derived from the silver production.

23 ADJACENT PROPERTIES

There are no material properties adjacent to the Property.

24 OTHER RELEVANT DATA AND INFORMATION

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

25 INTERPRETATIONS AND CONCLUSIONS

The present study indicates that the Project has positive economics for the production scenario considered in the PEA and based upon the stated assumptions. In WSP's opinion, the PEA shows that the Project has merit, with Mineral Resources of sufficient quantity and quality that support additional investigation at more advanced levels of engineering study (pre-feasibility or feasibility study).

25.1 ECONOMIC ANALYSIS

The economic analysis contained in this report is based in part on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too speculative geologically to have mining and economic considerations applied to them to be categorized as Mineral Reserves. There is no certainty that the economic results of this PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The evaluation indicates:

- A 48% pre-tax internal rate of return and a $184 million pre-tax NPV at 5% discount rate over a 10.8 year mine life.
- A 39% after-tax internal rate of return and a $136 million after-tax NPV at a discount rate of 5%.
- The payback period is expected to occur within 0.9 and 3.1 years from start of production for pre-tax and after-tax respectively.
- Approximately 85% of the gross revenue is derived from gold production, with the remaining 15% from silver production.
- Total Cash Costs + Sustaining Capital Costs for the LOM period is estimated at $472.8 million or $11.41/t heap leach feed material. Included in these costs are the operating costs for mining, processing, general & administrative, and reclamation, as well as refining charges, royalty charges, Nevada net proceeds tax, and sustaining capital costs.
- Initial Capital Expenditures is estimated at $51.2 million, including $7.8 million for indirect costs, $5.7 million in contingencies, and $2 million allowance for reclamation bond.

25.2 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 operators prior to Gunpoint were 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 31.2 Mt with an average grade of 0.032 oz/ton gold and 0.437 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.
- The specific gravity value used to determine that tonnage was derived from a larger data set than used in previous estimates.

25.3 METALLURGY

The conclusions from the historical test work are summarized below.

- The Main zone oxidized mineralization samples as tested showed amenability to column and agitated cyanidation, but generally were not amenable to flotation. This is possibly due to gold and silver being associated with oxide minerals and a lack of refractory sulphides.
- The HW-type unoxidized mineralization from the Main, East Hill, Dyke Adit, and Bear Creek HW zones and the FW-type mineralization of the Bear Creek FW zone were moderately amenable to cyanide leaching, but gave lower recoveries than the oxidized Main zone material.
- Historic and recent metallurgical bottle roll and column testing indicates there is a strong inverse relationship between particle size and gold and silver recovery.
- Based on historic and recent metallurgical testing, heap leach gold and silver recoveries derived from a P_{80} = 10 mesh HPGR crush are estimated by mineralization type as presented in Table 25.1:

Table 25.1 - Leach Recoveries

	Au (%)	Ag (%)
Oxidized (HW and FW types)	77%	47%
HW type (unoxidized)	65	60
FW type (unoxidized)	59	45

- The Bear Creek zone FW-type and HW-type unoxidized samples as tested also showed amenability to flotation, but were less amenable to column and agitated cyanidation, than the Main zone oxidized samples. The presence of sulphides in the Bear Creek HW and FW zones is probably responsible for both the higher flotation recoveries and the lower leach recoveries. Additional testing will be required to determine if this alternate processing method is optimal for the Talapoosa unoxidized material
- The gold occurs 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 is present as acanthite native silver, electrum, and argentian gold.
- Test work indicated that when silver recovery increased, gold recovery increased.
- The presence electrum appeared to cause slow gold and silver leach kinetics.
- Agglomerating heap 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.

25.4 MINING METHODS

There are no Mineral Reserves identified for the Project at this time, only resources in the Measured, Indicated and Inferred categories that were used for this PEA.

The pit optimization and mine plan were based on $1,150/oz gold and $16/oz silver prices. There was no detailed mine design for the project incorporating bench design and haul ramp design.

The project has an existing approved Plan which was prepared in 1996 and filed with the BLM. The Plan permitted 42 million tons of mineralized material to be processed at the heap leach facility prescribed in the Plan. Therefore, a pit shell with less than 42 million tons was selected for the ultimate pit limit. The pit shell at revenue factor 0.83 was selected for the ultimate pit limit. The pit limit analysis yielded economic cut-off grades of:

- Oxide Type: 0.006 oz/st Au, 0.720 oz/st Ag;
- HW Type: 0.007 oz/st Au, 0.564 oz/st Ag;

→ FW Type: 0.008 oz/st Au, 0.752 oz/st Ag.

The Project's LOM production schedule is based on Measured, Indicated, and Inferred Mineral Resources that are potentially mineable by open pit. This provides a LOM plan of approximately 41.4 M tons of heap leach feed at an overall grade of 0.022 oz/st Au and 0.339 oz/st Ag. Approximately 37.4 M tons of these Mineral Resources (90%) are classified as Measured and Indicated, with the remaining 4.0 M tons (10%) classified as Inferred. The heap leach feed consists of 28% oxidized material, 45% unoxidized HW-type material, and 28% unoxidized FW-type material.

At a production rate of 3.8 M st/a, the mine life is approximately 10.8 years. During this time, approximately 898,800 ounces of gold and 14,042,000 ounces of silver are delivered to the heap leach pad. The heap leach process is estimated to recover 593,000 ounces of gold and 7,365,000 ounces of silver, for an overall average gold recovery of 66% and silver recovery of 52%.

The mine design consists of three mining areas: the Main Pit, Dyke Adit, and the East Hill Pit. Under the current analysis, the Main Pit and Dyke Adit combine into one open pit instead of two separate pits. Approximately 90% of production is from the Main Pit area, with 7% from the Dyke Adit area, and the remaining 3% from the East Hill area.

The overall strip ratio is 1.47 units of waste to each unit of mineralized material production.

Waste rock storage areas will be located toward the northeast and southwest sides of the pit areas in as close proximity as possible to the mining areas so as to minimize waste haulage distances. These areas were sized to store a combined 90Mt of waste rock.

Industry standard surface mining techniques formed the basis of mine production. This Base Case Scenario used CAT 992 – 16.5 yd^3 wheel loaders and CAT 777 100st trucks as production prime movers. Mining operating and capital cost estimates were developed based on contract mining.

25.5 RECOVERY METHODS

A conceptual scheme to process resource material within the PEA pit shell at Talapoosa was developed for the PEA. The scheme is based on results from earlier metallurgical tests performed on mineralized material. The processing scenario developed considers industry standard heap application rates, leach durations, and reagent concentrations to be used for leaching the mineralized material. An intermediate stage of leaching was included to improve silver recoveries within the deposit.

Gold and silver from the Talapoosa deposit will be recovered using industry standard heap leach cyanidation and Merrill Crowe gold precipitation techniques. Merrill Crowe processing of the pregnant solution was preferentially selected as the appropriate metallurgical extraction and recovery scenario given the grade of the material and the relatively high ratio of recovered gold to silver. Saleable doré product containing the recovered gold and silver will be shipped to an established refinery.

The mineralized material of economic value will be crushed to a P_{80} = 10 mesh, the final crushing step employing an HPGR because of its' effectiveness at increasing fracture surfaces within a crushed product.

Crushed material will be drum agglomerated and conveyed to a radial stacker by which it is placed in 20 ft. lifts on the heap. Drip emitters will be used to distribute leach solution to the top of the lift at a rate of 0.004 gpm ft^2. The heap and solution pond is designed to allow for 60 days of primary leaching and 60 days if intermediate solution leaching (preg building).

Pregnant solution from the heap is pumped to a Merrill Crowe processing facility wherein the gold and silver are recovered. The filter cake is treated to separate and safely recover any mercury that may have entered the leach circuit. Once treated, the filter cake is transferred to smelting furnaces. Fluxes are added and the material is smelted, producing doré.

The Talapoosa Project site is located in an area with a negative water balance allowing the project to be designed and built as a zero-discharge facility.

Detoxification of residual cyanide in the heap and the process solutions will commence through natural degradation. The bulk of the cyanide will degrade naturally over time with more stringent chemical detoxification processes possibly being applied if needed before facilities closure.

25.6 PROJECT INFRASTRUCTURE

There is currently no permanent infrastructure at the site. New infrastructure will need to be constructed for this Project.

The following infrastructure will be constructed:

- Water – A freshwater well is located in the south portion of the Property. A well and water line will be constructed to convey water from the well to the plant area. This water will also feed a fire water system that will be constructed for the buildings and the water stand for dust control on haul roads.
- Power – Power will be brought onto the site from an existing power line to the south of the Property. Approximately 2.5 miles of overhead power lines and associated transformers will need to be constructed.
- Communications – Telephone and communications lines will be constructed along the access road. Standard mine radio communications will be established.
- Security – A fence will be constructed around the Property to keep trespassers and livestock off of the Property. A security station will also be constructed on the main access road.
- Access Roads – Access roads will be established to the major facilities on the site. Access roads will be designed to allow for the largest piece of equipment that will be required to construct and maintain the facility.
- Administration and Support Facilities – Administration and support facilities will be constructed on site. These facilities will be in support of the mining and processing operations.
- Sewage treatment infrastructure including septic tanks and leach fields.
- Assay laboratory.
- First aid and industrial hygiene room.
- Communication and IT systems.
- Explosive storage.

25.7 RISKS AND OPPORTUNITIES

Risks requiring mitigation strategies include the following.

- The economic feasibility of the resource has not yet been demonstrated. This PEA is based on M&I and Inferred Resources; there are no assurances that this material will all be converted to reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. However, it should be noted that 81% of the known gold resources at Talapoosa are in the higher confidence M&I Resource categories.
- The Issuer's future financial success depends on the ability to raise additional capital from the sale of equity, issuance of debt, the sale of assets and/or the development of a property leading to positive cash flow. Development of a property may take years to complete and resulting income, if any, is difficult to determine. The sales value of any mineralization potentially discovered by the Issuer is largely dependent upon factors beyond the Issuer's control, such as the market value of the products produced.
- The resource exploration industry is an inherently risky business with significant capital expenditures and volatile metals markets. The marketability of any minerals discovered may be affected by numerous factors that are beyond the Issuer's control and which cannot be predicted, such as market fluctuations, mineral markets and processing equipment, and changes to government regulations, including those relating to royalties, allowable production, importing and exporting of minerals, and environmental protection.
- This industry is intensely competitive and there is no guarantee that, even if commercial quantities are discovered, a profitable market will exist for their sale. The Issuer competes with other junior exploration companies for the acquisition of mineral claims as well for the engagement of qualified contractors. Metal prices have fluctuated widely in recent years, and they are determined in international markets over which the Issuer has no influence.
- Exploration and development on the Issuer's Property are affected by government regulations relating to such matters as environmental protection, health, safety and labor, mining law reform, restrictions on production, price control, tax increases, maintenance of claims, and tenure. There is no assurance that future changes in such regulations would not result in additional expenses and capital expenditures, decreasing availability of capital, increased competition, title risks, and delays in operations.
- Management of construction/engineering and procurement schedules, costs, and cost containment.
- Operating risks related to recruitment and training of mine and process workforce.
- Although extensive historic and recent metallurgical test work has been completed on gold and silver recoveries at Talapoosa, potential remains that additional metallurgical testing will not support the conclusion of acceptable heap leach recovery levels as estimated in the PEA. To mitigate this risk, forthcoming test work will evaluate further optimization of heap leach processing as well as evaluation of alternate recovery methods that historical data suggests may be an acceptable or, possibly, even a preferred alternative for unoxidized material. The sensitivity analysis completed in the PEA accounts for potential changes in recovery levels for heap leach processing.
- The PEA contains an economic analysis that is sensitive to the gold price. To an extent, downside risk is mitigated by a mine plan that provides operators with considerable flexibility in responding to short-term price fluctuations. Talapoosa demonstrates economic resilience at a $1,000/oz gold price and a $14.50/oz silver price, generating $138 million in after-tax net cash flow, an $84 million NPV at a 5% discount rate, and a 25.4% IRR.

- As with all mining projects, financial returns are capital sensitive. The current plan for the mining of Talapoosa includes contract mining at operational and cost terms based on comparable Nevada operations. Should qualified contractors not be available, capital expenditures for the project would likely be greater than estimated in this study as alternative approaches, including owned or leased equipment, would be employed. The sensitivity analysis completed in the PEA accounts for these potential changes.
- The PEA operating costs have been developed from contractor, vendor, and expert consultant input. Should industry conditions change and influence market rates for products and services, the project economics would vary. The sensitivity analysis completed in the PEA accounts for these potential changes.
- The PEA contemplates access to water and power sources similar to those anticipated in the previously permitted operation. In follow-up studies to the PEA, advanced review of these sources will be completed; however, if anticipated sources are not practical, alternatives may increase costs to the project.
- Pit slope design through geo-mechanics characterization and stability analysis.
- Process design through variability of the mineralized materials sampled and tested.
- Possibilities that the local and regional population does not accept the mining project.
- The Project obtained permit authorizations for mining activities in 1996. These current authorizations provide a foundation for future permitting efforts for the Project. Risks to the permitting time frames exist from the need for additional data collection and analysis, acquisition of water rights, NV Energy permitting, and the uncertainties of the regulatory permit process.

These risks are common for this stage of gold projects and are similar risk factors to other gold projects of this stage and nature.

Opportunities to improve upon the results presented above that may be evaluated as Talapoosa is advanced include:

- Drilling to bring current Inferred Resources into the Measured or Indicated Resource category.
- Further drilling may also extend the Talapoosa deposit where it remains open to resource expansion, particularly on-strike to the southeast.
- Implementation of a comprehensive metallurgical column test work program, including discrete testing by zone based on the current, updated geologic model, to confirm that heap leach processing of the mineralized material is the preferred approach.
- Evaluation of a milling scenario, wherein ground material would be processed via agitation leaching, flotation, or some combination thereof. Flotation concentrates may be direct shipped or require additional treatment.

26 RECOMMENDATIONS

WSP recommends that the Project proceed with data collection and analysis in support of an advanced engineering study (pre-feasibility study or feasibility study).

The recommendations for the Project are mainly concerned with confirming the assumptions used within the PEA study. Specifically, additional detailed studies should be completed with respect to mineral processing and recovery, drilling to advance inferred resources into indicated category and updates of previous environmental studies to allow application for updated State of Nevada permits. Tables 26.1 and 26.2 summarize the estimated costs.

Table 26.1 - Phase 1 Budget

Program	Cost ($)
1A - Drill Program	924,000
1B - Metallurgical Test Work	1,000,000
Total	**1,924,000**

Table 26.2 - Phase 2 Budget

Program	Cost ($)
Advanced Engineering Study (Pre-Feasibility)	
Mineral Resource Update	50,000
Environmental Studies	160,000
Permitting Activities	830,000
Engineering and Reporting	625,000
Total	**1,665,000**

Specific recommendations are detailed in the following sections.

26.1 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 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 successful completion of Phase 1.

PEA estimates of metallurgical recoveries as stated in Section 13 - Mineral Processing and Metallurgical Testing will require a substantial amount of additional test work to confirm amenability at the extractions quoted. Such test work would be conducted on representative samples crushed by a high pressure grinding roll (HPGR) to a nominal size of 1.7 mm (10 mesh).

Samples would be subjected to column leach testing to determine leach rate, overall gold and silver extraction, cyanide consumption, and cement and lime requirements. Such columns would be conducted in such a manner to also provide information about the permeability of the material being leached and agglomeration requirements. This testing would include optimization of agglomerating conditions and load/permeability type testing on leached agglomerates. Tailings from this testing should be evaluated to assess the long term potential for acid rock drainage and constituent mobilization.

Mineralogical characterization testing should be conducted on the different mineralized materials. This is recommended to determine the exact mineral composition and correlation / interfaces between valuable and gangue minerals.

There are also alternative processing scenarios including fine grinding followed by either flotation or cyanide leaching that may substantially increase gold and silver recovery. Test work to consider these alternatives needs to be conducted so as to not overlook the economic benefit that may be realized through potentially more attractive metallurgical processing methods.

Further metallurgical test work is warranted for Talapoosa. New samples representative of newly developed 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 HW and FW 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 economically efficient method to extract the precious metals. Gravity, flotation, agitated and column leaching have all been tested previously, and different zones of the Project behave differently for each method. The various mineralized volumes will need to be tested to determine the best process(es) for optimal recovery of the gold and silver resources over the entire deposit. It may be possible to apply a few different recovery methods over the mine life. Possible process combinations 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/agitated leach of flotation / gravity tails/fine regrind of 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, agitated leach, and heap leach processing conditions.

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.

More work is required for the leaching of flotation concentrates and possibly oxidation of the flotation concentrates prior to agitated leaching. Potential buyers of sulphide concentrates need to be identified and contacted to determine if selling flotation concentrate is an economically viable option.

26.1.1 PHASE 1A – METALLURGICAL DRILLHOLE AND RESOURCE EXPANSION PROGRAM

The scope of Phase 1 of the program is dependent on the results from this PEA. The decision of where to drill metallurgical sample holes will depend on the review of the results from this PEA and better understanding of what core is still available for testing.

The principal objectives of the Phase 1A program will be to:

- Extract core samples of sufficient quantity across the mineralized zones 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.
- Additional drilling to convert Inferred Resources to Indicated category at East Hill and Dyke Adit.
- Advance the metallurgical understanding of the deposit by completion of crush, column leach, and mill test analysis.

The program is estimated to cost $924,000. Table 26.3 summarizes the Phase 1A program proposed.

Table 26.3 - Proposed Phase 1A Diamond Drill Program

	Unit Rate ($)	No. of Units	Unit	Cost ($)
Diamond Drilling	175	5,000	ft.	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

26.1.2 PHASE 1B – METALLURGICAL TEST PROGRAM

The metallurgical test program will utilize samples gathered in Phase 1A. The program will test the metallurgical process combinations discussed in Section 26.1 and other work evolving from the results of this PEA. 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 $1,000,000. Table 26.4 summarizes the Phase 1B program proposed.

Table 26.4 - Proposed Phase 1B Exploration Program

	Unit Rate ($)	No. of Units	Unit	Cost ($)
Metallurgical Test Work	1,000,000	1	unit	1,000,000

26.1.3 PHASE 2 – PRE-FEASIBILITY STUDY

The scope of Phase 2 of the program is dependent on the results from Phase 1.

The principal objectives of the program will be to:

- Complete a mineral resource update to include drilling results of Phase 1 and convert Dyke Adit and East Hill zones from Inferred to Indicated Resource.
- Complete updated waste rock geochemical characterization and pit lake hydrogeochemical modeling to meet 2015 standards.
- Initiate update of time-critical State of Nevada Permit Applications.
- Complete Pre-feasibility engineering and report.

The program is estimated to cost $1,665,000. Table 26.5 summarizes the Phase 2 program proposed.

Table 26.5 - Proposed Phase 2 Pre-Feasibility Program

Program	Cost ($)
Advanced Engineering Study (Pre-Feasibility)	
Mineral Resource Update	50,000
Environmental Studies	160,000
Permitting Activities	830,000
Engineering and Reporting	625,000
Total	**1,665,000**

26.2 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 for samples 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.

26.2.1 MINING

- Obtain multiple budget quotes from mining contractors to perform the proposed mining operations and site-wide construction of infrastructure.
- The Project has accumulated an extensive amount of data through past years of exploration and investigation. Several historical reports with respect to pit slope geotechnical investigation and pit dewatering investigation were available.
- Recommendations for further development of the Project are primarily concerned with confirming the existing data and the assumptions used for the PEA, specifically in the areas of geotechnical, hydrology, and hydrogeological studies. These data should be updated and reviewed with respect to the new pit limits.
- Complete further optimization of the final pit designs and confirm pit slope geotechnical parameters based on the new pit limits. The overall pit slope angle of the southwest side of pit may be substantially increased.
- Complete further optimization of the pit sequencing.

26.2.2 RECOVERY METHODS

There are several scenarios that may be applicable to the processing of mineralized material at Talapoosa. Consideration of alternate processing scenarios and the costs associated therewith will be key to determining the best processing method to use.

Given the potential to substantially increase metallurgical extractions and recoveries, several alternative scenarios should be considered in addition to refining the costs associated with heap leaching of the mineralized materials (as considered in this PEA). Process methods and costs for grinding, flotation, agitated leaching, extraction of values from a flotation concentrate and alternatives to sell flotation concentrates should be considered in conjunction with test work verified recoveries to determine the best process alternative or set of alternatives.

- Results from the Phase 1A and 1B metallurgical sample collection and test work program will be used to compare alternative processes.
- Grind, grade, recovery test work will be performed to estimate optimal grind and recovery to be achieved through the alternative methods
- Cost estimates will be prepared for each alternative to a level of detail sufficient to accept or reject that process alternative as a viable contender for being the select process by which to treat the material(s).
- Abbreviated economic modeling of the various process scenarios will allow comparisons and filtering of the alternatives such that sound decisions regarding the preferred process can be made.

It may be possible that several processes are best used during the mine life to treat different portions of the deposit.

Ultimately a definitive plan will be developed wherein the optimal economic processes are fully developed to a Pre-Feasibility Study level of engineering and cost detail.

26.2.3 PROJECT INFRASTRUCTURE

- Following additional optimization of pit, update volumes for waste and heap material. With updated material volumes, update mass and water balance models.

- Complete further optimization of heap leach pad layout and sequencing.
- Complete further optimization of process and storm water ponds.
- Optimize crusher and plant facility layout.

26.2.4 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

With current authorizations in place for mining activities, the recommendation for any new required permit is to obtain them in accordance with the construction, operation, and reclamation activities described in the currently approved Plan and NRP to maximize use of existing data and NEPA permitting time frames.

To operate the Project as outlined in the approved Plan and NRP, a WPCP, air quality operating permit, and a mercury operating permit to construct will be required. Completion of the geochemical test work should commence as early as possible due to the time required to collect samples, perform the regulatory testing protocols, and interpret the data in coordination with the regulatory agencies prior to submittal of a permit application. Engineering should commence at the earliest opportunity due to the need for the inclusion of final designs in the permit applications.

Investigations of available water rights should be completed in order to secure adequate water rights from the Nevada Division of Water Resources for the Project.

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28 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 Preliminary Economic Assessment on the Talapoosa Project, Nevada, dated May 20, 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 1.3, 1.5, 1.12, 2 to 12, 14, 23, 24, 25.2, 25.7, 26.1, 27.1 and 28 of the Technical Report.
- I am independent of Timberline Resources Inc. 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 7th day of August, 2015 at Sudbury, Ontario.

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

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

JOANNE ROBINSON, P.ENG.

I, Joanne Robinson, P.Eng., of Toronto, Ontario do hereby certify::

- I am a Mining Engineer with WSP Canada Inc. with a business address at 1300 Yonge St., Suite 801, Toronto, Ontario, M4T 1X3.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Talapoosa Project, Nevada, dated May 20, 2015 (the "Technical Report").
- I am a graduate of Queen's University with a Bachelor of Science in Mining Engineering. I am a member in good standing of the Association of Professional Engineers of Ontario (PEO), License Number 100049603. I have been working as a mining engineer from 1997 to 2000 and 2004 to present. My relevant experience includes 7 years working at various Canadian open pit operations in progressively senior roles doing production engineering, mine design, and mine planning, over 3 years with an open pit mine development project focusing on the pit optimization, mine design, mine planning, cost estimation, and project management, and over 2 years in mine consulting completing the open pit mine design, optimization, planning, and mine cost estimation aspects for a number of technical studies. 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 1.6, 1.7, 1.11, 15, 16, 19, 21, 22, 25.1, 25.4, and 27.3 of the Technical Report.
- I am independent of Timberline Resources Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- 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 7th day of August, 2015 at Toronto, Ontario.

"Original document signed and stamped by Joanne Robinson, P.Eng."

Joanne Robinson, P.Eng.
Principal Mining Engineer
WSP Canada Inc.

RICHARD JOLK, P.E.

I, Richard Jolk, P.E. of Golden, Colorado do hereby certify that:

- I am a Minerals Engineer with Mineral Property Development, Inc. (MPDI), with a business address at 3296 Alkire Way, Golden, CO 80401.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Talapoosa Project, Nevada, dated May 20, 2015 (the "Technical Report").
- I am a graduate of the Colorado School of Mines. I am a registered professional engineering in the State of Colorado, #24448. I am a certified minerals appraiser with the International Institute of Mineral Appraisers. I am a member in good standing with the Society of Mining Engineers.
- I have visited the Property twice during daylight hours between January and March 2015.
- I am responsible for Sections 1.8, 18.5 to 18.7, 25.5, and portions of 26.2 of the Technical Report.
- I am independent of Timberline Resources Inc. as defined by Section 1.5 of the Instrument.
- I have had no prior involvement with Timberline Resources or the Talapoosa Project.
- 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 7th day of August, 2015 at Sparks, Nevada.

"Original document signed and stamped by Richard Jolk"

Richard Jolk, P.E.
Minerals Engineer
Mineral Property Development, Inc.

JACK MCPARTLAND, M.S. METALLURGICAL ENGINEERING

I, Jack McPartland, M.S of Sparks, Nevada, do hereby certify that

- 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 Preliminary Economic Assessment on the Talapoosa Project, Nevada, dated May 20, 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 1.4 and 13, 25.3, portion of 26.2, and 27.2 of the Technical Report.
- I am independent of Timberline Resources Inc. 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 7th day of August, 2015 at Sparks, Nevada.

"Original document signed and stamped by Jack McPartland"

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

MICHAEL HENDERSON, P.E.

I, Michael Henderson of Evergreen, Colorado do hereby certify that:

- I am an Engineer with DOWL LLC., with a business address of 32045 Castle Court, Suite 101, Evergreen, Colorado 80439
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Talapoosa Project, Nevada, dated May 20, 2015 (the "Technical Report").
- I am a graduate of Colorado State University (BSCE) and the University of Pittsburgh (MSCE). I am registered in multiple States in the US and in the Provinces of Saskatchewan and British Columbia in Canada. All of these licenses are in good standing. My License Number in Nevada is #7611. My relevant experience includes 35 years of directly related experience in mining operations, including design of heap leach facilities. 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 1.9, 18.1 to 18.4, 25.6, and portion of 26.2 of the Technical Report.
- I am independent of Timberline Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that includes supervision of the work completed by WESTEC from 1995 to 1997.
- 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 7th day of August, 2015 at Evergreen, Colorado.

"Original document signed and stamped by Michael Henderson"

Michael Henderson,
Practice Area Leader - Mining
DOWL, LLC.

RICHARD DELONG, P.G.

I, Richard DeLong, P.G. of Reno, Nevada do hereby certify that:

- I am the President of, and Principal Scientists at Enviroscientists, Inc.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Talapoosa Project, Nevada, dated May 20, 2015 (the "Technical Report").
- I am a graduate of the University of Idaho where I received two masters' degrees; Geology and Resource Management. I am a member in good standing of the Mining & Metallurgical Society of American and a Qualified Professional Member with special expertise in Environmental Permitting and Compliance, Member Number 01471QP. My relevant experience includes 28 years of experience in acquisition of environmental and other permits for exploration and mining operations, with a majority of that experience with project in Nevada. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not visit the Property as part of the preparation of the Technical Report; however, I have visited the Property in the past five years.
- I am responsible for Sections 1.10, 20, and portion of 26.2.4 of the Technical Report.
- I am independent of Timberline Resources Inc. as defined by Section 1.5 of the Instrument.
- I have prior involvement with the Property that includes supervision of some of the permit acquisition and permit compliance activities for the Property.
- 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 7th day of August, 2015 at Reno, Nevada.

"Original document signed and stamped by Richard DeLong"

Richard DeLong, P.G.
President
Enviroscientists, Inc.