



# TECHNICAL REPORT ON THE TRINITY PROJECT

PERSHING COUNTY, NEVADA, USA



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# TABLE OF CONTENTS

<b>1.0 SUMMARY .....</b>	<b>1</b>
1.1 Location and Ownership .....	1
1.2 History, Exploration, and Past Production .....	1
1.3 Geology and Mineralization .....	2
1.4 Metallurgical Testing and Mineral Processing .....	3
1.5 Conclusions and Recommendations .....	4
<b>2.0 INTRODUCTION .....</b>	<b>5</b>
2.1 Project Scope and Terms of Reference .....	5
2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure .....	6
<b>3.0 RELIANCE ON OTHER EXPERTS .....</b>	<b>7</b>
<b>4.0 PROPERTY DESCRIPTION AND LOCATION .....</b>	<b>8</b>
4.1 Location .....	8
4.2 Land Area .....	8
4.3 Agreements and Encumbrances .....	12
4.3.1 Exploration Agreement with Option to Purchase between K9 Gold Corp. and Newmont .....	12
4.3.2 Royalties .....	13
4.3.2.1 Royalties Applicable to Sections 9 and 15 .....	13
4.3.2.2 Royalties Applicable to the Remainder of the Trinity Property .....	14
4.4 Environmental Permitting and Liabilities .....	14
4.4.1 Surface and Groundwater .....	14
4.4.1.1 Special Status Species .....	15
4.4.1.2 Access Route .....	15
4.4.1.3 Permitting Risk .....	15
<b>5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY .....</b>	<b>16</b>
5.1 Access .....	16
5.2 Climate .....	16
5.3 Local Resources and Infrastructure .....	16
5.4 Physiography .....	16
<b>6.0 HISTORY .....</b>	<b>18</b>
6.1 Exploration and Mining History .....	18
6.1.1 U.S. Borax and Chemical Corporation .....	18
6.1.2 Santa Fe Pacific Mining, Inc. .....	20
6.1.3 Renaissance Gold Inc. .....	21
6.1.4 Liberty Silver Corp. .....	21
6.2 Historic Mineral Resource and Reserve Estimates .....	22
6.3 Historic Mine Production .....	27
<b>7.0 GEOLOGIC SETTING AND MINERALIZATION .....</b>	<b>29</b>

7.1	Geologic Setting .....	29
7.1.1	Regional Geology.....	29
7.1.2	Local Geology.....	31
7.1.3	Project Geology.....	33
7.2	Mineralization .....	36
<b>8.0</b>	<b>DEPOSIT TYPES .....</b>	<b>39</b>
<b>9.0</b>	<b>EXPLORATION.....</b>	<b>40</b>
<b>10.0</b>	<b>DRILLING.....</b>	<b>41</b>
10.1	U.S. Borax.....	42
10.2	Santa Fe Pacific Mining Inc.....	43
10.3	Renaissance Gold Inc. ....	44
10.4	Liberty Silver Corp. ....	45
10.5	Summary of Drilling and Sampling .....	45
<b>11.0</b>	<b>SAMPLE PREPARATION, ANALYSIS, AND SECURITY .....</b>	<b>46</b>
11.1	U.S. Borax.....	46
11.2	Santa Fe Pacific Mining Inc.....	47
11.3	Renaissance Gold Inc. ....	48
11.4	Summary Statement .....	49
<b>12.0</b>	<b>DATA VERIFICATION .....</b>	<b>50</b>
12.1	Site Visit.....	50
12.2	Historical Data Verification Studies .....	50
12.3	RESPEC Data Verification .....	51
12.3.1	CMS vs. USBRC Checks on Silver - 1983 .....	51
12.3.2	Bondar-Clegg vs. USBRC Checks on Silver - 1984.....	53
12.3.3	USBRC vs. USBRC Checks on Silver – 1984-1985 .....	54
12.3.4	Hunter vs. USBRC Pulp Checks on Silver – 1985 .....	56
12.3.5	AAL vs. AAL Pulp Checks on Silver – 2006-2007.....	58
12.3.6	CMS vs. USBRC Analyses for Lead – 1983.....	60
12.3.7	USBRC vs. USBRC Checks on Lead – 1984/85.....	61
12.3.8	Hunter vs. USBRC Analyses for Lead - 1985.....	62
12.3.9	AAL vs. AAL Pulp Checks on Lead from RC Samples 2006/07 .....	64
12.3.10	USBRC vs. USBRC Checks on Zinc - 1985.....	65
12.3.11	Hunter vs. USBRC Pulp Checks on Zinc - 1985.....	66
12.3.12	CMS vs. USBRC Checks on Zinc - 1985.....	68
12.3.13	AAL vs. AAL Pulp Checks on Zinc from RC Samples.....	69
12.4	Summary Statement on Data Verification.....	71
12.5	Assay Database Audit.....	71
<b>13.0</b>	<b>MINERAL PROCESSING AND METALLURGICAL TESTING.....</b>	<b>72</b>
13.1	Historical Testing and Mineral Processing .....	72



13.1.1	1983 U. S. Borax Research Corp. and Hazen Research Inc. ....	72
13.1.2	1983-1984 U. S. Borax Research Corp. ....	73
13.1.3	1984-1985 U. S. Borax Research Corp. ....	74
13.1.4	1986-1987 Kappes, Cassidy & Associates.....	75
<b>14.0</b>	<b>MINERAL RESOURCE ESTIMATES .....</b>	<b>76</b>
<b>15.0</b>	<b>MINERAL RESERVE ESTIMATES .....</b>	<b>77</b>
<b>16.0</b>	<b>MINING METHODS.....</b>	<b>78</b>
<b>17.0</b>	<b>RECOVERY METHODS .....</b>	<b>79</b>
<b>18.0</b>	<b>PROJECT INFRASTRUCTURE .....</b>	<b>80</b>
<b>19.0</b>	<b>MARKET STUDIES AND CONTRACTS .....</b>	<b>81</b>
<b>20.0</b>	<b>ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT .....</b>	<b>82</b>
<b>21.0</b>	<b>CAPITAL AND OPERATING COSTS.....</b>	<b>83</b>
<b>22.0</b>	<b>ECONOMIC ANALYSIS .....</b>	<b>84</b>
<b>23.0</b>	<b>ADJACENT PROPERTIES .....</b>	<b>85</b>
<b>24.0</b>	<b>OTHER RELEVANT DATA AND INFORMATION .....</b>	<b>86</b>
<b>25.0</b>	<b>INTERPRETATION AND CONCLUSIONS .....</b>	<b>87</b>
<b>26.0</b>	<b>RECOMMENDATIONS.....</b>	<b>88</b>
<b>27.0</b>	<b>REFERENCES.....</b>	<b>89</b>
<b>28.0</b>	<b>DATE AND SIGNATURE PAGE.....</b>	<b>93</b>
<b>29.0</b>	<b>CERTIFICATE OF QUALIFIED PERSONS.....</b>	<b>94</b>
	<b>APPENDIX A LIST OF SIGNIFICANT INTERSECTIONS.....</b>	<b>A-1</b>



# LIST OF TABLES

TABLE	PAGE
Table 1-1. Recommendations for Trinity .....	4
Table 4-1. Unpatented Lode Mining Claims of the Trinity Property .....	10
Table 6-1. 1986 Trinity "Geologic Reserve Calculation" Prior to Mining .....	23
Table 6-2. 1986 Southwest Extension Oxide "Reserve" .....	24
Table 6-3. July 1987 Initial and January 1988 Revised "Reserve" Estimate for the Oxide Ore Body .....	24
Table 6-4. 1988 Trinity Total Undeveloped "Reserves" .....	25
Table 6-5. 1989 Borax Summary of Remaining "Mineral Reserves" at Trinity .....	25
Table 6-6. 1989 Mineral "Reserve" Stockpile at the Trinity Mine .....	26
Table 6-7. Trinity Inferred Mineral Oxide Resources .....	26
Table 6-8. Trinity Inferred Mineral Sulfide Resources .....	27
Table 10-1. Trinity Database Summary .....	41
Table 12-1. CMS Analyses vs. USBRC Analyses for Silver – 1983 .....	53
Table 12-2. Bondar-Clegg Analyses vs. USBRC Analyses for Silver – 1984 .....	54
Table 12-3. USBRC Checks vs. USBRC Original Analyses for Silver – 1984/85 .....	56
Table 12-4. Hunter Checks vs. USBRC Original Analyses for Silver – 1985 .....	58
Table 12-5. AAL Checks on Pulps from RC and Core Samples vs. AAL Analyses for Silver – 2007 .....	59
Table 12-6. CMS Checks on Rotary Percussion Samples Relative to Original USBRC Analyses for Lead – 1983 .....	61
Table 12-7. USBRC Checks from Rotary Percussion Samples Relative to Original USBRC Analyses for Lead – 1984/85 .....	62
Table 12-8. Hunter Checks on Pulps Relative to Original USBRC Analyses for Lead – 1985 .....	63
Table 12-9. AAL Checks on Pulps from RC and Core Samples Relative to Original AAL Analyses for Lead – 2006/07 .....	65
Table 12-10. USBRC Checks from Rotary Percussion Samples Relative to Original USBRC Analyses for Zinc – 1985 .....	66
Table 12-11. Hunter Checks on Pulps Relative to Original USBRC Analyses for Zinc – 1985 .....	67
Table 12-12. CMS Checks Relative to Original USBRC Analyses for Zinc – 1985 .....	69
Table 12-13. AAL Checks on Pulps from RC and Core Samples Relative to Original AAL Analyses for Zinc – 2006 .....	70
Table 26-1. Estimated Costs of the Trinity Program .....	88

# LIST OF FIGURES

FIGURE	PAGE
Figure 4-1. Location of the Trinity Property, Pershing County, Nevada .....	9
Figure 4-2. Property Map for the Trinity Project .....	13
Figure 5-1. Physiography of the Trinity Mine Area .....	17
Figure 6-1. View of the Historic Liberty Silver Open-Pit .....	28
Figure 7-1. Regional Geology of the Trinity Range Area, Pershing County .....	30
Figure 7-2. Generalized Geology of the Trinity Area .....	32
Figure 7-3. Geology of the Trinity Mine Area .....	35
Figure 7-4. Sulfide Mineralization at Trinity .....	37
Figure 10-1. Drillhole Location Map for the Trinity Project.....	42
Figure 12-1. CMS Checks Relative to Original USBRC Analyses for Silver – 1983.....	51
Figure 12-2. Bondar-Clegg Checks Relative to Original USBRC Analyses for Silver – 1984 .....	54
Figure 12-3. USBRC Checks Relative to Original USBRC Analyses for Silver – 1984/85.....	55
Figure 12-4. Hunter Checks Relative to Original USBRC Analyses for Silver – 1985.....	57
Figure 12-5. AAL Checks on Pulps from RC and Core Samples Relative to Original AAL Analyses for Silver – 2007.....	59
Figure 12-6. CMS Checks Relative to Original USBRC Analyses for Lead – 1983 .....	60
Figure 12-7. USBRC Checks Relative to Original USBRC Analyses for Lead – 1984/85.....	61
Figure 12-8. Hunter Checks on Pulps Relative to Original USBRC Analyses for Lead – 1985.....	63
Figure 12-9. AAL Checks on Pulps Relative to Original AAL Analyses for Lead – 2006/07.....	64
Figure 12-10. USBRC Checks from Rotary Percussion Samples Relative to Original USBRC Analyses for Zinc – 1985.....	66
Figure 12-11. Hunter Checks on Pulps Relative to Original USBRC Analyses for Zinc – 1985 .....	67
Figure 12-12. CMS Checks Relative to Original USBRC Analyses for Zinc – 1985.....	68
Figure 12-13. AAL Checks on Pulps Relative to Original AAL Analyses for Zinc – 2007 .....	70

## 1.0 SUMMARY

RESPEC Company, LLC ("RESPEC") has been engaged to prepare this technical report on the Trinity Project, Pershing County, Nevada, for K9 Gold Corp. ("K9"), a Canadian company based in West Vancouver, Canada that is listed on the TSX Venture Exchange (TSX-V KNC). This report has been prepared in accordance with the disclosure and reporting requirements outlined in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014.

RESPEC (formerly Mine Development Associates) previously prepared a technical report, which was filed by Liberty Silver in April 2011, in support of the first NI 43-101-compliant mineral resource estimate of the Trinity silver, lead, and zinc deposit. The current updated report has been prepared in support of K9 Gold Corp.'s acquisition of the property. This report updates the project's land status and provides new recommendations for the Trinity Project.

### 1.1 LOCATION AND OWNERSHIP

The Trinity Project is located along the west flank of the Trinity Range in Pershing County, Nevada, about 23 miles by road northwest of Lovelock, the county seat.

The Trinity property consists of approximately 15,783.4 acres, 13,243.68 acres of which are mineral and mining leases and private land through an Exploration Agreement with Newmont USA Limited, a Delaware Corporation ("Newmont"). The remaining acreage consists of 70 unpatented lode mining claims controlled by K9. The Exploration Agreement with Newmont was made effective September 16, 2025. K9 has acquired all of Newmont's rights to explore for minerals in the Trinity property, and through the agreement, the option to purchase Newmont's right, title, interest, and obligations to the property, subject to certain term conditions.

### 1.2 HISTORY, EXPLORATION, AND PAST PRODUCTION

The Trinity Project lies in the Trinity mining district, which had limited production of silver, lead, zinc, and gold from 1864 through 1942, primarily from the east side of the Trinity Range. In the vicinity of the Trinity Project, which is located on the west side of the range, there was historic prospecting with unrecorded but presumed minor silver production.

Minor exploration activity took place in the vicinity of the Trinity Project in the 1950s, and in the 1960s, Phelps Dodge Corporation completed trenching, IP surveying, and limited drilling in the area.

U. S. Borax and Chemical Corp. ("Borax") became interested in what is now the Trinity Project in 1982 based on reconnaissance geochemical sampling that indicated the presence of anomalous lead and silver in the Willow Canyon area. By 1984, Borax had acquired a property position and had entered into a joint-venture with Southern Pacific Land Company (later Santa Fe Pacific Mining, Inc. ("SFPM") and still later Newmont Mining Corp. ("Newmont")), in which Borax was the operator. From 1982 to 1986, Borax

and its joint-venture partner explored the property and developed the Trinity mine. Borax operated the open-pit heap leaching mine, through a mining contractor, on behalf of the joint-venture from September 3, 1987, to August 29, 1988, with leaching continuing into 1989. During this period, the mine produced about five million ounces of silver from about 1.1 million tons of oxidized ore grading six ounces of silver per ton. Borax drilled and conducted extensive metallurgical testing on the sulfide mineralization, but metal prices at the time were too low to support mining of this material.

In 1984-1985, 1987-1989, and 1990, SFPM conducted exploration and drilling on its property in the vicinity of the joint-venture lands. In 1991, SFPM acquired sole interest in the joint-venture lands, including Borax's claims, and conducted further exploration through 1992. SFPM's 1990-1992 exploration work concentrated on down-dip and lateral extensions of mineralization underlying the oxide pit and the sulfide mineralization, as well as extensions of mineralization outside the immediate mine area.

There was no exploration on the Trinity property from 1993 to 2005. In August 2005, Renaissance leased the property from Newmont, which had acquired SFPM's Nevada holdings. Under an earn-in agreement with Piedmont Mining Company, Renaissance explored the property from September 2005 through July 2009, including limited drilling in 2006 and 2007 that encountered high-grade silver values below and adjacent to the open-pit.

Liberty Silver entered into an earn-in agreement with Renaissance in March 2010. The company conducted extensive data compilation, geophysical surveys, and drilled 20 exploration holes. Liberty Silver terminated their agreement in 2017.

K9 acquired the Trinity Project through an agreement with Newmont in September 2025.

### 1.3 GEOLOGY AND MINERALIZATION

The Trinity Project lies on the western flank of the Trinity Range, one of the generally north-trending ranges formed during Tertiary extension of the Basin and Range Province.

Within the Trinity Range, the basement rocks are comprised of the Middle Triassic to Early Jurassic near-shore deltaic deposits of the Auld Lang Syne Group, which are represented by phyllite, argillite, quartzite, and dirty limestone at the Trinity Project. The best represented pre-Cenozoic deformation in this portion of the Trinity Range is the Jurassic and Cretaceous Nevadan Orogeny, which resulted in low-grade regional metamorphism, variably directed folding, and thrust faulting. A Cretaceous intrusive episode culminated the Nevadan Orogeny and is exemplified by a Cretaceous granodiorite stock just northeast of the Trinity Project.

Tertiary volcanic and sedimentary rocks and Quaternary sediments are abundant in the Trinity Project area. There is a thin Tertiary rhyolite sequence along the central north-south axis of the property that includes the resource area. These volcanic rocks overlie the Mesozoic phyllite and argillite, exposed to the east, but are separated by an argillite breccia that is closely associated with faulting. The rhyolite includes interbedded rhyolitic flows, welded tuffs, air-fall tuffs, epiclastic tuffs, and lacustrine deposits. Several rhyolite domes, dikes, and sills have also been identified on the property, some of which may be

related to mineralization. Early Tertiary north- to northwest-trending faults are present in the Trinity Project area, as are younger north- to northeast-trending normal faults. Late Tertiary and/or Quaternary bench and channel gravel deposits and Quaternary alluvium and outwash unconformably overlie the rhyolites and cover the western part of the property.

Rhyolite porphyry, aphanitic rhyolite, and volcanoclastic rocks are the principal host rocks for mineralization in the Trinity mine area. Silicification and quartz-adularia-sericite alteration are associated with the mineralization. Tertiary rhyolitic tuffs and flows were extensively altered and form a halo extending 1.6 miles beyond the main mineralized area. This alteration affected the Auld Lang Syne Group only locally along faults and breccia zones.

Mineralization at the Trinity Project is controlled by a northeast-trending zone of normal faults. Silver, lead, and zinc mineralization occur in fractures and bedding planes in Tertiary rhyolite in the hanging-wall block of the fault zone. Although mineralization continues downward into the underlying Triassic rocks, it is more tightly constrained to fractures that host higher-grade vein mineralization. The original Trinity silver deposit can generally be divided into two parts: a sulfide zone below the current pit and to the northeast, and an overlying oxide zone. Borax's mining in the late 1980s focused on a portion of the oxide zone.

Mineralization occurs as oxidized and unoxidized sulfides in veinlets, as fracture-controlled mineralization, and as disseminations within the host rocks, including breccia matrix. Sulfide mineralization consists mainly of pyrite, sphalerite, galena, marcasite, and minor arsenopyrite with various silver minerals, including tetrahedrite-freibergite, pyrargyrite, minor argentite, and rare native silver, with traces of gold, pyrhotite, stannite, and chalcopyrite. Low-grade lead and zinc have the potential to add value as byproducts.

## 1.4 METALLURGICAL TESTING AND MINERAL PROCESSING

The Trinity oxide and sulfide mineralization underwent extensive metallurgical analyses during the early pre-production time period (1983–1987). The metallurgical studies were conducted by U.S. Borax Research Corporation ("USBRC"), Hazen Research Incorporated, and Kappes, Cassidy & Associates and included mineralogy studies, head material analyses, grinding studies, gravity separation, flotation, and leach test work. This work suggests that both oxidized and unoxidized types of silver mineralization are amenable to cyanidation, but the oxide performs better than the sulfide. The sulfide mineralization yielded significant extraction in the flotation tests, whereas the oxide did not.

## 1.5 CONCLUSIONS AND RECOMMENDATIONS

The author considers Trinity to be a property of merit that is worthy of further investment based on the existence of widespread mineralization at the property. In addition, the large area of geochemically anomalous lead and zinc encountered in the drilling is suggestive of an intrusive-related system that has not been systematically explored. The author recommends a two-phased approach to further exploration at the property, consisting of data compilation, evaluation, exploration targeting, and drilling.

Total cost for this work is estimated at \$450,000, which is provided in Table 1-1.

Table 1-1. Recommendations for Trinity

Item	Estimated Cost
Phase 1: Data Compilation and Evaluation	\$ 150,000
Phase 2: RC Drilling and Assaying (~4,000 feet)	300,000
<i>Total</i>	<i>US\$450,000</i>

## 2.0 INTRODUCTION

RESPEC Company, LLC (“RESPEC”) has been engaged to prepare this updated technical report on the Trinity Project, Pershing County, Nevada, for K9 Gold Corp. (“K9”). K9 is a Canadian company based in West Vancouver, British Columbia, that is listed on the TSX Venture Exchange (TSX-V KNC; OTC: WDFCF). K9 acquired its interest in the Trinity property through an exploration with an option to purchase agreement with Newmont USA Limited, a Delaware Corporation (“Newmont”).

This technical report has been prepared in compliance with the disclosure and reporting requirements outlined in the Canadian Securities Administrators’ NI 43-101, Companion Policy 43-101CP, and Form 43-101F1, as amended.

### 2.1 PROJECT SCOPE AND TERMS OF REFERENCE

Mr. Jeffrey Bickel of RESPEC Company LLC. prepared this technical report. The report draws extensively from Hartley et al (2011), previously prepared by Mine Development Associates (MDA), an engineering firm which was acquired by RESPEC. The current updated report has been prepared in support of K9’s acquisition of the property, updates the project’s land status to reflect the acquisition of additional claims, and provides new recommendations for the Trinity Project.

Mr. Jeffery Bickel, C.P.G. (AIPG), is a Registered Member of the Society of Mining, Metallurgy, and Exploration, a Registered Geologist (Arizona), and Senior Geologist with RESPEC Company LLC. Mr. Bickel is a “qualified person” as defined by NI 43-101 and is the sole author of all sections. There is no affiliation between Mr. Bickel and K9 except that of an independent consultant/client relationship. Unless otherwise noted, references to “the author” in this report are to Mr. Bickel.

Mr. Bickel has relied on the data and information provided by K9 for the completion of this report. In addition, he has relied upon the individuals described in Section 3.0 and the references cited in Section 27.0.

Mr. Bickel visited the Trinity property on October 16, 2025, as well as the core storage facility in Sparks, Nevada, on October 17, 2025. He has reviewed much of the available data and made judgments about its reliability. Where deemed either inadequate or unreliable, he has disclosed the lack of confidence in the information.

The effective date of this report is October 16, 2025.

## 2.2 FREQUENTLY USED ACRONYMS, ABBREVIATIONS, DEFINITIONS, AND UNITS OF MEASURE

Unless otherwise indicated, all references to dollars (\$) in this report refer to the currency of the United States. Frequently used acronyms and abbreviations are listed below.

AA	atomic absorption spectrometry
Ag	silver
As	arsenic
Au	gold
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
Cu	copper
FA-AA	fire assay with an atomic absorption finish
ft	feet
ha	hectare
Hg	mercury
ICP	inductively coupled plasma
in.	inch
oz/ton	troy ounces per short ton
Pb	lead
QA/QC	quality assurance and quality control
RC	reverse circulation drilling method
RQD	rock-quality designation
ton(s)	short ton(s) (one short ton = 2,000 pounds)
Zn	zinc

### Analytical Values

	<u>percent</u>	<u>grams per metric tonne</u>	<u>troy ounce per short ton</u>
1%	1%	10,000	291.667
1 gram/tonne	0.0001%	1	0.0291667
1 oz troy/short ton	0.003429%	34.2857	1
100 ppm			2.917

**Currency** Unless otherwise indicated, all references to dollars (\$) in this report refer to the currency of the United States.

### 3.0 RELIANCE ON OTHER EXPERTS

Mr. Bickel is not an expert in legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, and property agreements in the United States. He did not conduct any investigations into the environmental or socioeconomic issues associated with the Trinity Project and is not an expert with respect to these issues.

Mr. Bickel has fully relied on Jeff Poloni of K9 to provide information concerning the legal status of K9 Gold Corporation, as well as current land and legal title, material terms of all agreements, existence of applicable royalty obligations, and material environmental and permitting information that pertains to the Trinity property. These details were provided to Mr. Bickel via email on October 16, 2025.

## 4.0 PROPERTY DESCRIPTION AND LOCATION

Mr. Bickel is not an expert in land, legal, environmental, and permitting matters. The information in this section was prepared by Mr. Bickel with information received from Mr. Jeff Poloni, CEO of K9, from documents via electronic mail in October of 2025. Beyond what is described in this report, Mr. Bickel does not know of any significant factors or risks that may affect K9's access, title, or the right or ability to perform work on the Trinity property.

### 4.1 LOCATION

The Trinity Project is in Pershing County, Nevada, approximately 23 miles by road northwest of Lovelock, the county seat (Figure 4-1). The property lies along the west flank of the Trinity Range of northern Nevada. The Rochester silver mine, one of the larger silver mines in the U.S., lies about 25 miles southeast of the project, and the Seven Troughs gold district lies about 10 miles northwest of the project.

The center of the Trinity Project is approximately located at 40.3965°N, 118.6101°W. The property is covered by the U.S. Geological Survey Natchez Spring 7.5' topographic map.

### 4.2 LAND AREA

The Trinity property comprises approximately 15,783.4 acres as follows:

- / Land leased by K9 from Newmont Mining Corp. ("Newmont") in September 2025 that includes 13,243.68 acres of mineral and mining leases controlled by Newmont. The leases are valid subject to the conditions outlined in section 4.3.1.
- / 70 unpatented mining claims (KS- Claims) controlled by K9. The mining claims remain in good standing as long as annual BLM maintenance fees are paid by September 1 annually.

K9 acquired its interest in the Trinity property through an Exploration Agreement with Newmont effective September 16, 2025. K9 has acquired all of Newmont's rights to explore for minerals in the Trinity property, and through the agreement, has the option to purchase Newmont's right, title, interest, and obligations to the property subject to certain conditions (see Section 4.3.1).

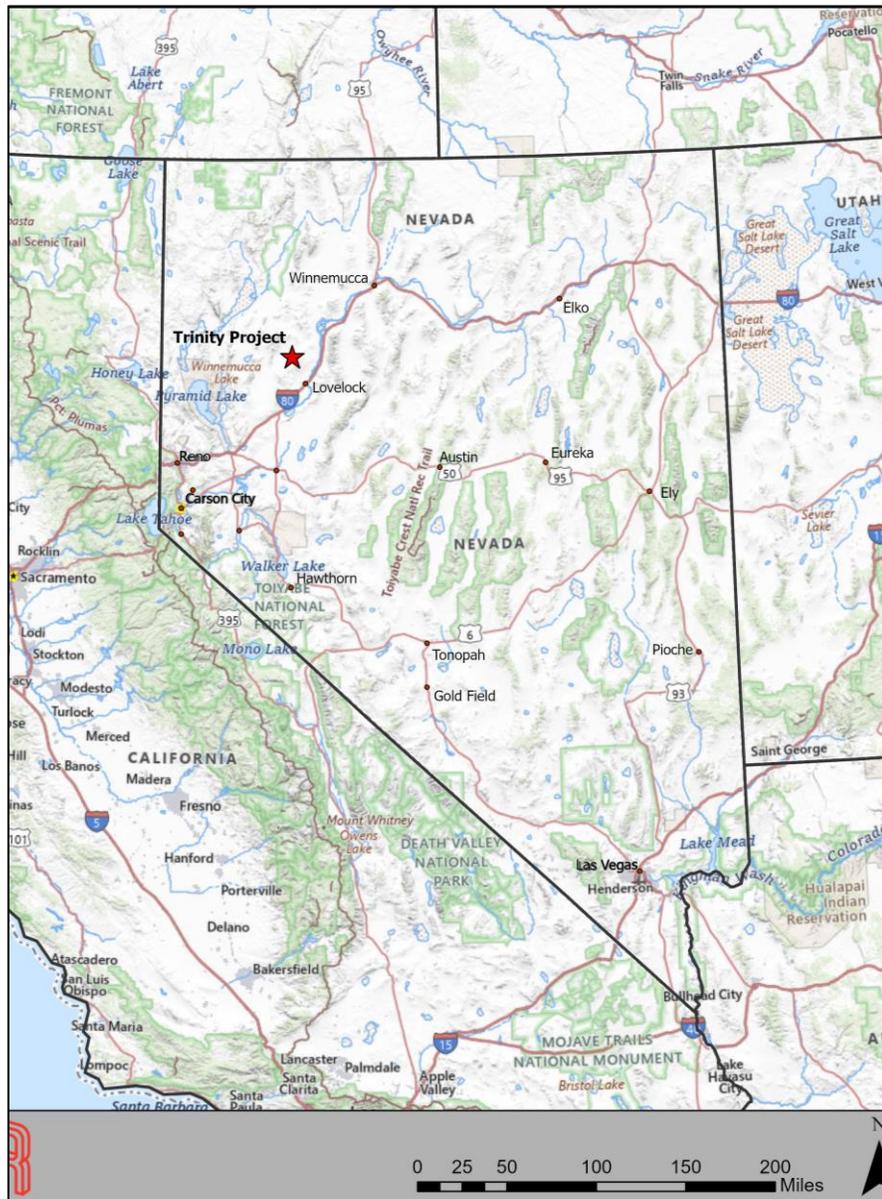


Figure 4-1. Location of the Trinity Property, Pershing County, Nevada

The agreement between K9 and Newmont covers mineral or mining leases of Section 1 of Township 28 North, Range 30 East, Section 25 of Township 29 North, Range 29 East, all or portions of Sections 1, 3, 5, 7, 8, 11, 13, 17, 19, 21, 23, 25, 27, 29, and 35 of Township 29 North, Range 30 East and all or portions of Sections 23, 25, 27, 33, and 35 of Township 30 North, Range 30 East, Mount Diablo Base and Meridian, all in Pershing County, Nevada (Figure 4-2). These lands comprise the Trinity property subject to agreement with Newmont. Additionally, K9 has control of 70 unpatented lode (KS- claims on Table 4-1) and select portions of nine sections of private land (discussed below).

Table 4-1. Unpatented Lode Mining Claims of the Trinity Property

Claim Name	Owner of Claims
KS 1-70	K9 Gold Corp.
<i>70 claims total</i>	

Private land controlled by Newmont and leased by K9 from Newmont consists of:

LEASED LANDS: (Minerals Lease – 9426.08 acres, according to Exhibit B of the Newmont-K9 agreement)

Newmont leases the mineral rights on the land described below from The Atchison, Topeka, and Santa Fe Railway Company and Santa Fe Pacific Minerals Corporation through an Exploration Agreement and Option to Lease dated November 29, 1990. Newmont does not control the surface rights on this leased property.

Township 28 North, Range 30 East, MDB&M:

Section 1: All (643.36 acres)

Township 29 North, Range 29 East, MDB&M:

Section 25: All (640 acres)

Township 30 North, Range 30 East, MDB&M:

Section 23: All (640 acres)

Section 25: All (640 acres)

Section 35: N1/2, SE1/4, and N1/2 of SW1/4 (560 acres)

Township 29 North, Range 30 East, MDB&M:

Section 1: All (640.6 acres)

Section 3: All (639.12 acres)

Section 11: All (640 acres)

Section 17: All (640 acres)

Section 21: E/2, W/2 W/2, SE/4 SW/4, E/2 E/2 NW/4 (565 acres)

Section 23: All (640 acres)

Section 25: All, except E/2 NW/4 NW/4 (620 acres)

Section 27: All (640 acres)

Section 29: All (640 acres)

Section 35: All (640 acres)

LEASED LANDS: (Mining Lease – 1280.00 acres, according to Exhibit B of the Newmont-K9 agreement)

Newmont’s Mining Lease on the land described below from New Nevada Resources, LLC, through Mining Lease dated February 2, 2009.

Township 30 North, Range 30 East, MDB&M:

Section 23: All (640 acres)

Section 25: All (640 acres)



LEASED LANDS: (Mining Lease – 3826.36 acres, according to Exhibit B of the Newmont-K9 agreement) Newmont’s Mining Lease on the land described below from Scott Cooley, through Mining Lease dated April 1, 2010.

Township 28 North, Range 30 East, MDB&M:  
Section 1: All (643.36 acres)

Township 29 North, Range 29 East, MDB&M:  
Section 25: All (640 acres)

Township 29 North, Range 30 East, MDB&M:  
Section 23: All (640 acres)  
Section 25: All, except E/2 NW/4 NW/4 (620 acres)  
Section 27: All (640 acres)  
Section 35: All (640 acres)

LEASED LANDS: (Mining Lease – 1900.28 acres, according to Exhibit B of the Newmont-K9 agreement) Newmont’s Mining Lease on the land described below from New Nevada Resources, LLC, and New Nevada Lands, LLC, through Mining Lease dated December 3, 2014.

Township 29 North, Range 30 East, MDB&M:  
Section 7: All (639.64 acres)  
Section 13: All, except S/2 SE/4 NE/4 (620 acres)  
Section 19: All (640.64 acres)

LEASED LANDS: (Mining Lease – 1917.32 acres, according to Exhibit B of the Newmont-K9 agreement) Newmont’s Mining Lease on the land described below from New Nevada Resources, LLC, and New Nevada Lands, LLC, through Mining Lease dated October 28, 2015.

Township 29 North, Range 30 East, MDB&M:  
Section 5: All (637.32 acres)

Township 30 North, Range 30 East, MDB&M:  
Section 27: All (640 acres)  
Section 33: All (640 acres)

OWNED (Private) LANDS: Surface & Minerals (1,280 acres)  
Newmont owns the mineral and surface rights on the land described below.

Township 29 North, Range 30 East, MDB&M:  
Section 9: All (640 acres)  
Section 15: All (640 acres)

In addition to the above, the agreement includes two Temporary Use Permits by the BLM, currently held by Newmont:

Temporary Use Permit N-46510 for an access road on the following parcels:

Township 29 North, Range 29 East, MDB&M:

Section 11: S2NE4

Section 12: S2N2

Township 29 North, Range 30 East, MDB&M:

Section 8: N2S2

Temporary Use Permit N-46512 for a water well site and pipeline on the following parcels:

Township 29 North, Range 30 East, MDB&M:

Section 10: SW4NW4, N2SW4

Finally, the agreement includes Newmont's lease for its core storage facility in Sparks, NV.

Annual holding costs for K9's unpatented lode claims include collective fees payable to the U. S. Bureau of Land Management ("BLM") of \$274 per claim.

## 4.3 AGREEMENTS AND ENCUMBRANCES

### 4.3.1 EXPLORATION AGREEMENT WITH OPTION TO PURCHASE BETWEEN K9 GOLD CORP. AND NEWMONT

The following information has been provided by K9.

K9 acquired the exclusive right to conduct exploration work for minerals on all Newmont-owned and leased properties for Trinity through an exploration with option to purchase agreement with Newmont, effective September 16, 2025. K9 is the sole lessee of a 100% interest in that portion of the Trinity property which is controlled by Newmont (see Section 4.3.1), provided the following conditions:

1. Signing payment of \$50,000,
2. Initial Issuance of 100,000 common shares of K9 Gold Corp. to Newmont or its designated affiliate,
3. \$100,000 payment on or before the 1st anniversary of the effective date of the agreement, plus issuance of 300,000 K9 shares.

K9 also has the option to purchase the property by or on the second anniversary of the agreement by meeting the following obligations:

- / \$350,000 cash payment,
- / Issuance of 700,000 K9 shares to Newmont or Newmont's designated affiliate.

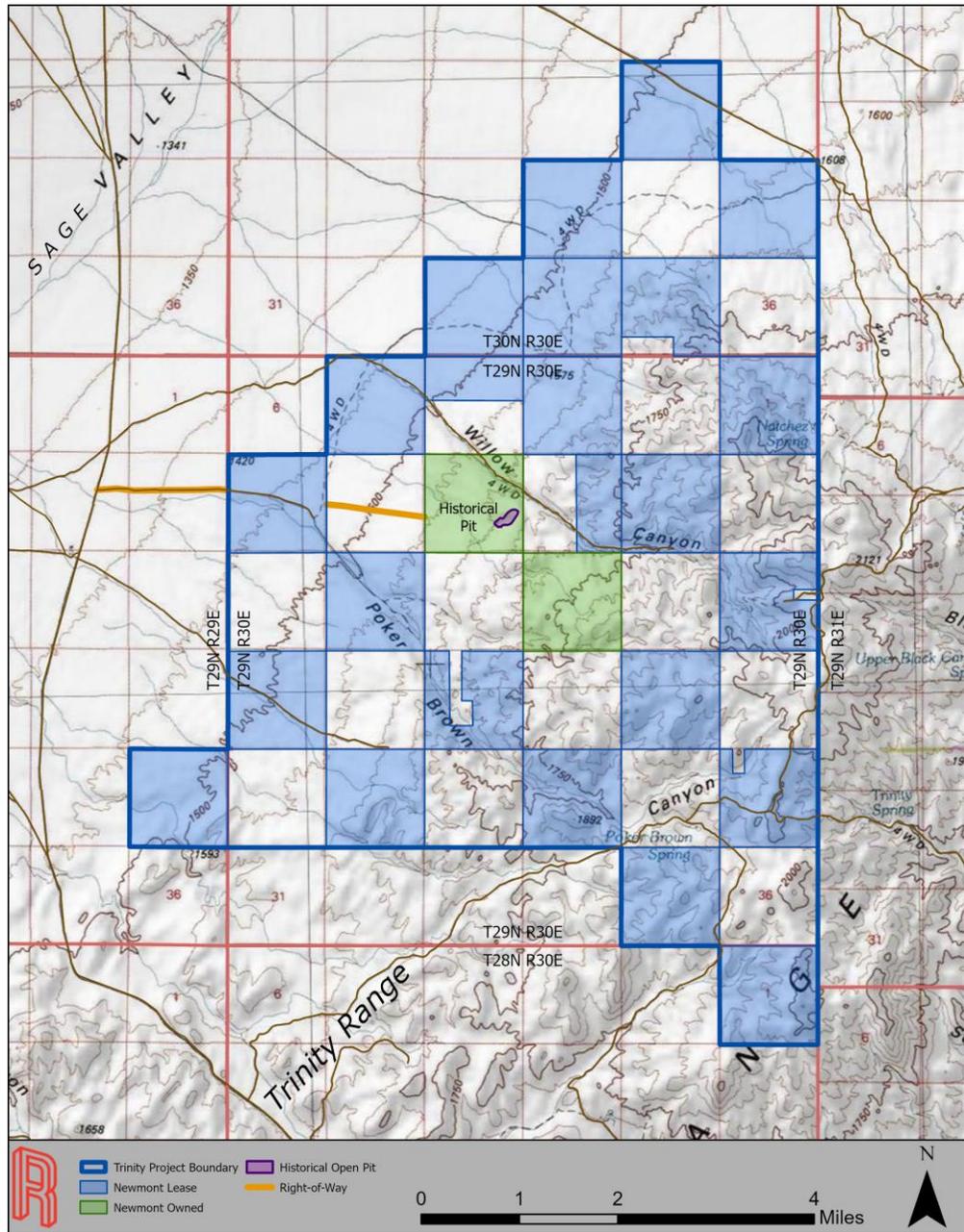


Figure 4-2. Property Map for the Trinity Project

### 4.3.2 ROYALTIES

#### 4.3.2.1 ROYALTIES APPLICABLE TO SECTIONS 9 AND 15

The land in Sections 9 and 15 that is covered by the agreement between K9 and Newmont is subject to a 2% net smelter returns royalty. Under the agreement, K9 has the option to buy back a portion of the royalty for \$3,500,000, reducing it from 2% to 1%.

#### 4.3.2.2 ROYALTIES APPLICABLE TO THE REMAINDER OF THE TRINITY PROPERTY

For the remainder of the lands leased for the Trinity Project, excluding Sections 9 and 15, there is an underlying 1.5% royalty associated with various leases in the agreement. These royalties are complex (Hartley et al., 2011). The author recommends that the validity of these royalties be determined by legal counsel.

### 4.4 ENVIRONMENTAL PERMITTING AND LIABILITIES

Exploration at the Trinity mine in the early to mid-1980s led to open-pit mining, milling, and heap leach processing in the late 1980s. The leach pad located to the south of the open-pit was closed with an evaporative cap and underwent drain-down monitoring until 2006, when the leach pad drain-down ceased. No further monitoring of the leach pad is required by the Nevada Department of Environmental Protection (“NDEP”), Bureau of Mining Regulation and Reclamation (“BMRR”). The base of the pad was lined prior to initiating processing in the 1980s. Any future work on, or expansion of, the leach pad will require discussion with NDEP prior to engaging in the work (Hartley et al., 2011).

Previous reclamation work conducted at the Trinity mine includes work performed by Borax, Santa Fe, Renaissance, and Newmont. The most recent reclamation work on the site was completed by Newmont as a condition of bond release. Reclamation work by Newmont was conducted in August 2010, which involved upgrading the safety berm around the pit and abandoning a potable water well. The Trinity mine property has been released from all previous permits and reclamation obligations.

K9 is planning to conduct exploration on the private and public lands in the vicinity of historical exploration that was conducted by prior operators. Permitting for work on private land is regulated by BMRR. Exploration on public land will be regulated by the BLM and requires compliance with the National Environmental Policy Act. Exploration on public land may occur under a Notice of Intent (“NOI”) to the BLM if surface disturbance is limited to five acres or less. If warranted or if planned surface disturbance is greater than five acres, the BLM may ask for supporting surveys, monitoring, or an Environmental Assessment (“EA”) prior to approval of permits.

If the BLM requires an EA-level of study to conduct exploration or other project activities, surveys to document the baseline conditions at the property may be called for. Common types of baseline studies include cultural, surface water and groundwater, vegetation, mammals and raptors, sensitive species, and socioeconomic impacts. Additionally, consultation with local stakeholders such as nearby communities and Native American groups may be required, along with public notices of the proposed action. Baseline surveys can typically be completed in one year and should be conducted before initiating the NEPA process for an EA, which can take an additional year to 18 months to complete, including time for agency reviews, public notice periods, and preparation of the EA document and public record.

#### 4.4.1 SURFACE AND GROUNDWATER

The Trinity Range has few surface waters, and none exist in the vicinity of the project site. The local drainages flow west to the playa during significant storm events, but are otherwise ephemeral channels. Previous mining did not encounter groundwater, and the water table in the area is reported to be several hundred feet below the bottom of the historical pit.



RESPEC

#### 4.4.1.1 SPECIAL STATUS SPECIES

The project property is on the northwestern flank of the Trinity Range, which is listed as known and potential habitat for the Greater Sage Grouse by the BLM. The project site is immediately south of the habitat boundary line, and therefore permitting for exploration on the public lands may require periods of non-activity to accommodate the grouse breeding season during the spring months. No other species of concern have been identified in the area (Hartley et al., 2011).

#### 4.4.1.2 ACCESS ROUTE

K9 will need to maintain the BLM road-use easement for the portion of the access road that extends from the county road to the private land. This is an existing easement on an existing improved road. Obtaining the BLM road-use easement is a matter of a renewal of the existing one and transferring it into K9's name.

#### 4.4.1.3 PERMITTING RISK

Based on current understanding of the site conditions and plans for future exploration of the property, the author believes that the risk of significant permit delays or denial is very low on this project. Due to the previous history of development on the property, good environmental use and reclamation record, and limited potential environmental exposure to surface and groundwater, the environmental concerns are believed to be minimal.

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

### **5.1 ACCESS**

The Trinity Project is located in the Willow Canyon area on the west side of the Trinity Range in the Trinity mining district. The project can be reached from Reno, Nevada, by traveling on Interstate 80 about 89 miles northeast to Lovelock, then northwest on Nevada state route 399 for 13 miles, then north on the Sulfur-Seven Troughs road about nine miles, and then east on the mine access road four miles to the property. The Sulfur-Seven Troughs road is a county-maintained gravel road. The Willow Canyon road is an improved dirt road.

### **5.2 CLIMATE**

The climate of Pershing County is typical of the high desert of northern Nevada. Annual precipitation is about six inches in the valleys and as much as 20 inches in the mountains, with most of it falling as rain or snow in the winter months and occasional summer thunderstorms (Johnson, 1977). Winter temperatures are generally in the 30s and 40s, but occasionally fall below zero. Summer temperatures are generally in the 70s, but reach 90°F to 100°F in the valleys.

Exploration and mining activities can be conducted year-round.

### **5.3 LOCAL RESOURCES AND INFRASTRUCTURE**

Agriculture and mining are the principal economic activities in this area. Lovelock, the county seat of Pershing County, has an estimated population of 2,411 as of 2009 (Nevada State Demographer website) and is a source of experienced labor, materials, and lodging. Reno, Nevada, is a major metropolitan area located about 89 miles southwest of Lovelock.

The Burlington Northern Santa Fe Railroad passes through Lovelock, as does Interstate Highway 80.

The nearest source of power is at Lovelock, Nevada. Power for mining at the project will need to be generated on-site.

Water used in prior mining operations came from wells that have since been reclaimed; new wells would have to be drilled to supply water for future mining. Water for exploration is brought in from Lovelock.

### **5.4 PHYSIOGRAPHY**

The Trinity property lies on the northwest flank of the Trinity Range, one of the generally north-trending mountain ranges of the Basin and Range physiographic province.

The ranges in the area reach elevations of over 7,000ft, and the valleys are generally between 3,900 and 4,700ft in elevation. The principal areas of known mineralization on the Trinity property lie between 5,300 and 5,500ft above sea level (Ashleman, 1984).

Vegetation in the area is very sparse and consists of sage and “salt” and “antelope” brush, with small junipers in some of the canyons and at higher elevations (Ashleman, 1984).

Figure 5-1 shows the physiography in the vicinity of the Trinity mine area.



**Figure 5-1. Physiography of the Trinity Mine Area**

View looking west from the Trinity Range, overlooking the old Trinity open-pit.

## 6.0 HISTORY

### 6.1 EXPLORATION AND MINING HISTORY

The following information has been taken from Ashleman (1987, 1988), Leonard et al. (1986), Muntean (1992), Whateley et al. (2006), Liberty Silver's website, and information provided by Liberty Silver, with other references as cited.

Mineralization was first discovered in the Trinity Range in 1859. There was limited, intermittent production of silver, lead, zinc, and gold from the Trinity district from 1864 through 1942. Most of the known production from the district came from gold-silver mines in the Trinity Canyon area on the east side of the range. The Trinity Project, located on the west side of the range in the Trinity district, was historically prospected with unrecorded but minor silver production. Johnson (1977) noted the presence of small mines on veins in the vicinity of the Trinity Project, but indicated their history was not known. Ashleman (1984) reported that lead-silver-gold veins occur along northeast-trending shear and breccia zones in Triassic sedimentary rocks and were mined locally; most of them occur in Sections 2 and 3, T29N, R30E, and in the Willow Canyon area, with several small prospect pits between the two areas. Tingley (1985) reported that older silver-lead workings north of Willow Creek in Section 3, T29N, R30E are located along brecciated and highly oxidized quartz veins in northeast-trending shears in Triassic-Jurassic metasedimentary rocks. Two samples from old workings in Section 3, taken as part of Tingley's work, returned 0.20 and 0.45 ppm gold, >2,000 ppm arsenic, >1,000 ppm antimony, >2,000 ppm zinc, 500 and 1,000 ppm silver, and 100 and 300 ppm tin, with anomalous bismuth and cadmium. Ashleman (1984) reported that there are several small prospect pits and other old workings, as well as evidence of minor activity in the 1950s, in the Willow Canyon area, and that Phelps Dodge Corp. ("PD") completed trenching, IP surveys, and limited drilling in the 1960s. Most of PD's work was in the Triassic rocks north of Willow Creek, but some trenching was done south of Willow Creek along the contact of Tertiary rhyolite and Triassic metasedimentary rocks.

#### 6.1.1 U.S. BORAX AND CHEMICAL CORPORATION

As described by Ashleman (1984), Knox, Kaufman, Inc. of Spokane, Washington, conducted precious-metals exploration for U. S. Borax and Chemical Corp. ("Borax") in northern Nevada in early 1982 and sent a geologist to return to the Trinity area to duplicate an anomalous gold sample he had taken several years earlier in the Willow Creek area. Although the original sample was not duplicated, anomalous silver values were obtained. Additional work showed an extensive anomalous lead zone with many anomalous silver values in rhyolites to the south. Knox, Kaufman, Inc. recommended the Willow Canyon area to Borax as a potential bulk-tonnage precious-metal occurrence (Ashleman, 1988).

Land agreements were entered into with the owners of five existing lode mining claims and with Southern Pacific Land Company, later Santa Fe Pacific Mining, Inc. ("SFPM"), who owned private land in the area. The "SEKA" unpatented mining claims were staked by Pacific Coast Mines, Inc., a wholly owned subsidiary of Borax, to cover adjacent open ground. Pacific Coast Mines, Inc. and SFPM entered into a joint-venture operating agreement on January 30, 1984, for the Trinity Project. Borax was named operator and managed exploration and development work on the Trinity Project.

From 1982 to 1986, Borax and its joint-venture partner for part of the project lands, SFPM, explored the property (primarily Sections 9 and 15). During exploration, Borax conducted an IP and resistivity survey in 1983, airborne magnetic surveys in 1984 and 1987, and a VLF survey in 1985. Mining Geophysical Surveys, Inc. of Tucson, Arizona, performed the 1983 IP resistivity survey (Wieduwilt, 1983). The survey consisted of 15 east-west lines with a 1,000ft line interval that formed the gradient array grid. The lines were about 8,500ft long with readings taken every 500ft. Two test lines using a 200ft dipole-dipole configuration were also run to further analyze anomalous IP trends. The VLF survey, conducted by SFPM at Borax's request, included north-south and east-west lines (Hendrickson, 1985). Of the geophysical surveys conducted over the sulfide mineralization, the magnetic survey was of little value because of poor contrast in magnetic values. Gamma-ray spectrometry using K, Th, and U was tested but was also found to be of little value. The gradient array IP resistivity survey did reveal a chargeability anomaly that coincided with the area of sulfide mineralization, and it also revealed a north-northwest-trending belt of high resistivity located adjacent to the eastern margin of an area of known subsurface mineralization (Ashleman, 1984).

Borax also carried out surface and trench rock-chip sampling; geologic mapping; a soil geochemical survey; percussion, RC, and core drilling; and metallurgical test work on sulfide and oxide mineralization.

Geologic mapping was initially at a scale of 1:500, followed by detailed mapping of the mineralized zone at a scale of 1:100. A soil geochemical survey was completed over the oxide and sulfide mineralized areas, with samples analyzed for lead, zinc, and silver. Because it was more stable in the soil horizons, lead was used as a pathfinder element for the silver mineralization (Ashleman, 1984). Significant lead anomalies were identified over the sulfide zone. Anomalies with >100 ppm lead defined potentially mineralized areas, and higher lead levels of >1,000 ppm coincided with silver mineralization at the surface. Anomalous lead values persisted even over buried silver mineralization.

Surface rock geochemistry was used as part of reconnaissance exploration, with results helping to define the extent of surface mineralization (Ashleman, 1984). Rock chips from drilling, trenching, and surface sampling were analyzed for lead, zinc, silver, and gold. Anomalies from the rock and soil geochemical surveys were used to plan the early drilling programs.

According to Leonard et al. (1986), 236 holes totaling 90,342 feet were drilled on the property from 1982 through the 1985 field season, which delineated the mineralization on approximately 100-foot centers. Additional drilling was undertaken during mining, and as of May 1989, a total of 281 holes totaling 104,266 feet had been drilled since 1982 (Baele and Pelletier, 1989). According to Baele and Pelletier (1989), drilling was generally performed on a 100ft grid aligned N11°W of true north.

Borax and its joint-venture partner SFPM developed the Trinity open-pit mine, which Borax mined on behalf of the joint-venture from September 3, 1987, to August 29, 1988, through a contract miner, with leaching continuing into 1989. The mine was subsequently reclaimed. From 1987 to 1989, the mine produced about five million ounces of silver from about 1.1 million tons of oxidized ore grading about six ounces of silver per ton (Baele and Pelletier, 1989). Borax drilled and conducted extensive metallurgical testing of the sulfide mineralization. However, metal prices were too low to support mining of the sulfide mineralization. Leonard et al. (1986) presented an economic analysis and mining plan for the deposit as estimated in 1986, prior to the mining by Borax. Borax prepared a subsequent

development plan for the sulfide and undeveloped oxide mineralization following completion of oxide mining in 1988 (Anon., 1988).

#### **6.1.2 SANTA FE PACIFIC MINING, INC.**

In 1984 and 1985, SFPM drilled 26 holes (TR-series), mostly targeting extensions of the Trinity silver mineralization in Section 9 onto their own private ground in Sections 3 and 17. Their best intercept was 20 feet of 1.25 oz Ag/ton in a narrow shear zone in Mesozoic argillite (hole TR-6 in Section 3; Muntean, 1992). SFPM also tested a separate, unrelated area of mineralization in Section 27 of T30N, R30E. In 1987-1989, SFPM staked claims and drilled 22 holes (TR 87-series, TR 88-series, and TS-series), which did not identify significant mineralization (Muntean, 1992). Drilling in the alluvial-filled valley to the west found thicknesses of alluvium of up to at least 700 feet in places.

In 1990, SFPM undertook a CSAMT geophysical survey, an incline dipole-dipole survey, and an IP resistivity survey. The CSAMT survey, conducted by Phoenix Geoscience, Inc., consisted of 27 line miles in five N45°W profiles and one tie line run at N45°E with a station spacing of 660ft (Ostrander, 1990). It confirmed the thicknesses of alluvium encountered in drilling in the valley and indicated the location of the northeast-trending range-front fault. SFPM subsequently dropped many of the claims located west of the inferred fault. The IP resistivity survey along and south of Willow Creek, conducted by Practical Geophysics for Kennecott Exploration Company ("Kennecott") (who by then controlled Borax's interest as described below), was plotted and interpreted by Great Basin Geophysical, Inc. (Lide, 1991). Three northwest-trending lines spaced 1,000 feet apart were surveyed with a dipole-dipole array with a dipole length of 300 feet; the middle line was extended to the southeast using a 200-foot dipole length. The lines identified a generally north-striking range-front structure. Moderate IP response associated with the higher-resistivity rock in the southeast portion of the lines suggested potential for sulfide mineralization in this area.

SFPM acquired sole interest in the joint-venture lands (Sections 9 and 15) and Borax claims (Sections 4, 8, 10, and 16), which by then were owned by Kennecott, that surrounded the joint-venture area through an agreement dated January 31, 1991. SFPM proceeded to compile all Borax and Kennecott data and conduct further exploration on the property through 1992. Kennecott had acquired Borax's unpatented mining claims by quitclaim deed dated May 1, 1990, and Borax's leased mining claims by an assignment effective May 1, 1990 (Roesch, 1990) when Rio Tinto Zinc Corp., which had previously acquired Borax in 1968, bought Kennecott in 1989. Kennecott then reconveyed the claims to Borax prior to termination of the SFPM-Borax joint-venture in January 1991 (Trubey, 1991b). SFPM subsequently dropped the leased claims in Section 10. The 1990-1992 exploration work concentrated on down-dip and lateral extensions of mineralization underlying the oxide pit and the sulfide mineralization, as well as extensions of mineralization outside the immediate mine area. Seven RC angle holes (DTS-1 through DTS-7) were drilled around the immediate mine area based on analysis of silver grade-thickness plots (Muntean, 1992). Based on that drilling, SFPM concluded that the mineralization does not plunge, but instead maintains its intensity to the southwest. Drilling beneath the ridge of silicified tuffs that were thought to be the center of mineralization did not encounter any significant silver or gold mineralization. SFPM's drilling identified additional low-grade silver, lead, and zinc mineralization at depth, but the results also indicated that the mineralization continues to narrow.

In exploring beyond the mine area, SFPM analyzed over 2,000 rock-chip, trench, and soil geochemical samples; carried out geologic mapping and air photo interpretation; examined existing drill data; and reviewed aeromagnetic, CSAMT, IP/resistivity gradient array, and dipole/dipole geophysical surveying (Muntean, 1992). In 1992, SFPM drilled four holes (DTS-8 through DTS-11) outside the mine area on SFPM private land, but failed to encounter significant mineralization.

### **6.1.3 RENAISSANCE GOLD INC.**

There was no significant exploration at the Trinity property from 1993 to 2005. In August 2005, Renaissance leased the property from Newmont, which had acquired SFPM's Nevada holdings. Renaissance explored the property with Piedmont Mining Company under the terms of an earn-in agreement signed in September 2005. Renaissance drilled 10 angled core holes (TSD-series) in 2006 and 15 RC holes (TS07-11 through TS07-25) in 2007 and encountered high-grade silver values in the sulfide zone below and adjacent to the open-pit. Renaissance's earn-in agreement with Piedmont was terminated on July 17, 2009 (AuEx website, news release dated September 1, 2009).

Renaissance and Yellowcake Mining, Inc. entered into a Letter of Intent in August 2009 that expired December 31, 2009. No exploration work was conducted during this period.

### **6.1.4 LIBERTY SILVER CORP.**

Liberty Silver Corp. ("Liberty") entered into an earn-in agreement with Renaissance in March 2010. Liberty began exploration of the Trinity Project in May 2010 by contracting with Industrial Imaging Co. of Salt Lake City, Utah, to conduct a geophysical survey of the project area using its proprietary telluric-magnetotelluric ("TMT") method. The survey was designed to identify additional concealed mineralized zones under the pediment cover to generate a 3D subsurface image of the project area. Measurements were taken at 112 stations within the core area of the property, measuring frequencies from 2Hz to 220Hz.

In addition to the geophysical work, Liberty conducted extensive data compilation for the project, including the results of soil and rock surveys, geophysical surveys, geologic mapping, lithology logging, and multi-element analyses for about 400 drill holes, and metallurgical work, as well as previous production of heap leach silver.

In 2012, Liberty drilled 20 reverse circulation holes at Trinity. Concurrent to the drilling, Liberty conducted a gravity survey by MaGee Geophysics and an induced polarization (IP) survey by Zonge Geophysics.

Liberty terminated its agreement with Renaissance in August 2017. Subsequently, Renaissance apparently relinquished the property back to Newmont prior to K9's involvement, but the author is not aware of the details related to this change in control.

## 6.2 HISTORIC MINERAL RESOURCE AND RESERVE ESTIMATES

All estimates described in this section were prepared prior to establishment of NI 43-101 reporting requirements. There are insufficient details available on the procedures used in these estimates to permit the author to determine if the estimates meet NI 43-101 standards. The classification terminology is presented as described in the original references, but it is not known if it conforms to the meanings ascribed to the measured, indicated, and inferred mineral resource classifications or proven and probable reserve classifications by the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Standards). Accordingly, these estimates should not be relied upon, and are presented herein merely as an item of historical interest with respect to the exploration targets at Trinity, and should not be construed as being representative of actual mineral resources or mineral reserves (under NI 43-101) present at the Trinity Project.

Ashleman (1984) reported preliminary "geologic ore reserves", based on exploration in 1982 and 1983, of 2.72 million tons grading 3.04 oz Ag/ton with a 5.63:1 stripping ratio at a 1.5 oz Ag/ton cutoff and 4.01 million tons grading 2.49 oz Ag/ton with a 3.48:1 stripping ratio using a 1.0 oz Ag/ton cutoff within a pit defined using the 1.5 oz Ag/ton cutoff. This estimate was based on holes S-1 to S-72 and used a tonnage factor of 13 ft<sup>3</sup>/t. No mining dilution or metallurgical recovery factors were applied.

In 1986, Borax completed three "total geologic reserve" calculations for the Trinity deposit, including both Borax and joint-venture ground (Ashleman, 1987; Table 6-1). A polygonal "ore reserve" calculation was done in January 1986 using a 1.0 oz Ag/ton cutoff within the main zone and a 1.5 oz Ag/ton cutoff within the Southwest Extension (the small high-grade oxide body). Deep and/or peripheral intercepts were not used when it was felt they would have a high strip ratio. Two separate cross-sectional "reserve" calculations were completed using a 1.0 oz Ag/ton cutoff. Thin, deep, or isolated low-grade intercepts were discarded. Grade zones, based on knowledge of the geology and the nature of the mineralization, were subdivided by drawing boundaries midway between each hole and were not extended more than 50 feet from a drillhole without additional evidence of the mineralization extending further. The reported oxide values in Table 6-1 include the material that was subsequently mined from the Trinity open-pit. Mr. Bickel has not sufficiently evaluated these historic estimates 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. These historic estimates should not be relied upon.

Table 6-1. 1986 Trinity "Geologic Reserve Calculation" Prior to Mining

(From Ashleman, 1987; Leonard et al., 1986; Reim, 1989b)

Method and Source	Tonnage Factor (ft <sup>3</sup> /t)	Min. Intercept Length (feet)	Cutoff (oz Ag/ton)	Material Type	Tons (millions)	Average Grade (oz Ag/ton)	Total Ag (million ounces)
N-S Cross-sections (Ashleman, 1987; Leonard et al., 1986; Reim, 1989b) "demonstrated geologic reserves"	13.0	10	1.0	Total	9.90	2.98	29.5
				Oxide	3.08	3.63	11.2
				Sulfide	6.81	2.68	18.3
			approx. 1.5	Total	7.99	3.40	27.1
				Oxide	2.18	4.63	10.09
				Sulfide	5.81	2.93	17.02
			2.0	Total	5.83	4.01	
				Oxide	1.48	6.00	
				Sulfide	4.35	3.36	16.926
E-W Cross-sections Ashleman, 1987	13.0	10	1.0	Total	9.275	3.06	28.4
				Oxide	2.803	3.91	11.0
				Sulfide	6.472	2.70	17.5

\*The polygon calculation is not comparable to the two cross-section calculations because it covered a restricted area. Cutoff is not strictly 1.5 oz Ag/ton cutoff, as material between grading between 1.0 and 1.5 oz Ag/ton was selectively included.

Borax (Ashleman, 1987) and Santa Fe (Whateley et al., 2006) made undiluted "reserve" estimations of the high-grade oxide mineralization at the Southwest Extension using a tonnage factor of 13.3ft<sup>3</sup>/t (Table 6-2). Mr. Bickel has not sufficiently evaluated these historic estimates 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. These historic estimates should not be relied upon.

**Table 6-2. 1986 Southwest Extension Oxide "Reserve"**

(From Ashleman, 1987, and Whateley et al., 2006)

Method and Company	Composite Length (feet)	Cutoff (oz Ag/ton)	Tons (millions)	Average Grade (oz Ag/ton)	Total Ag (million oz)
Polygons Borax	20	1.5	0.967	6.95	6.22
N-S cross-sections Borax	10	1.5	1.304	6.16	8.03
E-W cross-sections Borax	10	1.5	1.293	5.90	7.63
N-S cross-sections Borax	10	1.5	0.932	7.69	7.17
Polygons Santa Fe	20	3.0	0.669	9.10	6.09
N-S cross-sections Santa Fe & Borax	10	2.0	0.870	8.00	6.96

After mining less than two months and completing the first three 15-foot benches, Borax found that the tons of ore and contained ounces of silver were significantly different from those estimated in the initial mine plan (Reim et al., 1988). Additional drilling was then completed and a new hand-calculated ore "reserve" for the oxide ore body was undertaken in January 1988 using a lower cutoff grade that resulted in about 14% less material to be mined, but at a higher-grade than in the July 1987 estimate (Table 6-3). Both estimates used a specific gravity of 13.7 ft<sup>3</sup>/t and a silver recovery of 79%, although Reim et al. (1988) noted that in the range of 1 to 2 oz Ag/ton, recovery was estimated to be 65% of the total contained silver.

**Table 6-3. July 1987 Initial and January 1988 Revised "Reserve" Estimate for the Oxide Ore Body**

(From Reim et al., 1988)

	Cutoff (oz Ag/ton)	Tons	Grade (oz Ag/ton)	Silver (oz)
Initial July 1987 estimate	2.0	1,175,633	5.12	6,022,810
Revised January 1988 estimate	1.6	850,624	7.33	6,290,677

In August, 1988, following mining of the oxide deposit, Borax estimated the undeveloped oxide and sulfide silver "reserves" adjacent to and east of the Trinity open-pit (Reim, 1988; Anon., 1988; Table 6-4). Mr. Bickel has not sufficiently evaluated these historic estimates 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. These historic estimates should not be relied upon.

Table 6-4. 1988 Trinity Total Undeveloped "Reserves"

(Modified from Reim, 1988; Anon., 1988)

Material Type	Cutoff (oz Ag/ton)	Mineable Tons (millions)	Grade (oz Ag/ton)	Recovery (%)	Recoverable Ag (million ounces)
Oxide	1.5	0.213	2.66	66	0.374
Sulfide		4.400	2.7	78.4	9.314
Total					

Baele and Pelletier (1989) updated the mineral inventory for the larger but lower-grade sulfide mineralized zone northeast of the open-pit and the remaining oxide reserves in May 1989 using all available drillhole data (Table 6-5). Their estimates used a cutoff grade of 1.0 oz Ag/ton and a bulk-density factor of 13.3ft<sup>3</sup>/t.

The "geologic reserves" include: (1) the main sulfide zone and associated near-surface oxide mineralization, which Baele and Pelletier categorized as "measured reserves;" (2) remaining oxide and sulfide "reserves" under the existing open-pit; and (3) low-grade oxide and sulfide material encountered in the pit that was stockpiled separately during mining. Baele and Pelletier felt that their estimates were somewhat conservative due to dilution introduced by compositing, the relatively large block size (50 x 50 x 15ft high), and the smoothing effect of the interpolation process. Baele and Pelletier (1989) further estimated that the total minable reserves of the sulfide body within an ultimate pit shell constructed using a 1.37 oz Ag/ton break-even cutoff grade, an overall 50° pit slope, mining of all material above the cutoff grade without regards to stripping ratio, and a \$10/oz silver price was 4.367 million tons grading 2.41 oz Ag/ton (10,518,286 ounces) with a stripping ratio of 4.3:1.

Table 6-5. 1989 Borax Summary of Remaining "Mineral Reserves" at Trinity

(From Baele and Pelletier, 1989)

	Cutoff Grade (oz Ag/ton)	Tons (thousands)	Grade (oz Ag/ton)	Ounces Ag (thousands)
<b>OXIDE</b>				
Project area	1.0	867	1.84	1,595
Under open-pit	1.0	146	2.02	295
Stockpile	NA	398	1.17	466
<i>Subtotal Oxide</i>		<i>1,411</i>	<i>1.67</i>	<i>2,356</i>
<b>SULFIDE</b>				
Project area	1.0	4,803	2.15	10,326
Under open-pit	1.0	522	2.92	1,524
Stockpile	NA	31	11.59	359
<i>Subtotal Sulfide</i>		<i>5,356</i>	<i>2.28</i>	<i>12,209</i>
<i>Total</i>		<i>6,767</i>	<i>2.15</i>	<i>14,565</i>

The stockpiled material summarized in Table 6-5 had been estimated by Borax in March, 1989 (Reim, 1989a; Table 6-6). Reim noted that these estimates are based on fire assays.

Table 6-6. 1989 Mineral "Reserve" Stockpile at the Trinity Mine  
(From Reim, 1989a)

	Tons	Grade oz Ag/ton	Contained Ag oz
Oxide	397,744	0.94 to 1.40	417,022
Sulfide	30,890	9.48 to 13.70	365,747
Total	428,634		782,769

FSFPM reported that the Trinity deposit contained a sulfide "reserve" of four million tons averaging 2.5 oz Ag/ton at a cutoff of 1.4 oz Ag/ton. These reserves lay directly to the northeast of the existing open-pit (Santa Fe Pacific Mining, Inc., undated; Muntean, 1992).

Hartley et al. (2011) estimated mineral resources for Renaissance at the Trinity Project. According to the report, the Trinity mineral resources were modeled and estimated by:

1. Statistically evaluating the drillhole data to determine silver, lead, and zinc mineral domains;
2. Interpreting the mineral domains in cross section at 100ft spacings;
3. Geostatistical analysis of modeled domains to determine estimation parameters; and
4. Interpolating grades into a three-dimensional block model.

The estimates were classified as inferred and tabulated by separate redox zones (Table 6-7 and Table 6-8). The sulfide zone included estimates for lead and zinc as well as silver and were presented at various cutoff grades.

Table 6-7. Trinity Inferred Mineral Oxide Resources

Cutoff (oz Ag/ton)	Inferred Oxide Resources		
	Tons	oz Ag/ton	oz Ag contained
0.30	12,019,000	0.54	6,490,000
0.40	5,506,000	0.78	4,295,000
0.50	2,863,000	1.1	3,149,000
<b>0.65</b>	<b>1,901,000</b>	<b>1.37</b>	<b>2,605,000</b>
1.00	1,019,000	1.87	1,906,000
2.00	203,000	4.08	828,000

Table 6-8. Trinity Inferred Mineral Sulfide Resources

Cutoff (oz/ton Ag equiv)	Inferred Sulfide Resource				
	Tons	oz Ag/ton	% Pb	% Zn	oz Ag
1.00	8,408,000	1.27	0.23%	0.43%	10,691,000
1.20	6,113,000	1.56	0.25%	0.43%	9,539,000
<b>1.30</b>	<b>5,336,000</b>	<b>1.69</b>	<b>0.25%</b>	<b>0.43%</b>	<b>9,036,000</b>
1.50	4,119,000	1.97	0.26%	0.42%	8,100,000
2.00	2,288,000	2.70	0.30%	0.37%	6,170,000
3.00	902,000	4.14	0.32%	0.33%	3,731,000

Mr. Bickel has not sufficiently evaluated these historic estimates 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. These historic estimates should not be relied upon.

### 6.3 HISTORIC MINE PRODUCTION

The Trinity deposit was placed into production in September 1987 as an open-pit, cyanide heap leach operation. Mining was done under contract by Lost Dutchman Construction Company of Sparks, Nevada, at an average production rate of 18,000 tons per day and at a cutoff grade of 1.3 oz cyanide extractable silver per ton (Ashleman, 1988). Ore was crushed to -3/4 inch, agglomerated, and placed onto cyanide leach pads. Silver was recovered by the Merrill-Crowe process.

Borax reported that they mined a total of 1,085,790 tons of silver oxide ore at an average grade of 6.32 oz Ag/ton from the Trinity mine (Baele and Pelletier, 1989). A total of 0.14 ounces of gold were reportedly recovered with every 1,000 ounces of silver. An undated, anonymous summary that is believed to have been produced by Renaissance staff reported that the estimated silver recovery was 75% and the cutoff grade during mining was 1.3 oz Ag/ton. The pre-mining estimate for the oxide Southwest Extension pit consisted of 1.33 million tons of "mineable ore" averaging 6.05 oz Ag/ton (diluted) for a total of 8.05 million ounces of silver, of which 6.04 million ounces were thought to be recoverable based on 75% recovery from initial column leach tests (Leonard et al., 1986).

Figure 6-1 is a recent photograph taken from within the Trinity pit.



**Figure 6-1. View of the Historic Liberty Silver Open-Pit**

Northeast end of the historic open-pit. The major controlling fault for the mineralization lies at the back of the pit on the right side.

## 7.0 GEOLOGIC SETTING AND MINERALIZATION

### 7.1 GEOLOGIC SETTING

#### 7.1.1 REGIONAL GEOLOGY

The following information was largely taken from Johnson (1977).

Pershing County lies in the northern portion of the Basin and Range Province, a region characterized by generally north-trending mountain ranges and intervening alluvial-filled basins formed by Tertiary extension. The Trinity Range is one of these generally north-trending ranges.

During parts of the Paleozoic to Middle Jurassic time, northern Nevada was the site of dominantly marine deposition, with the continental shelf and associated carbonate rocks lying in the eastern part of the state and deeper water deposition of siliceous rocks with associated volcanism to the west. Representative units in Pershing County are generally heterogeneous sequences of mafic volcanic rocks, chert, clastic sedimentary rocks, and minor limestone, which are primarily exposed in the eastern third of the county, east of the Trinity Range. From Middle Triassic to Early Jurassic time, there was regional uplift, producing near-shore deltaic deposits of mudstone, shale, and sandstone that were deposited over much of what is now Pershing County; the Auld Lang Syne Group, exposed in the Trinity Range and found in the Trinity Project area, is part of these deposits.

Pre-Cenozoic deformation in northern Nevada was characterized by three major compressional events, of which the youngest is best represented in western Pershing County, including the Trinity Range. The oldest was the Late Devonian and Early Mississippian Antler Orogeny, during which deep-water sedimentary and volcanic rocks were thrust eastward tens of miles over shallow-water sedimentary rocks of the continental shelf. Evidence of this deformation, particularly the Roberts Mountains thrust, is best displayed in northeastern Nevada, east of Pershing County. Compression during the Early Triassic Sonoma Orogeny again involved transport of deep-water sequences eastward over shallow-water units, now exemplified by the Golconda thrust that is exposed in the Sonoma Range of Pershing County and in ranges to the east. The Nevadan Orogeny took place during Jurassic and Cretaceous time and resulted in low-grade regional metamorphism, variably directed folding, and thrust faulting that affected all pre-Tertiary rocks in Pershing County, including the Trinity Project area.

Plutonic rocks of Early Triassic, Middle Jurassic, Cretaceous, and Late Cretaceous ages are found in Pershing County, of which a Cretaceous granodiorite stock present just northeast of the Trinity Project is the best example in the project area. The Cretaceous intrusive episode represented the culmination of the Nevadan Orogeny.

Tertiary volcanic and sedimentary rocks and Quaternary sediments are widespread in Pershing County and are abundant in the Trinity Project area. Cenozoic extension created the Basin and Range physiography that dominates present-day Pershing County.

Figure 7-1 shows the regional geology of much of Pershing County.

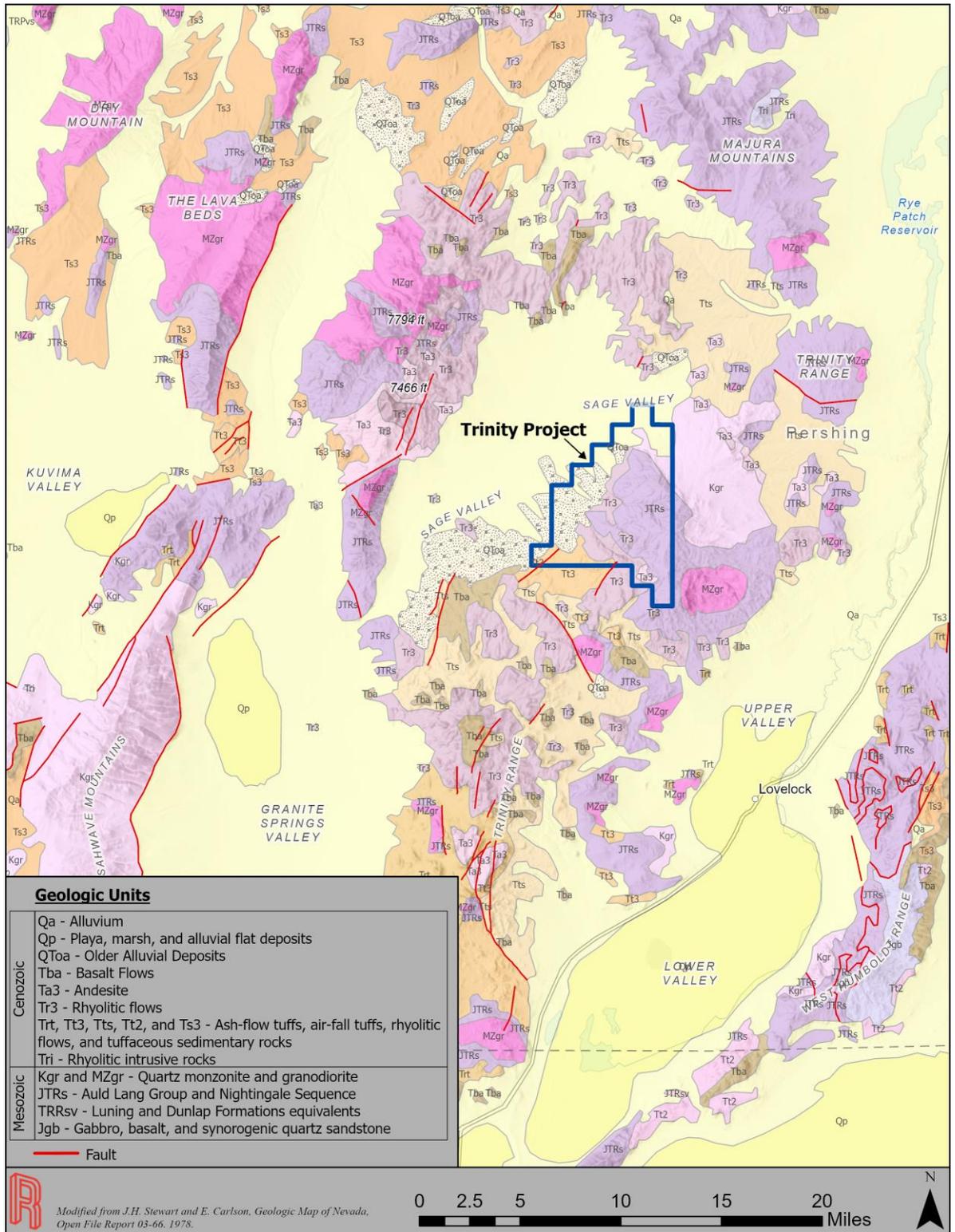


Figure 7-1. Regional Geology of the Trinity Range Area, Pershing County

### 7.1.2 LOCAL GEOLOGY

The following information has been taken from Leonard et al. (1986), Ashleman (1988), Santa Fe Pacific Mining, Inc. (undated), and Whateley et al. (2006).

In this part of the Trinity Range, a thin Tertiary rhyolite sequence is underlain by Triassic to early Jurassic phyllite and argillite of the Auld Lang Syne Group that form the basement of the Trinity Range. The Auld Lang Syne Group is composed of fine-grained clastic shelf and basin deposits with interbedded turbidites. Both regional metamorphism and contact metamorphism, related to intrusion of Cretaceous granodiorite dikes and stocks northeast of the mine, affected the Auld Lang Syne Group. A fine-grained, matrix-supported argillite breccia that is closely associated with faulting marks the contact between the Auld Lang Syne Group and the Tertiary rhyolitic volcanic rocks.

There is evidence of several episodes of structural deformation in the area. During the Nevadan Orogeny in Jurassic to early Cretaceous time, the Auld Lang Syne Group was strongly folded and faulted; low-grade regional metamorphism accompanied this deformation. Early Tertiary north- to northwest-trending faults are present within the Trinity mine area. The most prominent structural features in the area are north- to northeast-trending normal faults of Tertiary age, which offset all units except the gravel and alluvial deposits. .

Tertiary hydrothermal alteration altered the Auld Lang Syne Group only locally along faults and breccia zones. In contrast, Tertiary rhyolitic tuffs and flows were extensively altered with a halo extending 1.6 miles beyond the main mineralized area.

The Trinity property lies along the eastern edge of a Tertiary basin filled with rhyolitic tuffs, flows, and intrusive rocks. The basement rocks to the east consist of Mesozoic argillite and quartzite and a Cretaceous granodioritic stock. Figure 7-2 shows the geology of the Trinity area.

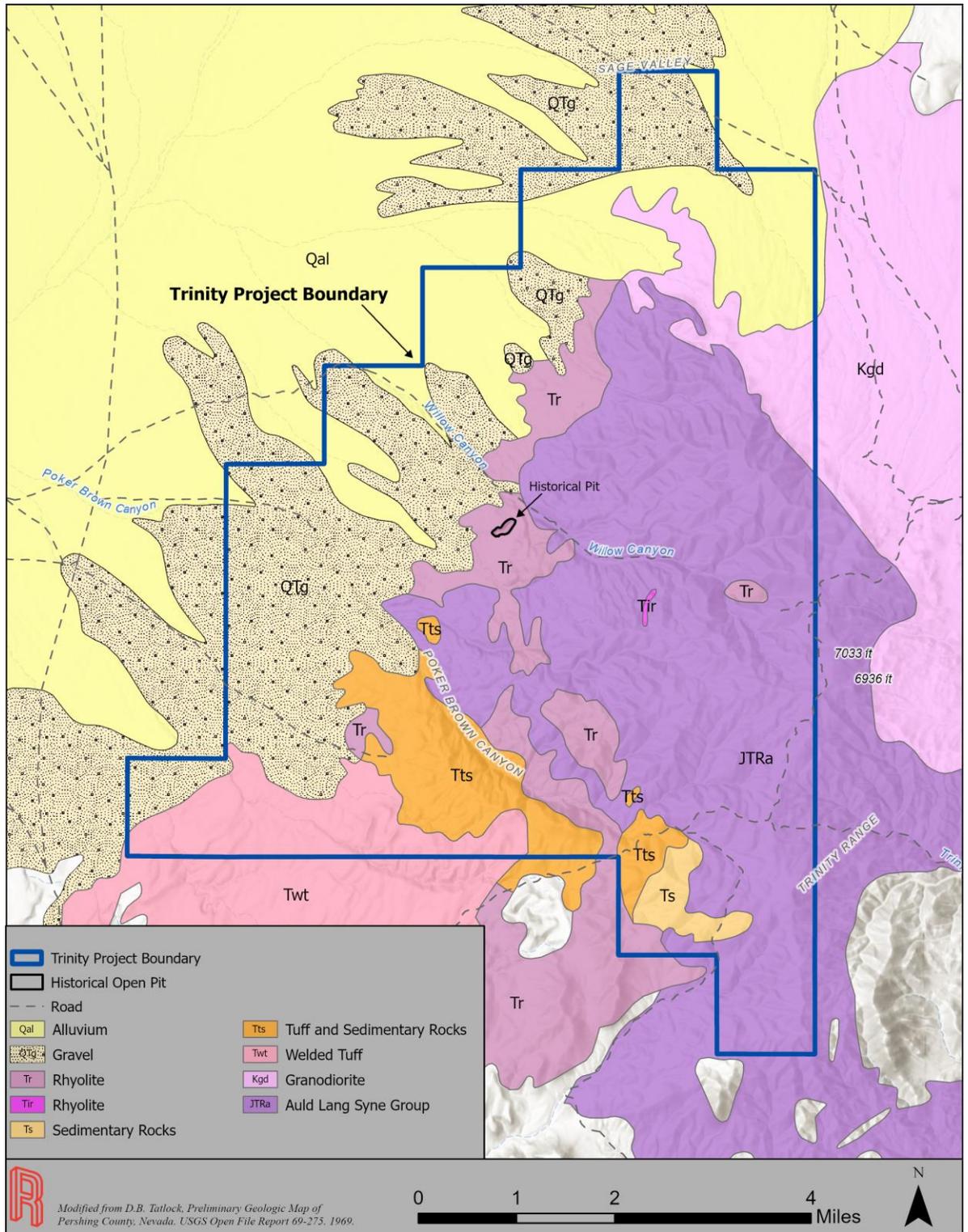


Figure 7-2. Generalized Geology of the Trinity Area

### 7.1.3 PROJECT GEOLOGY

The following information has been taken from Leonard et al. (1986), Ashleman (1984, 1988), Santa Fe Pacific Mining, Inc. (undated), and Whateley et al. (2006) with other references as cited.

In this portion of the Trinity Range, Mesozoic argillite and quartzite that form the basement of the range on the east are overlain by Tertiary rhyolite to the west. Alluvium covers the western and northern portions of the project area. There are numerous northeast-, northwest-, and north-trending faults.

The oldest rocks exposed within the Trinity Project area are Triassic marine sedimentary rocks of the Auld Lang Syne Group, which crop out in the eastern part of the property. Three units have been identified: locally calcareous quartzite and sandstone; dirty limestone with calcareous siltstone and sandstone; and argillite and siltstone with subordinate quartzite and sandstone. These rocks have been intruded by Cretaceous granodiorite to the northeast of the project area. The Auld Lang Syne Group has undergone low-grade regional metamorphism and local contact metamorphism.

A Tertiary tectonic breccia crops out in several locations within the project area; it is particularly abundant along the rhyolite-argillite contact in Section 10 just south of Willow Creek and also along the north side of Willow Creek. The breccia consists of angular fragments of argillite in a fine-grained matrix of argillite. The breccia is generally well cemented and forms resistant outcrops.

Tertiary rhyolite unconformably overlies and locally intrudes the Auld Lang Syne Group. The rhyolite includes interbedded rhyolitic flows, welded tuffs, air-fall tuffs, epiclastic tuffs, and lacustrine deposits. The lower part of the volcanoclastic section consists primarily of air-fall and reworked tuffs; pyroclastic rocks, including welded tuffs and pyroclastic flows along with local phreatic-clastic deposits dominate the upper part of the section. Several rhyolite domes, dikes, and sills have also been identified on the property, some of which may be related to mineralization (Leonard et al., 1986). Rhyolite porphyry and aphanitic rhyolite intruded into the volcanoclastic rocks and formed exogenous domes and flows. Late-stage latitic to rhyolitic dikes, locally called "sugary rhyolite," post-dated most of the rhyolitic rocks. The rhyolite porphyry, the aphanitic rhyolite, and the volcanoclastic rocks are the principal host rocks for the mineralization in the Trinity mine area (Ashleman, 1988).

Late Tertiary and/or Quaternary bench and channel gravel deposits and Quaternary alluvium and outwash unconformably overlie the rhyolites and cover the western part of the property.

Several episodes of structural deformation have affected the rocks of the Trinity area. A pre-Tertiary event strongly folded and faulted the rocks of the Auld Lang Syne Group and was accompanied by regional low-grade metamorphism. This event is thought to have been part of the Nevadan Orogeny. Tertiary deformation produced local north-trending open folds, high-angle faults, and local low-angle faults. Although northeast-trending, high-angle faults dominate the Trinity district, north-, northwest-, and west-northwest-trending, high-angle fault sets are well developed in the mine area and are important controls of mineralization (Ashleman, 1988). Still, the principal structural control on mineralization is the northeast-trending, northwest dipping normal fault running down the center of the open-pit.

All the rocks in the project area except the gravel and alluvial deposits locally exhibit various degrees of hydrothermal alteration and mineralization. The relatively permeable rhyolites and rhyolitic tuffs were most receptive to silver mineralization and associated silicification and quartz-adularia-sericite alteration, but alteration and mineralization also occur to a limited extent in adjacent argillite and tectonic breccia.

Silver mineralization was discovered in outcrop in altered Tertiary rhyolite at the break in slope between Triassic basement rock of the Trinity Range and pediment gravels that extend to the west. Northeast-trending normal faults lie along the break in slope and controlled mineralization.

Most of the mineralization is hosted by rhyolite tuffs, flows, volcanoclastic rocks, and intrusive rocks.  $^{40}\text{Ar}/^{39}\text{Ar}$  dating of sericite from highly altered rhyolite within the Trinity silver mine pit yielded an age of 26.829 Ma, but Appold and Muntean (1993) opined that a better upper limit for the age of mineralization is probably an age of 25.111 Ma from fresh sanidine phenocrysts within a relatively unaltered rhyolite porphyry that is a likely source of heat and/or metals for the deposit. They noted that this age inferred for Trinity is similar to the 24.7 to 26.4 Ma age inferred for the Majuba Hill porphyry system but is significantly older than the age of gold mineralization at Seven Troughs (14.1 Ma) across the valley to the west of Trinity.

A north-trending, gold-bearing, low-angle structure is present north of the Trinity mine (Santa Fe Pacific Mining, Inc., undated).

Figure 7-3 shows the geology in the vicinity of the old Trinity open-pit mine.

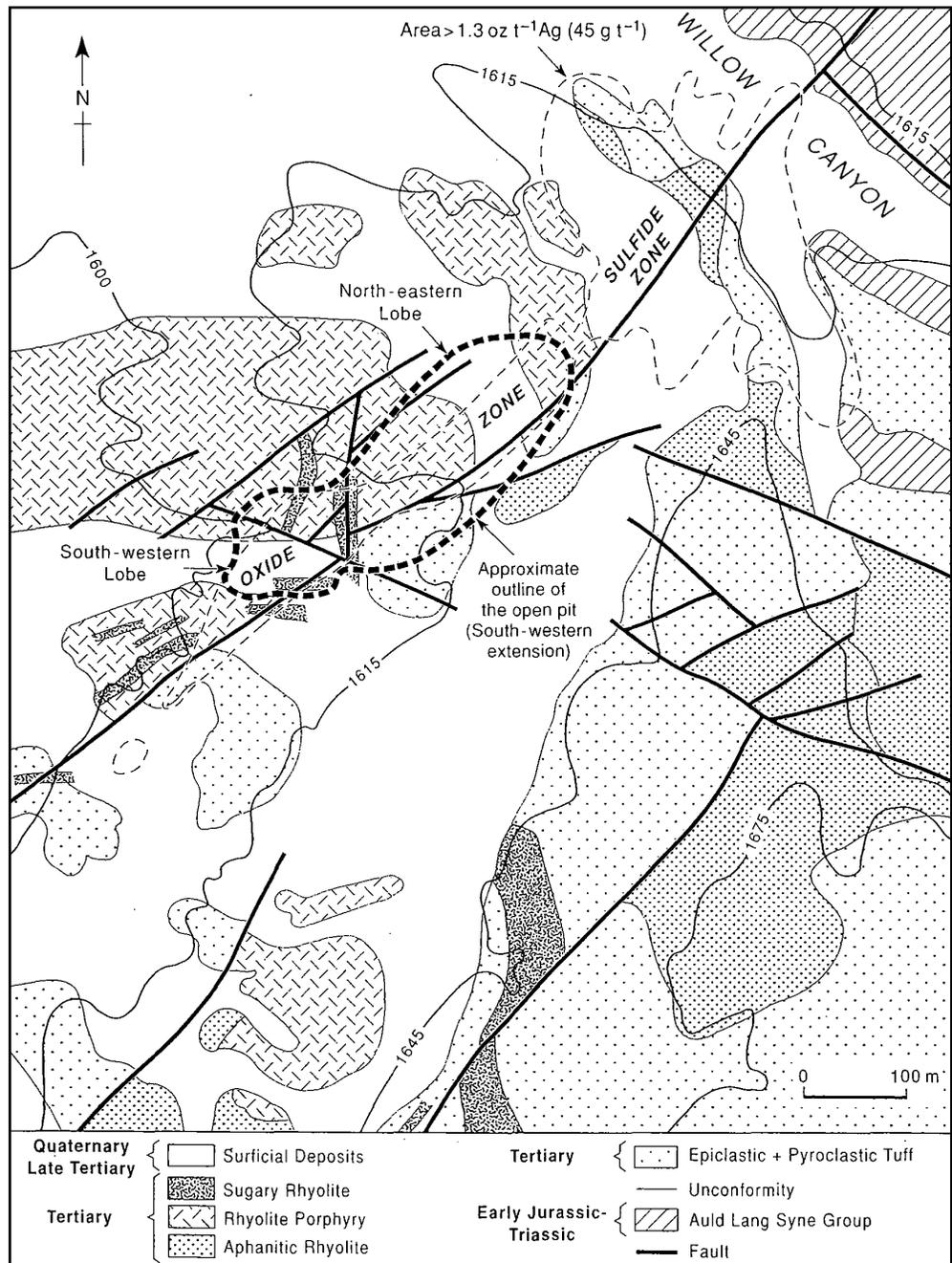


Figure 7-3. Geology of the Trinity Mine Area

(From Whateley et al., 2006)



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## 7.2 MINERALIZATION

The discovery location for the silver and base metal mineralization at the Trinity Project was an outcrop in altered Tertiary rhyolite at the break in slope between the Triassic phyllites and argillite that form the basement of the Trinity Range and pediment gravels that extend to the west. Mineralization lies within the hanging-wall block of a zone of normal faulting along this break in slope. The mineralization occurs primarily within rhyolite but is also hosted by Mesozoic argillites (Santa Fe Pacific Mining, Inc., undated). Although mineralization continues downward into the underlying Triassic rocks, it is more tightly constrained to fractures that host high-grade vein mineralization.

The original Trinity silver deposit can generally be divided into two parts: an oxide zone, a higher-grade portion of which was mined by Borax, and a low-grade sulfide zone that lies below and to the northeast of the oxide zone.

Alteration is widespread. Silicification, sericitization, and quartz-adularia-sericite alteration are the most common types. Propylitic (chlorite + calcite) alteration underlies and is peripheral to the other types. Kaolinite and illite are found along some late faults.

Silver, lead, and zinc mineralization occurs as oxidized and unoxidized sulfides within veinlets controlled by fractures and as disseminations within the host rocks, including breccia matrix. John and Muntean (2006) describe the form of the deposit as disseminated mineralization, including microfractures, stockwork zones, and breccia infill. High-grade zones in the oxide zone are spatially related to areas of strong jointing, sericitic alteration, and limonite (Ashleman, 1988, cited in Whateley et al., 2006). Sulfide mineralization consists mainly of pyrite, galena, sphalerite, marcasite, minor arsenopyrite, various silver minerals, and traces of gold, pyrrhotite, stannite, and chalcopyrite (Figure 7-4). The silver minerals are tetrahedrite-freibergite, pyrargyrite, minor argentite, and rare native silver. Freibergite and pyrargyrite occur as free particles as well as intergrowths in pyrite/marcasite and in sphalerite, respectively (Woods and Smith, 1984). Limited ore petrography indicated that the silver-bearing minerals are generally paragenetically late relative to the base metal sulfides.



Figure 7-4. Sulfide Mineralization at Trinity

Sulfide mineralization, in gray, in the bottom of the old Trinity open-pit

Muntean (1992) reports that detailed analysis of bench-level geology and blasthole silver assays, including calculation of grade-thickness plots, demonstrates that mineralization within the oxide pit is strongly controlled by a N45°E structure running the length of the pit and dipping 45-70° to the northwest; the structure extends beyond the oxide pit to the northeast and southwest. The >2 oz Ag/ton contour is about 150ft wide and runs parallel to the structure, entirely within the hanging-wall rhyolite porphyry, while grades in the footwall lithic tuff are <1 oz Ag/ton. Numerous northwest-trending structures do not abruptly cut off mineralization and appear to have been formed prior to mineralization (Muntean, 1992). Within the pit, the silver mineralization was about 100-200ft thick, but lead and arsenic mineralization was more pervasive, with lead mineralization increasing to the east away from the pit. Much of the lead and arsenic mineralization appeared to be concentrated in tuffs directly above the Mesozoic argillite basement.

Using grade-thickness plots of the data from Borax's drilling for silver, lead, zinc, copper, and their ratios, SFPM determined that the highest lead and zinc values are widely spread to the southeast (Muntean, 1992). Along the N45°E structure running through the pit, the highest lead and zinc grade-thicknesses appear to have an antithetical relationship to the silver grade-thicknesses. Copper grade-thicknesses are highest along a conspicuous northeast-trending ridge of silicified tuffs located southeast of the open-pit.

Ratios of the base metals and silver were used to search for zoning patterns (Muntean, 1992). The lead/silver and zinc/silver ratios show a strong zoning, increasing northwest to southeast. The lead/zinc ratio is relatively flat, except for a zinc-rich, northwest-trending structure (?) in the southeast part of the mine area. The lead/copper and copper/silver ratios suggest that the main center of the hydrothermal

system at Trinity may be below the ridge of silicified tuffs just southeast of the open-pit. Muntean (1992) thought that these zoning patterns suggest the possibility that there may have been early base metal mineralization centered largely on the ridge of silicified tuffs and later silver-rich, base metal-poor mineralization associated with the N45°E structure running the length of the pit.

Renaissance investigated the potential to add value to the project by considering lead and zinc, with lesser contributions from copper and tin; gold was not included because it had not been assayed during the 2006 and 2007 drilling by Renaissance and had not been consistently assayed in earlier drilling by Borax (Coolbaugh, 2009a). Coolbaugh (2009a) noted that gold was recovered from the leach pads in the ratio of 0.14oz gold per 1,000oz silver. Renaissance concluded that, depending on cutoff grades and metal recoveries, byproducts could increase the dollar value of material produced from the sulfide zones at Trinity by as much as 50% or more, with the added value predominantly coming from low-grade lead and zinc. The highest tin grade reported from the 2006 and 2007 drilling was 0.0486%, with the highest contribution to the value of individual 5ft intervals of as much as 20%; however, Coolbaugh (2009a) noted that analyzing for tin is difficult, and he recommended check analyses of selected intervals.

Silver-lead-zinc-copper zonation in the Trinity district is permissive of additional silver and base metal mineralization occurring east and south of the 2009 limit of drilling, and a plot of grade-thickness of equivalent silver grades suggests the deposit is open to the southeast, east, and potentially also to the northeast (Coolbaugh, 2009b).

In 1984, seven holes were drilled over outcropping Triassic rocks in Sections 27 and 3 that produced several intercepts of 0.015 to 0.005 oz Au/ton associated with a low-angle structure (Santa Fe Pacific Mining, Inc., undated). Although this mineralization also has an epithermal geochemical signature, it appears to be a different system from the silver-base metal signature of the Trinity mineralization, which suggests that there may be a separate Tertiary gold-bearing hydrothermal system in the area (Santa Fe Pacific Mining, Inc., undated).

## 8.0 DEPOSIT TYPES

The Trinity deposit has been variously categorized as a Tertiary volcanic-hosted, epithermal silver-base metal deposit (Ashleman, 1984, 1988; Santa Fe Pacific Mining, Inc., undated; AuEx website, 2010) and as a mesothermal silver-base metal deposit probably related to an underlying magmatic source expressed at the surface as rhyolitic dikes and sills (Leonard et al., 1986). Previous operators suggested the possibility that porphyry-copper-style mineralization occurs at depth. They based this hypothesis on the strength of illitic alteration and on a crude district-wide zonation that is characterized by a central copper-rich zone, an intermediate base metal-rich zone, and a peripheral silver-rich zone.

Silver-lead-antimony mineralization occurs about 12 miles east of the Trinity property in the Arabia mining district, which was discovered at about the same time as the Trinity district. The Arabia mineralization occurs in a series of parallel veins in Cretaceous granodiorite. The district was very active in the 1860s and 1870s, with sporadic mining until 1951. Production from the district is estimated to have been around 34,000 tons with a gross value of over \$1 million (Johnson, 1977).

The Rochester silver mine, one of the largest silver mines in the United States, lies in the Humboldt Range about 25 miles east of the Trinity property. The rich silver ores of the Rochester district were not discovered until 1912. Silver deposits occur in narrow fissure veins in Triassic rhyolite. The primary mineralization consisted of quartz, pyrite, sphalerite, galena, tetrahedrite, and chalcocopyrite, but the silver mineralization that was mined formed by supergene enrichment of the sulfide mineralization. The district produced over 8.8 million ounces of silver from 1912 to 1966 (Johnson, 1977), followed by production of over 127 million ounces of silver and 1.5 million ounces of gold since 1986 by Coeur d'Alene Mines Corp. ([www.coeur.com](http://www.coeur.com) as of February 2011).

The Trinity deposit is similar chemically and structurally to large silver deposits of Bolivia. These deposits occur as strings of deposits along controlling structures. Muntean (1992) noted that although vein mineralization is lacking at Trinity, many of the features seen in the Bolivian silver-tin deposits are observed at Trinity, such as explosion breccias, basal tuffs, intrusive rhyolite domes, mantle breccias, and minor mineralization hosted by basement rocks. Tin values range up to more than 100 ppm at Trinity. Pyrrhotite and stannite, which are observed at Trinity, are indicative of highly reducing conditions, similar to those of the Bolivian deposits. Like Trinity, the Bolivian deposits are noted for their paucity of gold.

Muntean (1992) observed that Trinity also resembles some of the epithermal silver-base metal bonanza veins deposits in Mexico. The Mexican deposits are noted for the occurrence of high-grade mineralization as bands of limited width and thickness but great length, with the bands often being sinusoidal in shape. Muntean (1992) thought that the mineralization at Trinity also appears to have a gross sinusoidal form.



**RESPEC**

## **9.0 EXPLORATION**

K9 has not conducted any exploration at the Trinity property.

## 10.0 DRILLING

This review is a compilation of the historical database. The author has not verified the validity of these data and has not performed any reinterpretations. This will be the focus of the recommended Phase 1 program. Therefore, the author has not included any cross sections of the drilling.

The project drilling database provided includes consists of historical rotary percussion, reverse circulation, and core drilling completed by Borax (including the joint-venture with SFPM for which Borax was the operator), SFPM, Renaissance, and Liberty Silver Corp. K9 has not conducted any drilling on the property.

Table 10-1 summarizes the historical holes in the project database. The author notes that the database contains holes outside the property boundary, and those holes have been excluded from the table.

Table 10-1. Trinity Database Summary

Hole Series	Number of holes	Total Footage	Period	Type	Operator
S-1 to S-209	199	74790	1982 - 1983	Vertical rotary percussion	Borax
SA-1 to SA-12	11	4715	1982 - 1983	Angled RC	Borax
SR-1 to SR-69	63	23225	1982 - 1983	Vertical RC	Borax
DTS-001 to DTS-011	8	8722	1992-1993	Angled RC	SFPM
TSD-001 to TSD-010	10	3712	2006	Core	Renaissance
TS07-011 to TS07-025	15	9355	2007	Angled RC	Renaissance
A12-1 to A12-20	20	22535	2012	Vertical RC	Liberty
<b>Total</b>	<b>326</b>	<b>147054</b>			

Figure 10-1 shows the drillhole locations for the holes drilled on the current Trinity property.

The mineralization at Trinity is controlled by the N45E-trending fault that dips to the northwest at 45 to 70 degrees and by the slightly (10 degree) northwest-dipping contact between the rhyolite sequence and underlying Triassic metasedimentary rocks. A total of 262 of the holes (86%) are vertical, and thus oriented approximately orthogonal to the shallow-dipping zones. In the vicinity of the pit, however, the vertical holes also intercepted the main structural zone at 20 to 45 degree angles and several angle holes were drilled close to down-dip. A list of significant intersections on the property has been tabulated in Appendix A.

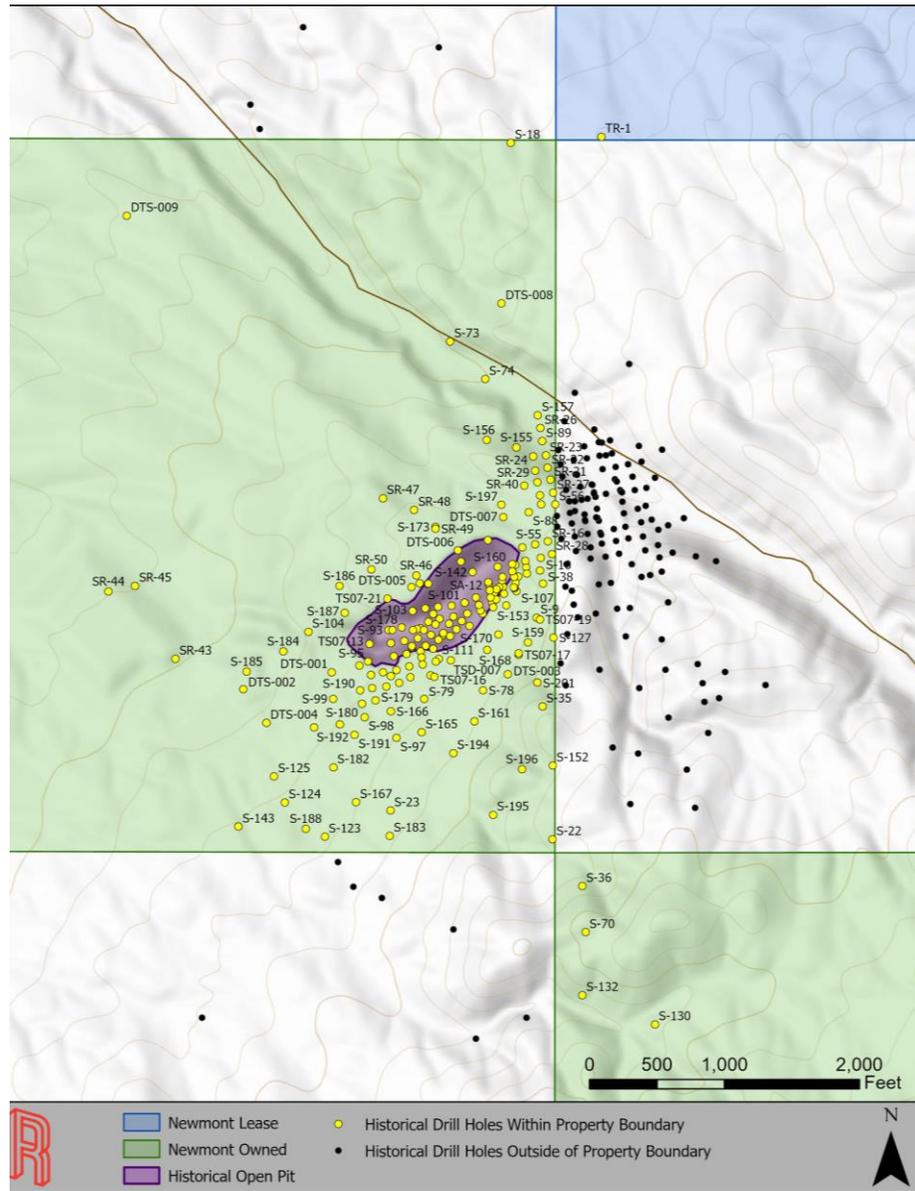


Figure 10-1. Drillhole Location Map for the Trinity Project

## 10.1 U.S. BORAX

Borax drilled conventional percussion holes as well as reverse circulation holes, with the latter used for holes that were expected to penetrate the water table (Ashleman, 1987). They also drilled diamond core holes, some of which were used to provide material for metallurgical test work. What information the author has found concerning drilling contractors, equipment used, and drilling details is provided below.

For their early drilling from 1982 through 1984, Borax used Eklund Drilling Company (“Eklund”) of Carlin, Nevada, as their drill contractor for conventional percussion and reverse circulation (“RC”) drilling (Ashleman, 1984, and information supplied by Liberty Silver). Holes S-1 through S-190, drilled in 1982 through 1985, were rotary down-hole hammer holes drilled with a TH-60 rig. Holes SA-1 through SA-8 were inclined RC holes drilled with a TR-60 rig; the same rig was used for RC holes SR-1 through SR-18,

drilled by Eklund in 1984. For core holes SC-1 through SC-5 drilled in 1984, Boyles Brothers of Salt Lake City was the drill contractor. Holes SA-9 through SA-12 and SR-19 through SR-37 were rotary down-hole hammer holes drilled by Eklund in 1985 with a TH-60 rig.

In 1986, Borax drilled rotary down-hole hammer holes S-191 through S-211 and SR-38 through SR-41 using Eklund with a TH-60 rig. For its 1986 core drilling (holes SC-6 through SC-10), Borax used Diamond Drill Contracting Company of Spokane, Washington. Ashleman (1987) reported that these core holes with a diameter of 2 9/16 inches, drilled to obtain representative rock samples for cyanide column leach tests. The author has no information on the type of rig used. Difficult drilling conditions were encountered, and the drillers were inexperienced with drilling in such conditions. Core recoveries were poor for the first two holes but improved on subsequent holes and were "*acceptable overall, providing high recoveries for enough of the mineralized zones to give sufficient representative samples for the metallurgical test program*" (Ashleman, 1987).

Leonard et al. (1986) reported that the rotary percussion holes were 5 1/4 inches in diameter, and the core holes were 2 9/16 inches in diameter.

Baele and Pelletier (1989) noted that some of the earlier percussion drill holes had poor sample recovery below the water table; they did not specify which holes were involved.

Borax drill samples for assaying were generally taken on five-foot intervals (Baele and Pelletier, 1989). Metallurgical core holes were sampled every 1 to 3ft (Whateley et al., 2006). Blastholes were sampled by "sample pan cut or manual cone cuts", each representing a 15ft composite (Whateley, et al., 2006); The author has no further details on this sampling methodology.

Ashleman (1989) discussed the reliability of the various types of drilling conducted by Borax, which included conventional percussion, RC, and core holes. He opined that the RC holes best represent the grade of the deposit. Although both conventional and RC holes adequately represented the grade of the rock above the water table, he noted that below the water table conventional rotary drilling poorly represented the grade of the rock due to dilution by erosion of rock from up hole, preferential loss of the heavier sulfide minerals to fractures in the rock, and loss of sulfides due to poor sampling procedures in the early conventional drill holes (e.g., S-3). He also noted that the core holes were drilled for metallurgical tests, not to test the grade of the deposit. He commented that they represent the smallest volume of rock, and because of variable core recovery in the areas of fracture-controlled mineralization, they give variable degrees of down-grading, which locally can be very significant.

Mr. Bickel has no further details on drilling programs or sampling by Borax.

## 10.2 SANTA FE PACIFIC MINING INC.

Santa Fe drilled RC holes TR-1 through TR-9 in 1984 with Eklund as the drill contractor using a TH-100 rig. RC holes TR-10 through TR-27 were drilled in 1985 by Becker Drilling, Inc. ("Becker") of Denver, Colorado. The log for TR-12 shows Becker as the contractor and indicates the rig was a Drill Systems RC rig. It does not appear that a hole numbered TR-23 was ever drilled.

RC holes TR 87-1 through TR 87-9 were drilled by Becker. There is no information on the logs about the type of rig used.

Mr. Bickel has found logs and/or assays but no details on the drilling in 1988 of RC holes TR 88-1 through TR 88-5 and TR 88-7 through TR 88-9 (RC); The author has no information on the drill contractor or types of rigs used. Holes TR 88-7 through TR 88-9 may be beyond the area the main area of mineralization.

Holes TS-1 through TS-5 were rotary down-hole hammer holes drilled in 1989 by Eklund; Mr. Bickel has no information on the type of rig used.

Logs for RC holes DTS1 through DTS11 drilled in 1991 and 1992 indicate that the drill contractor was Becker, but there is no information on the type of rigs used. Down-hole surveys were performed; logs for holes DTS2 through DTS6 indicate they were down-hole surveyed by Boyles-Welnav of Elko, Nevada, using a gyroscopic directional survey. Logs also indicate the drilling of hole DTS11 in 1992, but the contractor was Eklund.

Mr. Bickel has no information on the sampling methods and approach used by SFPM.

### 10.3 RENAISSANCE GOLD INC.

The following information came from new releases of Renaissance in 2006 and 2007 with other information, as cited.

For the ten core holes drilled in 2006 (TSD-001 to TSD-010), Renaissance used Kirkness Drilling Company, Inc. of Carson City, Nevada. Renaissance staff report (personal communication, 2010) that Kirkness used a skid-mounted rig that was probably a CS 14 Versadrill. All core recovered was HQ size (2.5-inch diameter). The core samples were photographed and logged on-site. Renaissance reported (August 3, 2006 news release) that the high-grade core intercepts encountered were thinner than had been suggested by earlier rotary drilling, as was expected.

For the 15 RC holes drilled in 2007 (TS07-11 to TS07-25), Renaissance used Layne Christensen Company of Chandler, Arizona. The rig was a Foremost Prospector W750 buggy rig drilling 5 ¼ to 5 ½-inch-diameter holes (Renaissance staff, personal communication, 2010).

All core recovered in Renaissance's 2006 drilling was HQ size. Renaissance reported (news release of August 3, 2006) that core loss occurred in several holes, especially in zones of mineralized rhyolite breccia. Sample intervals varied from 1 to 6ft and averaged 5ft in core length.

Sample lengths for RC drilling were 5ft. The rig had a rotating wet splitter mounted on a cyclone. Renaissance staff (personal communication, 2010) report that the RC holes were drilled wet except for setting surface casing.

## **10.4 LIBERTY SILVER CORP.**

Liberty Silver Corp. drilled 20 vertical reverse circulation holes in 2012 (A12-1 to A12-20) totaling 22,535 feet.

The company contracted Boart Longyear of West Valley City, Utah, to complete the drilling. Boart Longyear used a truck-mounted Foremost Explorer 1500 reverse circulation rig to drill 5.5-inch diameter bore holes which were sampled at 5-foot intervals. The cuttings were recovered through after being fed through a cyclone and rotating riffle splitter.

Collar locations were measured with a Trimble hand-held GPS unit connected to a backpack-mounted antenna.

## **10.5 SUMMARY OF DRILLING AND SAMPLING**

The Trinity database includes assay data from conventional rotary, RC, and core drill holes. In part due to the lack of documentation of drilling and sampling procedures, Mr. Bickel believes that the historic database is of sufficient quality for use in the scope of this report but future exploration should focus on verifying historical drill data beyond work done to date.

The preponderance of samples for all drill programs of all operators were taken at intervals of 10ft or less, which is significantly less than the thickness of the bulk-tonnage style of mineralization at Trinity. Each drill sample is therefore a small fraction of the true thickness of the mineralized zones.

## 11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

The following sections summarize the extent of Mr. Bickel's knowledge of sample preparation, analytical, and security procedures at Trinity.

### 11.1 U.S. BORAX

According to Baele and Pelletier (1989), Trinity samples taken by Borax during their drilling programs were assayed on an air-dried basis. Borax compiled the sample assay data into a computer drillhole database.

CMS, Inc. ("CMS"), a well-known lab based in Salt Lake City, Utah, was the initial assayer for Borax on the Trinity Project, assaying samples from holes S-1 through S-13 (Ashleman, 1984). Pulps from all the mineralized intercepts were sent to U. S. Borax Research Corp. ("USBRC") by CMS for check analyses, and composites of these intercepts were also sent to Union Assay Office, Inc. ("Union") of Salt Lake City for checks. The initial assays were for 10ft intervals; a sample was scooped from a 1/8 split of the rotary cuttings for two consecutive 5ft intervals and combined. Later, an assay sample was mechanically split with a Jones-type splitter out of a 1/8 split of the 5ft interval; these mechanically split 5ft assay samples were better, more representative samples, according to Ashleman (1984). CMS assayed for silver using atomic absorption spectrometry ("AA"); Mr. Bickel has no information on the sample digestion method. Union used fire assay methods for their analyses of silver. USBRC used AA (no information is available on the sample digestion method); when results exceeded 100 ppm Ag, the samples were analyzed by fire assaying with a gravimetric finish.

Most of the assays for the Trinity Project through 1987 were performed by USBRC (Reim et al., 1988). For holes S-25 through S-68, both CMS and Bondar-Clegg (since then purchased by ALS Chemex) performed check assays on samples initially assayed by USBRC (Ashleman, 1984). In addition, USBRC performed some internal assay checks. In a sample-check program for holes S-25 through S-30, CMS used AA. Bondar-Clegg used AA (Mr. Bickel has no information on the sample digestion method), but for samples assaying >50 ppm Ag, fire assay was used. USBRC again used AA, with fire assay when results exceeded 100 ppm Ag.

Apparently Hunter Mining Laboratory, Inc. ("Hunter") of Sparks, Nevada, performed check assaying for USBRC in 1985 on their 1985 TR-series holes; an invoice from Hunter included assay certificates.

In contrast to the AA analyses used for exploration drillhole samples, blasthole samples were analyzed at the mine site by cyanide leach (Whateley, et al., 2006). Comparisons between USBRC and on-site blasthole assays indicate an approximate 15% difference between the two methods, with the on-site assays yielding the lower values (Reim et al., 1988). Reim et al. (1988) noted that the relative amount of refractory (non-cyanide extractable) silver varies significantly throughout the deposit. There are also rapid lateral and vertical changes in silver grade. They also noted that the difference between assays appeared to be greatest at the lower grades (1 to 2oz Ag/ton range), which has implications for cutoff grade considerations.

Certification of assay laboratories is a comparatively recent development, and Borax's drilling at Trinity took place before certification was widely used as an industry standard. Both Union and Hunter are no longer in business, and the author has no information on whether they were registered or certified. Bondar-Clegg was ISO 9002 registered in 1998, well after their work for Borax at Trinity; Bondar-Clegg is now part of ALS Chemex. Mr. Bickel has no information on whether CMS or USBRC were certified or registered, but assumes they were not. Union, Hunter, and Bondar-Clegg were all well-known commercial analytical laboratories that used industry standard sample preparation and analytical techniques.

Mr. Bickel has no information on security procedures used by Borax. Except for check assaying described in Section 12.1, and no information is available on QA/QC procedures that Borax may have used.

## 11.2 SANTA FE PACIFIC MINING INC.

The author has obtained information about analyses on samples taken by SFPM from copies of assay certificates accompanying drill logs. These records for the TR-series holes drilled by SFPM in 1984 and 1985 indicate that assaying was performed by Chemex Labs Ltd. ("Chemex") of North Vancouver, BC. Samples were analyzed for silver, gold, and, in some cases lead, arsenic, zinc, and antimony. For holes TR-1 through TR-22, the assay certificate shows "NAA" after Au, which is presumed to mean the analysis was by neutron activation; the lower detection limit for gold was 1 ppb. A hand-written note on the drill logs for holes TR-10 through TR-12, TR-14 through TR-16, TR-18 through TR-21, and TR-26 through TR-27 suggests that the analyses, presumably for all metals analyzed, were by "fire with NAA finish." The Chemex assay certificates for holes TR-26 and TR-27 do not show "NAA" after Au, but instead indicate gold was determined by fire assaying with an AA finish. The silver assays show only "aqua R.", which likely indicates these analyses used *aqua regia* digestion with determination by AA.

Copies of assay certificates accompanying drill logs for holes TR-87-1, TR-87-2, TR-87-7, TR-87-8, and TR-87-9, drilled in 1987, indicate assaying was performed by Chemex in Sparks, NV; logs for the remaining four holes of the nine-hole series do not include assay certificates. Three of the certificates indicate gold was analyzed by 10g fire assay with AA finish, with a detection limit of 5 ppb. Silver was analyzed with HNO<sub>3</sub> – *aqua regia* digestion and AA, with a detection limit of 0.1 ppm.

Copies of assay certificates accompanying drill logs for holes 88-1 through 88-9 drilled in 1988 indicate analysis by both Barringer Laboratories Inc. ("Barringer") of Sparks, Nevada and by Chemex. It appears that Chemex was used for check assays on holes 88-1 and 88-2, but for holes 88-7 through 88-9 it is not evident who was the primary assayer. Barringer analyzed silver, arsenic, antimony, and lead by AA (Mr. Bickel has no information on the sample digestion method). Gold was analyzed by fire assay. Mercury was analyzed by hydride generation. The detection limits for silver and gold appear to have been 0.1 ppm and 0.005oz Au/ton, respectively. At Chemex, lead and arsenic were assayed by HNO<sub>3</sub> – *aqua regia* digestion and AA with a detection limit of 1 ppm and an over-limit of 10,000 ppm. Mercury and antimony were assayed by two-acid digestion and AA.

Copies of assay certificates accompanying drill logs for holes TS-1 through TS-5 drilled in 1989 indicate that the principal assay lab was Chemex. Samples were analyzed for gold, silver, copper, lead,

zinc, arsenic, mercury, antimony, and bismuth. Gold was analyzed using 10g charges that were fire assayed with AA finishes; the detection limit was 5 ppb, with an over-limit of 10,000 ppb. Silver was analyzed by HNO<sub>3</sub> – *aqua regia* digestion and AA; the minimum detection limit was 0.2 ppm, and the upper detection limit was 100 ppm. The other metals were analyzed by nitric or two-acid *aqua regia* digestion and AA.

Copies of assay certificates accompanying drill logs for holes DTS-001 through DTS-011 drilled in 1991 and 1992 indicate that assaying was performed by Chemex. Silver was analyzed by HNO<sub>3</sub> – *aqua regia* digestion and AA; the detection limit was 0.2 ppm, with an over-limit of 100 ppm. Pb, Zn, Cu, As, and other metals were analyzed using nitric, two-acid, or three-acid *aqua regia* digestions and AA. Gold was analyzed using a 30g fire assay with AA finish. The detection limit for gold was 5 ppb, and the over-limit was 10,000 ppb.

Although certification of assay laboratories is a comparatively recent development and SFPM's drilling at Trinity took place before certification was widely used as an industry standard, Chemex currently holds ISO 9001:2008 and 17025:2005 certifications. Barringer had no known certification, but was a widely recognized commercial lab that is now part of Inspectorate America Corp.

Mr. Bickel has no information on security procedures or QA/QC procedures used by SFPM, except for reference to check assays mentioned above.

### 11.3 RENAISSANCE GOLD INC.

The following information on Renaissance sample analysis and security was taken from Renaissance news releases with additional information provided by Renaissance staff (personal communication, 2010).

For their 2006 core drilling program, samples were shipped to American Assay Laboratories, Inc. ("AAL") in Sparks, Nevada, where the core was split in half for assaying. Renaissance also used AAL for their 2007 RC drilling program. Silver assays were performed using inductively coupled plasma ("ICP") analytical techniques with a four-acid digestion of sample pulps in 2006 and two-acid sample digestion in 2007; assays were also determined for a broad suite of trace elements. For the 2007 drilling, samples exceeding 100 ppm silver were re-assayed using a 30 g sample fire assay with a gravimetric finish. Blanks and standards were included for quality control. In addition for the RC drill program, Renaissance collected two to three sets of duplicates per hole at the splitter. The blanks consisted of coarse rhyolite. The two standards were from MEG Labs: MEG-Cu-1 containing about 25 ppm Ag and MEG-Ag-1 containing about 250 ppm Ag. Control samples including standards and blanks comprised about 8 to 10% of the samples per hole or about three control samples per 100 feet, alternating between blanks and standards (Hartley et al., 2011).

AAL is a widely used and accredited assayer in the mining industry.

AAL picked up the core and RC samples on-site under the supervision of the rig geologist. AAL cut the core at the lab after logging and marking of sample intervals by the rig geologist in the field.



RESPEC

## 11.4 SUMMARY STATEMENT

Mr. Bickel knows very little of the sample handling and sample security measures employed on any of the drilling programs undertaken at Trinity other than what was summarized in the previous sections. It should be noted that all the companies who conducted exploration at Trinity were reputable, well-known mining/exploration companies that likely followed the accepted industry standards relating to sampling preparation methods, analytical techniques, and sample security protocols.

Mr. Bickel has concerns about the quality of the AA analyses by USBRC at the pre-mining stage. These concerns are discussed in Section 12.3, and recommendations to address these concerns are addressed in Section 26.0.

## 12.0 DATA VERIFICATION

Data verification, as defined in NI 43-101, is the process of confirming that data have been generated with proper procedures, have been accurately transcribed from the original sources, and is suitable for use. Additional confirmation of the drill data's reliability is based on the author's evaluations of the Trinity Project QA/QC procedures and results and his experience working with the data.

### 12.1 SITE VISIT

Mr. Bickel visited the Trinity Project on October 16, 2025. His site visit included an inspection of mineralized geologic exposures in the pit, examination of drillhole collar locations, a review of the surface geology and infrastructure at the property, and a visit to the core storage facility in Sparks, Nevada to examine drill core and review the available inventory of exploration samples. In addition, he reviewed and verified geologic logs and cross-sections from historical reports during the site visit and engaged in geologic discussions and interpretations with K9 staff. As a result of his site visit and communications, the author has no significant concerns with the project data.

### 12.2 HISTORICAL DATA VERIFICATION STUDIES

As described in Section 11.1, CMS was the primary assayer for Borax at the Trinity Project for the first two months of 1983 (holes S-1 through S-13). The CMS analytical method was atomic absorption but there is no information on the digestion method. Neither analytical method nor digestion technique is identified in the USBRC reports. Pulps from mineralized intercepts in holes S-1 through S-13 sent to USBRC and composites of these intercepts were sent to Union for check analyses. According to Ashleman (1984), in general, CMS silver results were 10 to 20% lower than USBRC results, and Union's results were 1 to 5% lower than USBRC. An analysis of results on lead and zinc by CMS and USBRC for holes S-1, S-3, and S-4 indicated that CMS's lead results were 2 to 7% lower for the three holes, while CMS zinc results were 0.55% and 7% higher in two holes and 2% lower in the third hole (Ashleman, 1984).

USBRC was the primary lab from March 1983 through 1984 (holes S-14 through S-209, SA-1 through SA-12, and SR-1 through SR-69), and checks between USBRC and CMS showed better overall agreement in this time period (Ashleman, 1984). There also was good agreement between Union and USBRC and Bondar-Clegg and USBRC (Ashleman, 1984). A second sample-check program was conducted for holes S-25 through S-30. Nine pulps were sent to Bondar-Clegg, and 44 pulps were sent to CMS. The Bondar-Clegg checks were 9.8% lower than USBRC's initial results, but it is important to note that this observation is questionable because of the small number of samples. The CMS checks were 3.79% higher than USBRC's results, although CMS's results were generally lower for higher silver values (>1oz Ag/ton) and higher on the lower values (especially <0.5oz Ag/ton) (Ashleman, 1984).

In a third sample-check program, 21 composites were made from six 5ft intervals for each of 21 holes between S-8 and S-68 (Ashleman, 1984). These 21 composites were first assayed by USBRC and, as an internal check for consistency, were resubmitted to USBRC a second time using different sample numbers. It is not known if these were identical pulps, different splits or rejects that were resubmitted. In addition, the 21 samples were sent to Bondar-Clegg for silver assaying. Again, the sample type is not

known. Ashleman (1984) reported that there was good internal consistency for silver analyses, with the first set of composite assays being 2.3% lower than the original 5ft assays and the rerun of the composites 0.45% lower than the first set of assays. The Bondar-Clegg check assays were 1.79% lower than USBRC's first set of composite assays and 4.05% lower than the original 5ft assays by USBRC.

Ashleman (1984) concluded that USBRC silver values are "*reasonably consistent and reliable*" compared against CMS originals, CMS checks, and Bondar-Clegg checks.

### 12.3 RESPEC DATA VERIFICATION

Mr. Bickel compiled all the available historic check analyses from the various drill programs for review. There are no definitive records that indicate the type of sample used for check assaying, except for the 2006-2007 Renaissance series.

#### 12.3.1 CMS VS. USBRC CHECKS ON SILVER - 1983

There are 96 CMS vs. USBRC duplicates. All samples were from rotary percussion S-series holes from 1982 and 1983. As mentioned in Section 11.1, CMS was the primary lab at the start of the project, and then the primary lab became USBRC, with CMS completing check analyses. The original and check analyses were by AA except for samples above 100ppm Ag which were fire assayed by USBRC. The digestion method for the AA analyses is not known. The exact type of sample submitted as checks is not known.

Figure 12-1 is a graph that shows the difference, plotted on the y-axis, of each check analysis relative to the original analysis. The x-axis of the graph plots the means of the paired data, with each pair consisting of an original analysis and the corresponding check analysis. The red line is a moving average and provides a visual guide to the trend of the relative differences.

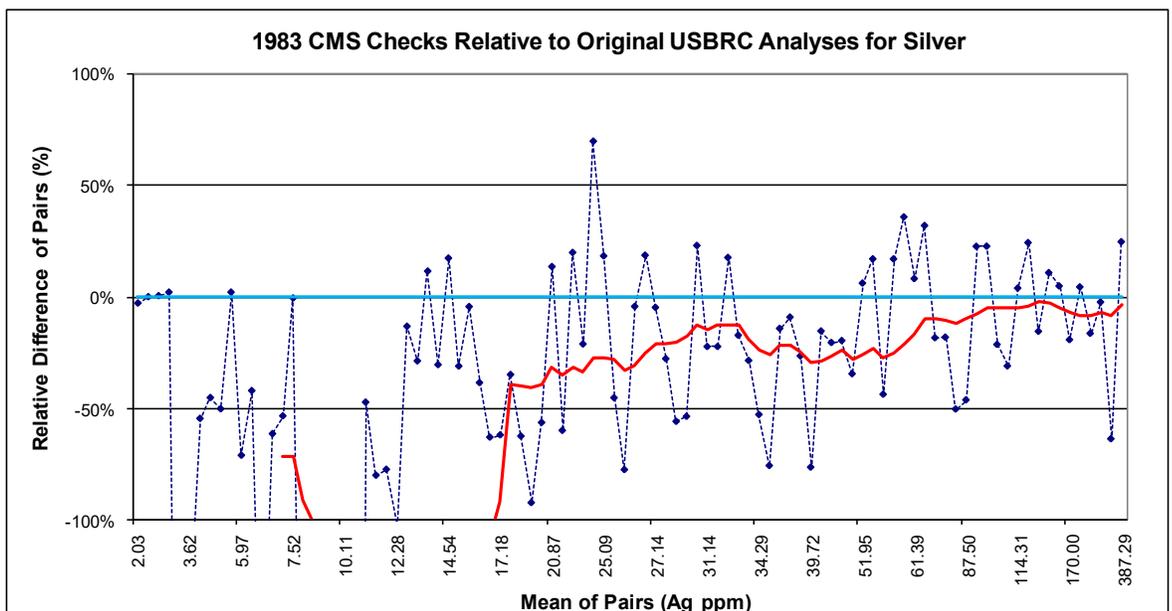


Figure 12-1. CMS Checks Relative to Original USBRC Analyses for Silver – 1983

Descriptive statistics of the paired data are summarized in 12-1.

If 11 outliers from the lower-grade analyses are removed, the average difference in values drops by a few percentage points, but the CMS values remain consistently lower, even at mean silver values greater than 35 ppm. In the 0.5 to 2 oz Ag/ton (17 to 68 ppm Ag) range, the CMS values are approximately 25% lower than the USBRC values.

Table 12-1. CMS Analyses vs. USBRC Analyses for Silver – 1983

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	96	96	96		96	96
Mean	49.037	51.729	46.344	-10%	-54%	64%
Median	25.779	26.500	24.650			
Std. Dev.	67.712	69.527	67.486			
CV	1.381	1.344	1.456			
Min.	2.029	2.057	1.300	-37%	-826%	0%
Max.	387.286	392.000	430.000	10%	70%	826%
<b>Mean <math>\geq</math>10.0 ppm Ag</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	76	76	76		76	76
Mean	60.458	63.366	57.549	-9%	-35%	46%
Median	32.054	34.800	28.800			
Std. Dev.	71.915	73.893	71.816			
CV	1.190	1.166	1.248			
Min.	10.107	12.686	2.000	-84%	-826%	2%
Max.	387.286	392.000	430.000	10%	70%	826%
<b>Mean <math>\geq</math>35 ppm Ag</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	35	35	35		35	35
Mean	106.532	110.120	102.943	-7%	-12%	25%
Median	79.107	85.714	70.400			
Std. Dev.	85.403	88.335	85.809			
CV	0.802	0.802	0.834			
Min.	35.200	39.086	25.500	-35%	-77%	2%
Max.	387.286	392.000	430.000	10%	36%	77%

"A.V. Rel. Diff." is the average absolute value of the difference between the original analysis and the check analysis.

"CV" is the Coefficient of Variation.

### 12.3.2 BONDAR-CLEGG VS. USBRC CHECKS ON SILVER - 1984

A total of 20 pulps of composited 5- and 10-foot intervals were analyzed by USBRC and were sent to Bondar-Clegg as checks in 1984. In addition, eight pulps from 5-foot samples that were analyzed by USBRC were also sent to Bondar-Clegg. All of these samples were from the rotary percussion S-series holes. The original and check analyses were by AA except for samples above 100ppm Ag which were fire assayed by USBRC. The digestion method for the AA analyses is not known. The exact type of sample submitted as checks is not known. Figure 12-2 is the graph that shows the difference of each check analysis relative to the original analysis.

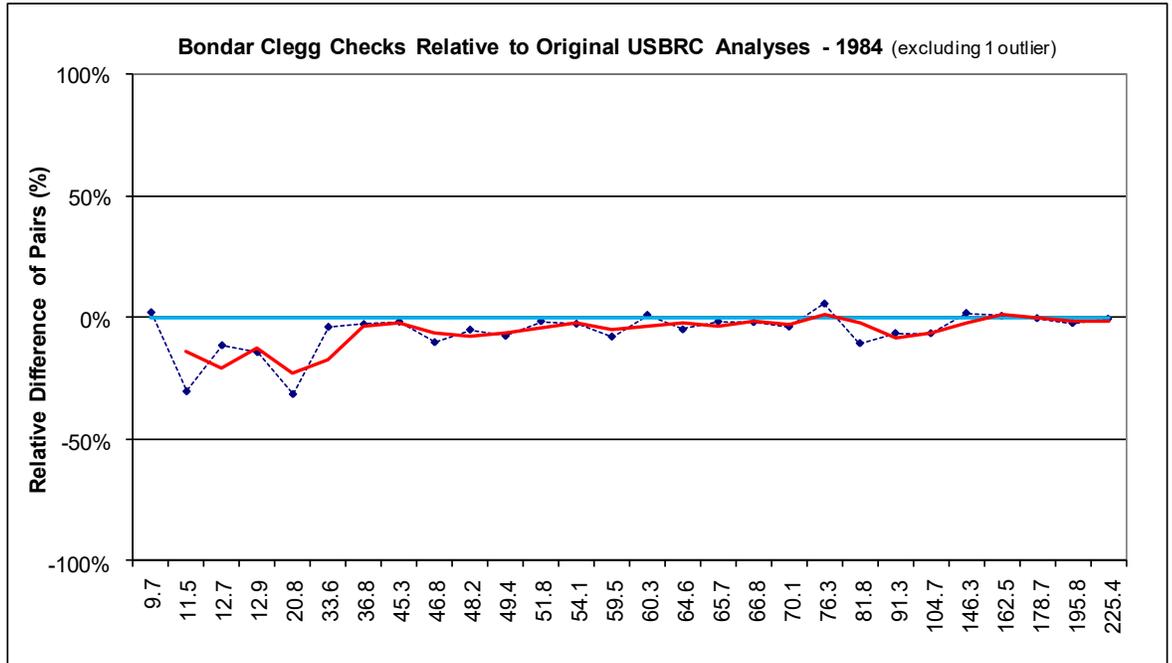


Figure 12-2. Bondar-Clegg Checks Relative to Original USBRC Analyses for Silver – 1984

Descriptive statistics of the paired data are summarized in Table 12-2.

Table 12-2. Bondar-Clegg Analyses vs. USBRC Analyses for Silver – 1984

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	28	28	28		28	28
Mean	74.394	75.438	73.350	-3%	-6%	6%
Median	59.875	60.850	58.900			
Std. Dev.	57.343	57.246	57.472			
CV	0.771	0.759	0.784			
Min.	9.700	9.600	9.800	2%	-31%	0%
Max.	225.350	226.000	224.700	-1%	6%	31%

In the 0.5 to 2 oz Ag/ton (17 to 68 ppm Ag) range, the Bondar-Clegg values are approximately 4% lower than the USBRC values.

### 12.3.3 USBRC VS. USBRC CHECKS ON SILVER – 1984-1985

A total of 311 samples originally analyzed by the USBRC were resubmitted to the USBRC for analyses. It is not known if these are pulps from the original analyses, new splits from the original pulverized material, or new pulps generated from the crushed reject material. All of these checks were from the rotary percussion S-series holes drilled in 1982 and 1983. The checks and original analyses were AA

but the digestion method is not known. Figure 12-3 is a graph that shows the difference of each check analysis relative to the original analysis.

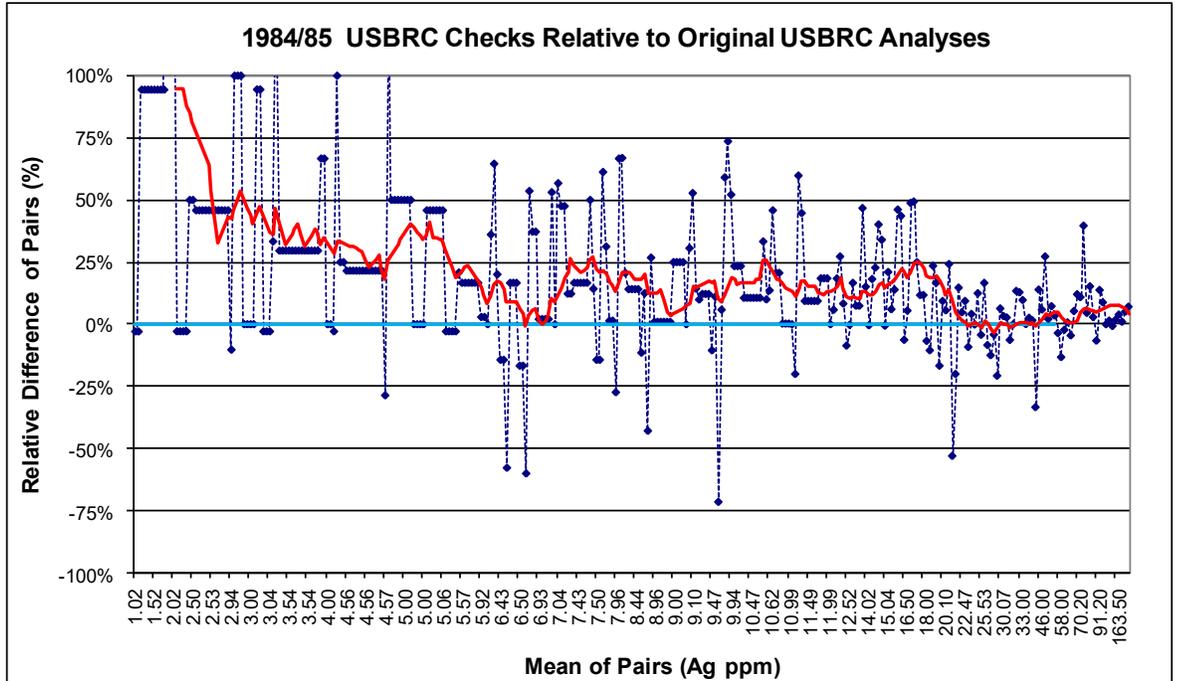


Figure 12-3. USBRC Checks Relative to Original USBRC Analyses for Silver – 1984/85

Descriptive statistics of the paired data are summarized in Table 12-3.

The check analyses are 20% higher between 7 and 22 ppm Ag.

At values greater than 35 ppm Ag the check analyses are 4% higher than the original analyses but the dataset is too small to draw conclusions.

Table 12-3. USBRC Checks vs. USBRC Original Analyses for Silver – 1984/85

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	311	311	311		311	311
Mean	17.826	17.144	18.508	8%	21%	18%
Median	8.443	7.886	9.000			
Std. Dev.	31.234	30.738	31.789			
CV	1.752	1.793	1.718			
Min.	1.015	1.000	1.000	0%	-71%	0%
Max.	234.000	226.000	242.000	7%	200%	200%
<b>Mean <math>\geq</math>10.0 ppm Ag</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	121	121	121		121	121
Mean	37.147	36.162	38.132	5%	9%	13%
Median	19.500	19.200	20.000			
Std. Dev.	43.525	42.826	44.319			
CV	1.172	1.184	1.162			
Min.	10.472	8.571	10.000	17%	-53%	0%
Max.	234.000	226.000	242.000	7%	60%	60%
<b>Mean <math>\geq</math>35.0 ppm Ag</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	33	33	33		33	33
Mean	90.152	88.309	91.994	4%	4%	8%
Median	70.200	67.400	74.200			
Std. Dev.	54.751	53.738	55.963			
CV	0.607	0.609	0.608			
Min.	34.986	34.971	33.000	-6%	-33%	0%
Max.	234.000	226.000	242.000	7%	40%	40%

#### 12.3.4 HUNTER VS. USBRC PULP CHECKS ON SILVER – 1985

A total of 98 duplicate pulps from original USBRC analyses in 1985 were sent to Hunter for checking. A billing invoice states that Hunter received pulps, and no sample preparation was charged. All of these pulps were from the rotary percussion S-series holes. The original and check analyses were by AA except for samples above 100ppm Ag which were fire assayed by USBRC. The digestion method for the AA analyses is not known. Figure 12-4 is a graph that shows the difference, of each check analysis relative to the original analysis. Eight extreme outliers were removed from this data set.

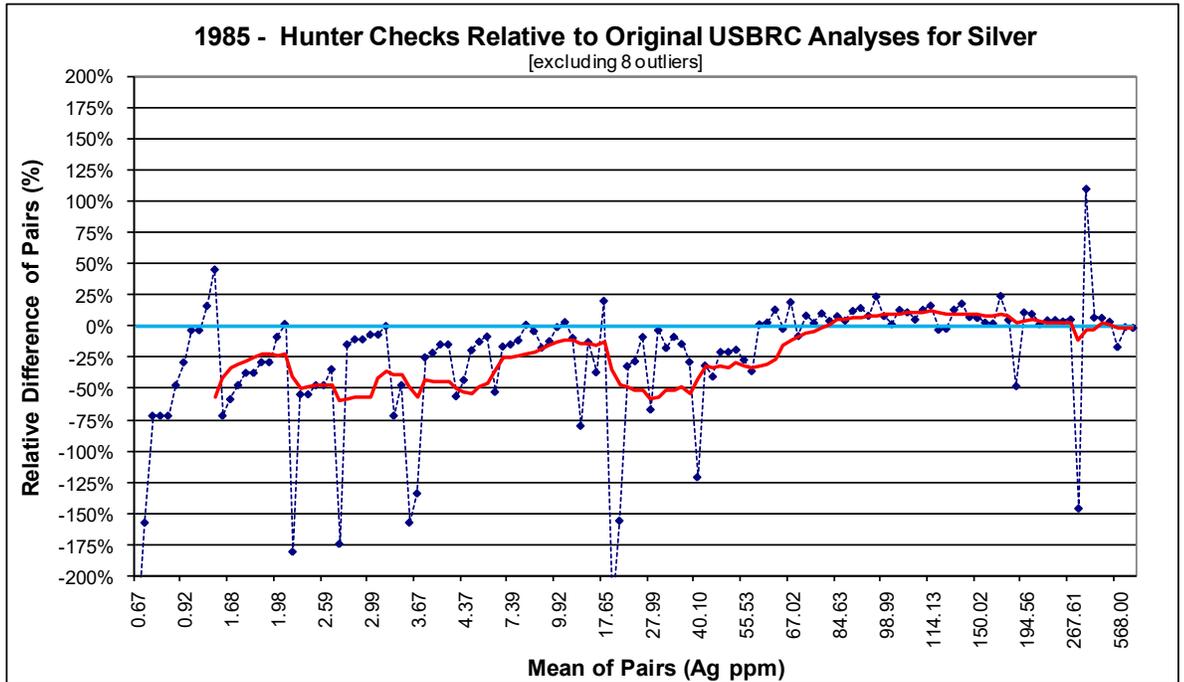


Figure 12-4. Hunter Checks Relative to Original USBRC Analyses for Silver – 1985

For mean values less than 55 ppm Ag, the Hunter analyses are between 15% and 55% lower than the USBRC analyses. For values greater than 55 ppm Ag, the Hunter analyses are generally a few percent higher than the USBRC values.

Descriptive statistics of the paired data are summarized in Table 12-4.

There is a substantial difference between Hunter silver analyses and the USBRC analyses in the 0.5 to 2 oz Ag/ton (17 to 68 ppm Ag) range, with the Hunter analyses 30% lower in this interval.

Table 12-4. Hunter Checks vs. USBRC Original Analyses for Silver – 1985

(8 outliers removed)

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	129	129	129		129	129
Mean	86.051	86.046	86.057	0%	-24%	33%
Median	25.057	30.171	21.000			
Std. Dev.	195.523	196.527	195.820			
CV	2.272	2.284	2.275			
Min.	0.665	1.029	0.300	-71%	-243%	0%
Max.	1910.000	1920.000	1900.000	-1%	110%	243%
<b>Mean <math>\geq</math>10.0 ppm Ag</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	74	74	74		74	74
Mean	147.448	147.149	147.747	0%	-11%	24%
Median	89.272	85.500	93.000			
Std. Dev.	240.967	242.571	241.212			
CV	1.634	1.648	1.633			
Min.	10.122	9.943	8.400	-16%	-229%	0%
Max.	1910.000	1920.000	1900.000	-1%	110%	229%
<b>Mean <math>\geq</math>35.0 ppm Ag</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	62	62	62		62	62
Mean	172.391	171.395	173.387	1%	-2%	17%
Median	102.286	98.572	109.000			
Std. Dev.	256.088	258.298	255.935			
CV	1.486	1.507	1.476			
Min.	35.500	36.000	25.000	-31%	-146%	0%
Max.	1910.000	1920.000	1900.000	-1%	110%	146%

### 12.3.5 AAL VS. AAL PULP CHECKS ON SILVER – 2006-2007

AAL ran checks on pulps of 173 samples that AAL also analyzed. These samples are from the holes drilled by Renaissance during 2006 and 2007. All of the check sample analyses were by ICP and the analyses were reported on the same analytical reports. AAL used a four-acid digestion of sample pulps in 2006 and two-acid sample digestion in 2007. Figure 12-5 is the graph of the difference.

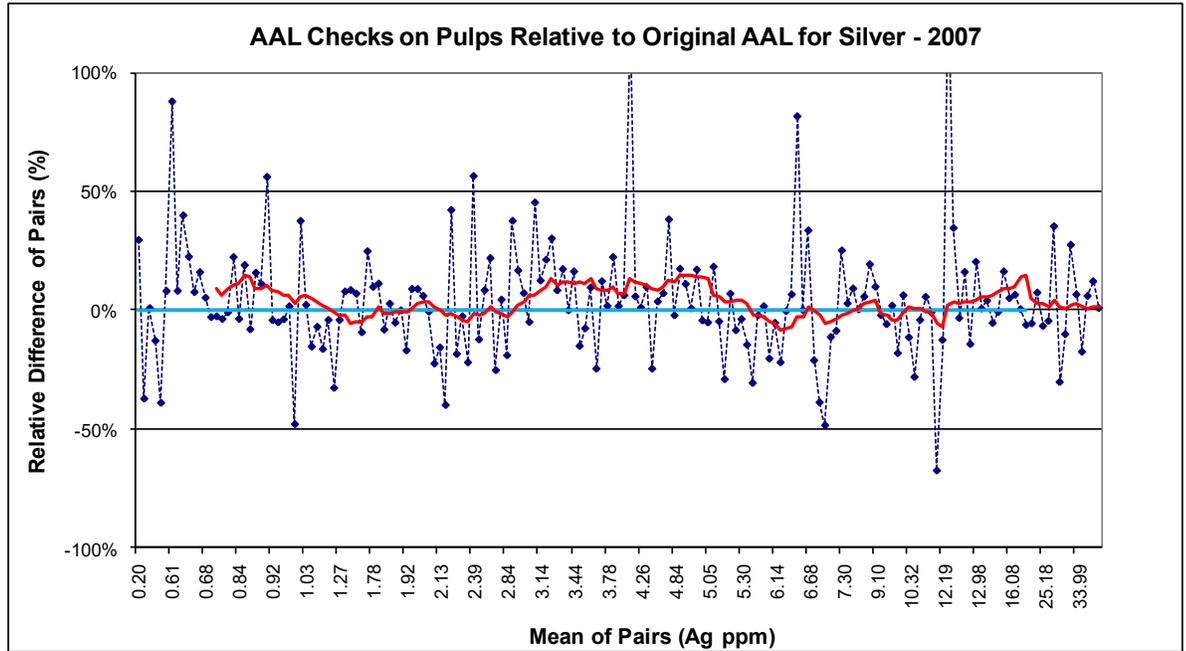


Figure 12-5. AAL Checks on Pulps from RC and Core Samples Relative to Original AAL Analyses for Silver – 2007

Descriptive statistics of the paired data for the 173 samples are summarized in Table 12-5.

Table 12-5. AAL Checks on Pulps from RC and Core Samples vs. AAL Analyses for Silver – 2007

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	173	173	173		173	173
Mean	7.768	7.689	7.846	2%	4%	16%
Median	3.890	3.794	3.988			
Std. Dev.	12.848	12.747	13.022			
CV	1.654	1.658	1.660			
Min.	0.199	0.173	0.224	30%	-67%	0%
Max.	110.422	109.900	110.944	1%	143%	143%
Mean >10.0 ppm Ag	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	35	35	35		35	35
Mean	24.698	24.441	24.954	2%	3%	16%
Median	15.889	15.679	17.089			
Std. Dev.	20.967	20.822	21.307			
CV	0.849	0.852	0.854			
Min.	10.318	7.243	8.884	23%	-67%	0%
Max.	110.422	109.900	110.944	1%	143%	143%

If six outliers are removed, the mean difference for all samples drops from 2% to 1%. Overall, the checks do not reflect any consistent bias, although the variability is surprisingly high for duplicate ICP silver analyses of pulps by the same lab on the same assay certificate. The variation could be explained by a nugget effect in the silver distribution. The mean of most of these checks is below 12 ppm Ag.

### 12.3.6 CMS VS. USBRC ANALYSES FOR LEAD – 1983

There were 52 duplicate samples submitted to CMS for lead analyses to compare against original USBRC analyses. The samples are from the 1982 and 1983 S-series holes. The original and check analyses were by AA. The digestion method for the AA analyses and the type of check samples submitted are not known. Figure 12-6 is the graph of the differences. One extreme outlier was removed in this plot.

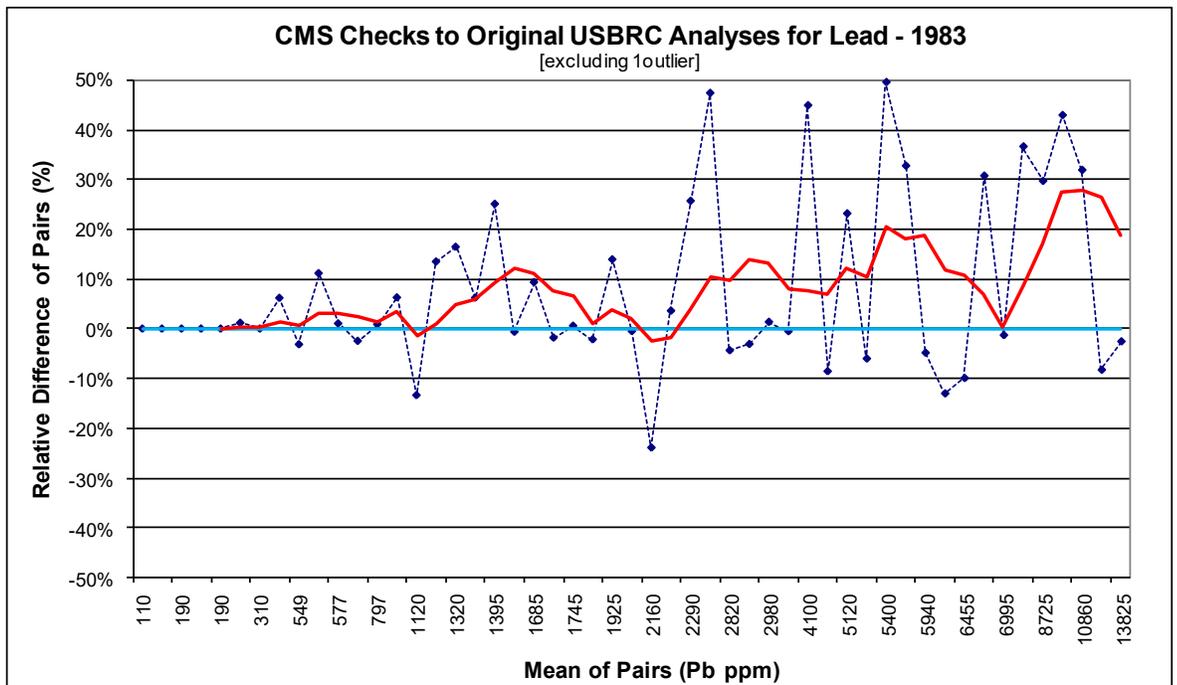


Figure 12-6. CMS Checks Relative to Original USBRC Analyses for Lead – 1983

Descriptive statistics of the paired data for the 51 samples are summarized in Table 12-6.

Table 12-6. CMS Checks on Rotary Percussion Samples Relative to Original USBRC Analyses for Lead – 1983

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	51	51	51		51	51
Mean	3337.04	3168.00	3506.08	11%	8%	12%
Median	2015.00	2020.00	2010.00			
Std. Dev.	3293.52	3135.32	3504.95			
CV	0.99	0.99	1.00			
Min.	110.00	110.00	110.00	0%	-24%	0%
Max.	13825.00	14000.00	13650.00	-3%	49%	49%

Although the data set is small, the CMS values are 10% higher than the USBRC values. The divergence becomes even more pronounced at higher lead values.

### 12.3.7 USBRC VS. USBRC CHECKS ON LEAD – 1984/85

There were 372 check samples submitted to USBRC for lead analyses to compare against original USBRC analyses. The type of check sample submitted is not known. All of the samples are from 1982 through 1983 S-series holes. The original and check analyses were by AA. The digestion method for the AA analyses is not known. Figure 12-7 is a graph that shows the difference.

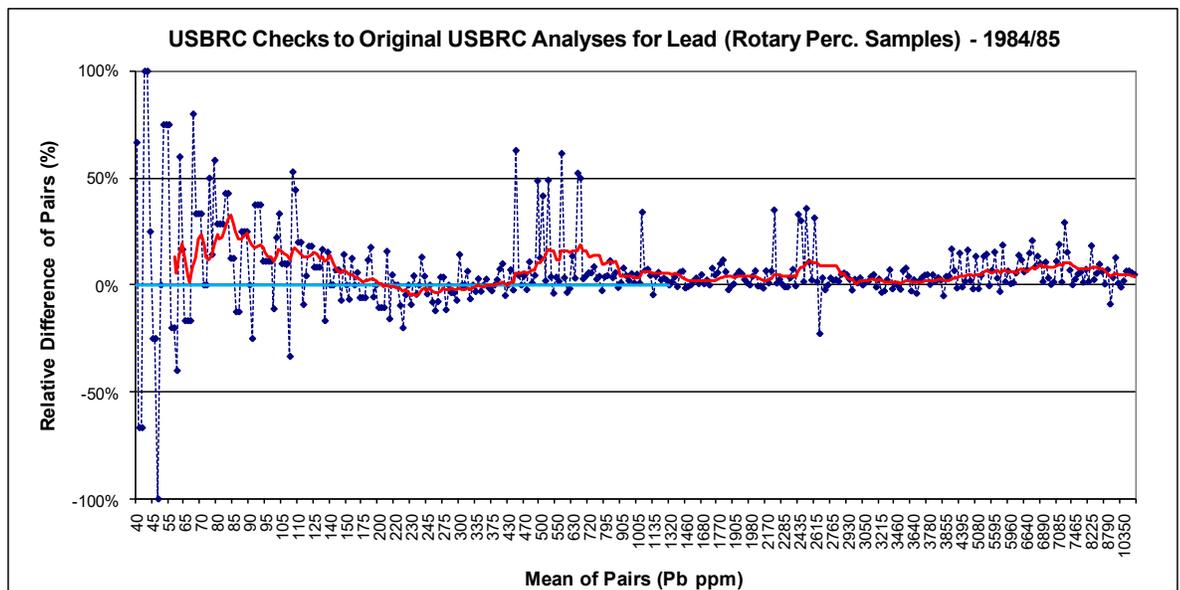


Figure 12-7. USBRC Checks Relative to Original USBRC Analyses for Lead – 1984/85

Descriptive statistics of the paired data for the 372 samples are summarized in Table 12-7.

Table 12-7. USBRC Checks from Rotary Percussion Samples Relative to Original USBRC Analyses for Lead – 1984/85

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	372	372	372		372	372
Mean	2245.37	2187.24	2303.49	5%	7%	11%
Median	1000.00	965.00	1010.00			
Std. Dev.	2747.76	2673.17	2826.53			
CV	1.22	1.22	1.23			
Min.	40.00	30.00	30.00	0%	-100%	0%
Max.	14550.00	14200.00	14900.00	5%	100%	100%
<b>Mean <math>\geq</math>1000 ppm Pb</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	186	186	186		186	186
Mean	4200.38	4092.90	4307.85	5%	5%	6%
Median	3312.50	3300.00	3310.00			
Std. Dev.	2719.29	2640.09	2806.72			
CV	0.65	0.65	0.65			
Min.	1005.00	880.00	1020.00	16%	-23%	0%
Max.	14550.00	14200.00	14900.00	5%	36%	36%
<b>Mean <math>\geq</math>5000 ppm Pb</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	62	62	62		62	62
Mean	7481.69	7253.23	7710.16	6%	7%	7%
Median	7050.00	6780.00	7300.00			
Std. Dev.	2033.61	2020.72	2070.38			
CV	0.27	0.28	0.27			
Min.	5040.00	4760.00	5030.00	6%	-9%	0%
Max.	14550.00	14200.00	14900.00	5%	29%	29%

The checks are consistently 5% to 6% higher than the original assays at the same lab, especially within the 0.1 to 1.0% lead range. There is no indication that different analytical techniques were used.

### 12.3.8 HUNTER VS. USBRC ANALYSES FOR LEAD - 1985

There were 118 duplicate pulps submitted to Hunter for lead analyses to compare against original USBRC analyses. The samples are from rotary percussion S- and reverse circulation SA- and SR-series holes drilled in 1982 and 1983. The original and check analyses were by AA. The digestion method for the AA analyses is not known. Figure 12-8 is a graph of the relative difference.

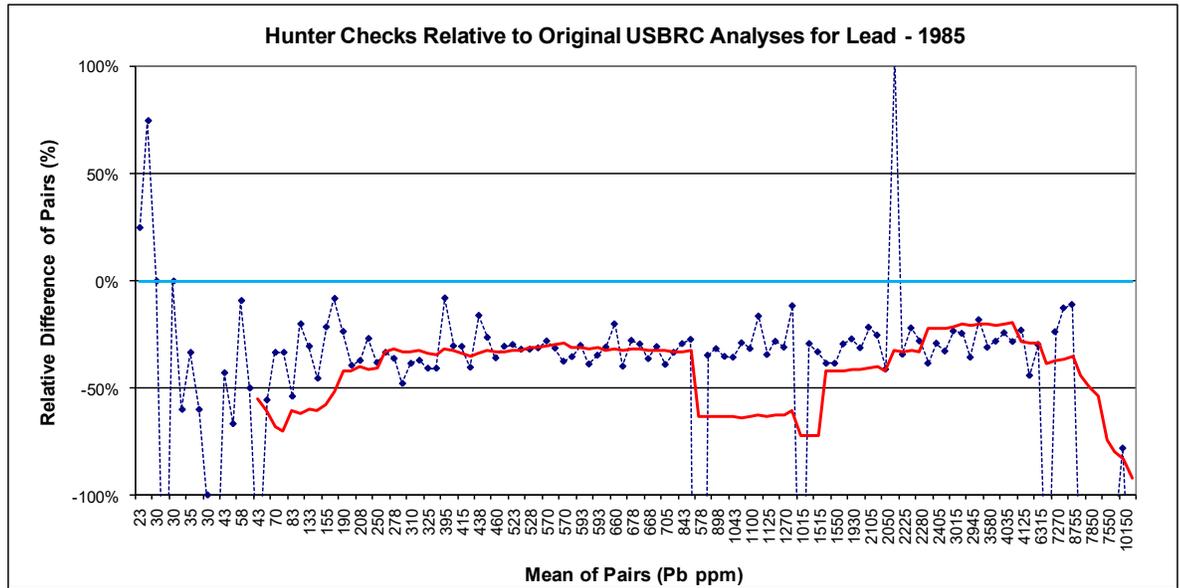


Figure 12-8. Hunter Checks on Pulps Relative to Original USBRC Analyses for Lead – 1985

Descriptive statistics of the paired data for the 118 samples are summarized in Table 12-8.

Table 12-8. Hunter Checks on Pulps Relative to Original USBRC Analyses for Lead – 1985

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	118	118	118		118	118
Mean	1768.347	2142.161	1394.534	-35%	-46%	50%
Median	663.750	765.000	550.000			
Std. Dev.	2596.792	3445.517	1872.989			
CV	1.468	1.608	1.343			
Min.	20.00	20.00	10.000	-50%	-500%	0%
Max.	15550.00	22500.00	8600.000	-62%	106%	500%
<b>Mean ≥ 1000 ppm Pb</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	49	49	49		49	49
Mean	3758.112	4582.653	2933.571	-36%	-49%	54%
Median	2405.000	2630.000	2100.000			
Std. Dev.	3069.800	4289.844	2083.299			
CV	0.817	0.936	0.710			
Min.	1015.00	1190.00	500.000	-58%	-321%	11%
Max.	15550.00	22500.00	8600.000	-62%	106%	321%

Hunter lead analyses are consistently 30% lower than the USBRC analyses.

### 12.3.9 AAL VS. AAL PULP CHECKS ON LEAD FROM RC SAMPLES 2006/07

The AAL set of checks includes 173 duplicate pulp analyses from the 2006/2007 Renaissance drilling and were reported on the same analytical reports. Lead analyses were completed using inductively coupled plasma ("ICP") analytical techniques with a four-acid digestion of sample pulps in 2006 and two-acid sample digestion in 2007. Figure 12-9 is a graph that shows the difference.

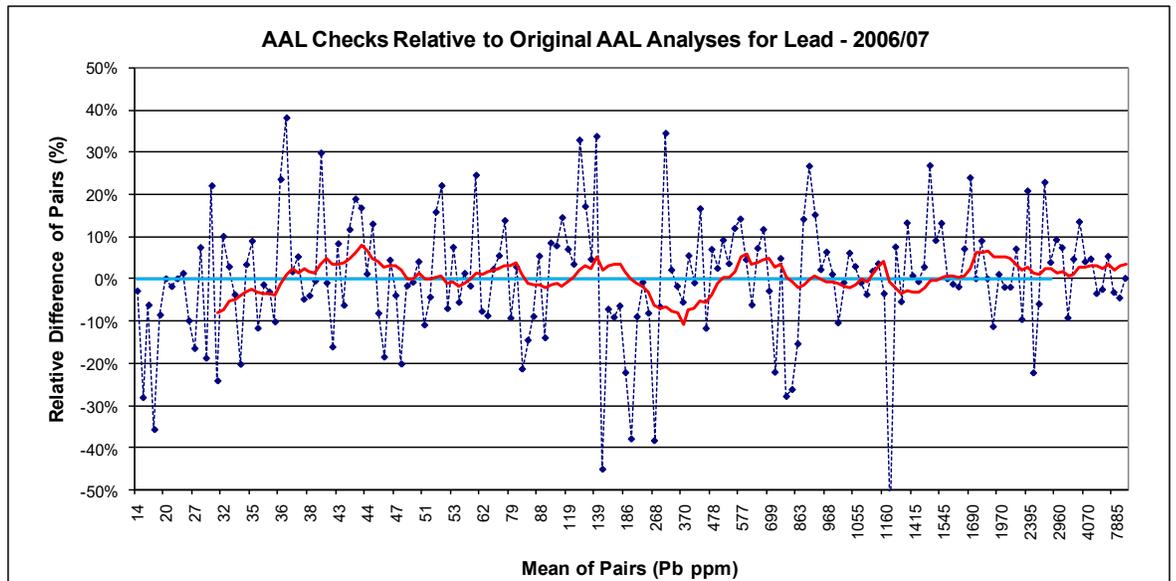


Figure 12-9. AAL Checks on Pulps Relative to Original AAL Analyses for Lead – 2006/07

Descriptive statistics of the paired data for the 173 samples are summarized Table 12-9.

The results for these checks are similar for the silver checks at AAL. There is variation in results, but no consistent divergence.

Table 12-9. AAL Checks on Pulps from RC and Core Samples Relative to Original AAL Analyses for Lead – 2006/07

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	173	173	173		173	173
Mean	890.645	885.560	895.731	1%	0%	10%
Median	195.000	219.000	172.000			
Std. Dev.	1480.493	1480.636	1482.967			
CV	1.662	1.672	1.656			
Min.	14.000	14.200	13.800	-3%	-57%	0%
Max.	9195.000	9190.000	9200.000	0%	38%	57%
<b>Mean <math>\geq</math> 1000 ppm Pb</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	49	49	49		49	49
Mean	2576.357	2558.571	2594.143	1%	2%	8%
Median	1940.000	1940.000	1940.000			
Std. Dev.	1896.197	1912.248	1886.517			
CV	0.736	0.747	0.727			
Min.	1020.000	990.000	943.000	-5%	-57%	0%
Max.	9195.000	9190.000	9200.000	0%	27%	57%

### 12.3.10 USBRC VS. USBRC CHECKS ON ZINC - 1985

There were 29 check samples submitted to USBRC for zinc analyses to compare against original USBRC analyses. The type of check sample submitted is not known. All of the samples are from 1982 through 1983 S-series holes. The original and check analyses were by AA. The digestion method for the AA analyses is not known. Figure 12-10 is a graph that shows the difference of each check analysis relative to the original analysis.

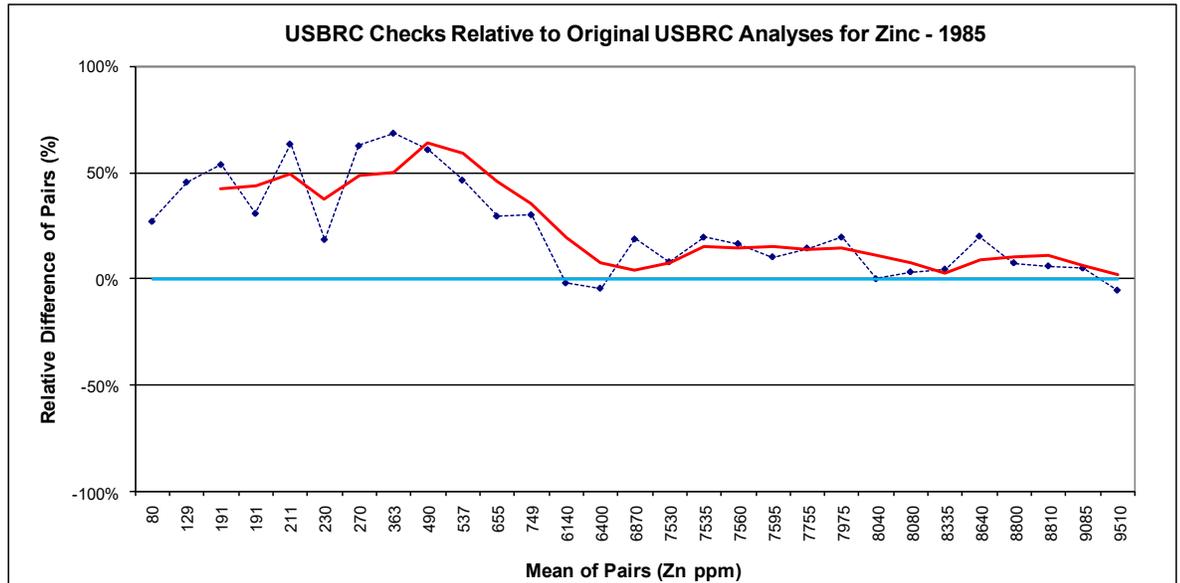


Figure 12-10. USBRC Checks from Rotary Percussion Samples Relative to Original USBRC Analyses for Zinc – 1985

Descriptive statistics of the paired data for the 29 samples are summarized in Table 12-10.

Table 12-10. USBRC Checks from Rotary Percussion Samples Relative to Original USBRC Analyses for Zinc – 1985

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	29	29	29		29	29
Mean	4784.534	4579.828	4989.241	9%	24%	24%
Median	6870.000	6540.000	7460.000			
Std. Dev.	3863.968	3749.635	3992.737			
CV	0.808	0.819	0.800			
Min.	79.500	70.000	89.000	27%	-5%	0%
Max.	9510.000	9760.000	9430.000	-3%	69%	69%

The USBRC checks are on zinc are 5% to 15% higher than originals in the 0.6% to 0.95% zinc range.

### 12.3.11 HUNTER VS. USBRC PULP CHECKS ON ZINC - 1985

There were 60 duplicate pulps submitted to Hunter for zinc analyses to compare against original USBRC analyses. The samples are from rotary percussion S- and from reverse circulation SA- and SR-series holes (drilled in 1983). Figure 12-11 is a graph that shows the difference.

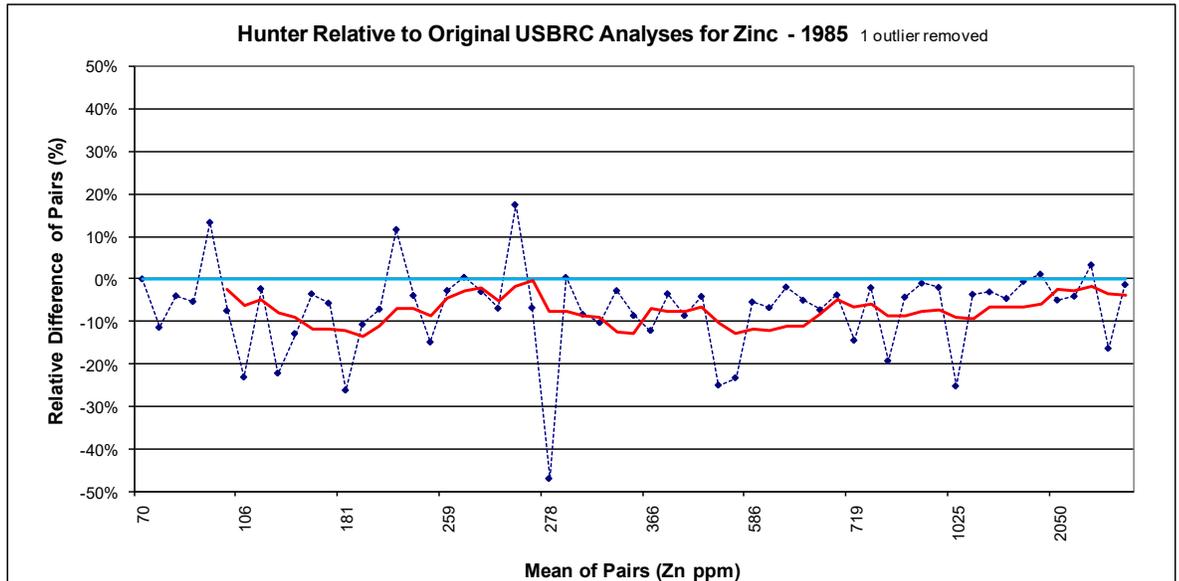


Figure 12-11. Hunter Checks on Pulps Relative to Original USBRC Analyses for Zinc – 1985  
(1 outlier removed)

Descriptive statistics of the paired data for the 59 (one outlier removed) samples are summarized in Table 12-11.

Table 12-11. Hunter Checks on Pulps Relative to Original USBRC Analyses for Zinc – 1985

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	59	59	59		59	59
Mean	729.280	749.831	708.729	-5%	-7%	9%
Median	349.500	364.000	335.000			
Std. Dev.	996.327	1017.095	977.061			
CV	1.366	1.356	1.379			
Min.	70.000	70.000	70.000	0%	-47%	0%
Max.	6040.000	6080.000	6000.000	-1%	18%	47%
<b>Mean ≥ 1000 ppm Zn</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	12	12	12		12	12
Mean	2165.833	2214.167	2117.500	-4%	-5%	6%
Median	1747.500	1745.000	1750.000			
Std. Dev.	1464.870	1494.836	1439.458			
CV	0.676	0.675	0.680			
Min.	1010.000	1020.000	910.000	-11%	-25%	1%
Max.	6040.000	6080.000	6000.000	-1%	3%	25%

These checks samples are at the low end of zinc values but are consistently lower than the USBRC analyses by 5% to 10%.

### 12.3.12 CMS VS. USBRC CHECKS ON ZINC - 1985

There were 52 duplicate pulps submitted to CMS for zinc analyses to compare against original USBRC analyses. The original and check analyses were by AA. The digestion method for the AA analyses is not known. The type of check sample submitted is not known. The samples are from rotary percussion S-series holes drilled in 1983. Figure 12-12 is a graph that shows the difference.

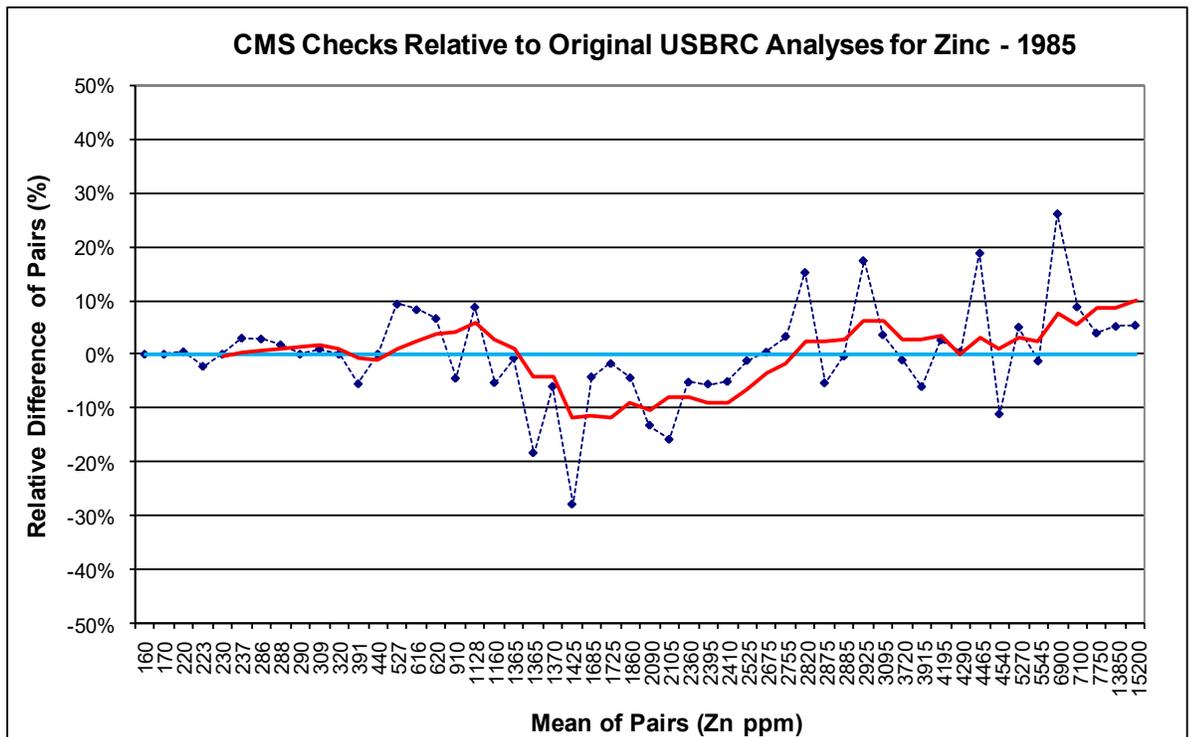


Figure 12-12. CMS Checks Relative to Original USBRC Analyses for Zinc – 1985

Descriptive statistics of the paired data for the 52 samples are summarized in Table 12-12.

The CMS checks are lower by 8% in the 0.13% to 0.27% Zn range and then trend upward and cross over the mean above 0.28% zinc.

Table 12-12. CMS Checks Relative to Original USBRC Analyses for Zinc – 1985

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	52	52	52		52	52
Mean	2691.740	2657.423	2726.058	3%	0%	6%
Median	1975.000	2060.000	1885.000			
Std. Dev.	3069.692	2973.946	3171.374			
CV	1.140	1.119	1.163			
Min.	160.000	160.000	160.000	0%	-28%	0%
Max.	15200.000	14800.000	15600.000	5%	26%	28%
Mean $\geq$ 1000 ppm Zn	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	35	35	35		35	35
Mean	3821.071	3771.429	3870.714	3%	0%	8%
Median	2820.000	2690.000	2800.000			
Std. Dev.	3178.028	3056.184	3307.838			
CV	0.832	0.810	0.855			
Min.	1127.500	1080.000	1130.000	5%	-28%	0%
Max.	15200.000	14800.000	15600.000	5%	26%	28%

### 12.3.13 AAL VS. AAL PULP CHECKS ON ZINC FROM RC SAMPLES

The AAL set of checks includes 173 duplicate pulp analyses from the 2006/2007 Renaissance drilling and were reported on the same analytical reports. Zinc analyses were completed using inductively coupled plasma (“ICP”) analytical techniques with a four-acid digestion of sample pulps in 2006 and two-acid sample digestion in 2007. Figure 12-13 is a graph that shows the difference.

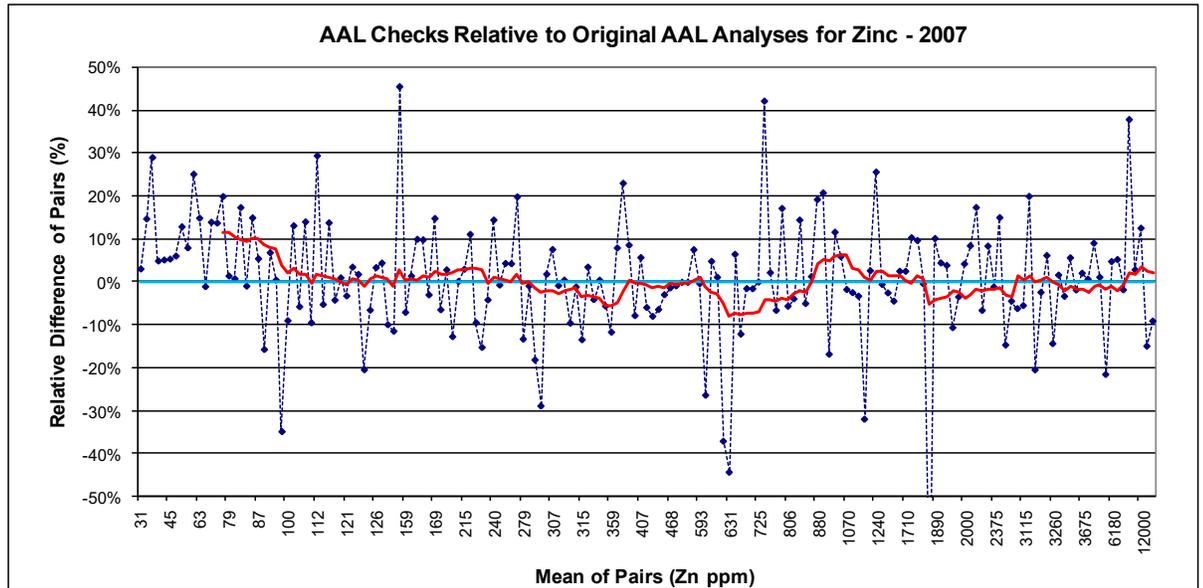


Figure 12-13. AAL Checks on Pulp Relative to Original AAL Analyses for Zinc – 2007

Descriptive statistics of the paired data for the 173 samples are summarized in Table 12-13.

Table 12-13. AAL Checks on Pulps from RC and Core Samples Relative to Original AAL Analyses for Zinc – 2006

All Pairs	Mean	Original	Duplicate	Diff.	Rel. Diff.	A.V. Rel. Diff.
Count	173	173	173		173	173
Mean	1344.862	1341.971	1347.754	0%	1%	10%
Median	408.000	421.000	407.000			
Std. Dev.	2380.521	2373.744	2401.872			
CV	1.770	1.769	1.782			
Min.	31.450	31.000	31.900	3%	-81%	0%
Max.	15900.000	16600.000	15200.000	-8%	45%	81%
<b>Mean &gt;1000 ppm Zn</b>	<b>Mean</b>	<b>Original</b>	<b>Duplicate</b>	<b>Diff.</b>	<b>Rel. Diff.</b>	<b>A.V. Rel. Diff.</b>
Count	55	55	55		55	55
Mean	3567.364	3557.273	3577.455	1%	0%	9%
Median	2375.000	2370.000	2370.000			
Std. Dev.	3246.390	3237.702	3288.579			
CV	0.910	0.910	0.919			
Min.	1015.000	960.000	1030.000	7%	-81%	1%
Max.	15900.000	16600.000	15200.000	-8%	38%	81%

The results for these checks are similar for the silver and lead checks at AAL. There is variation in results, but no consistent bias.

## 12.4 SUMMARY STATEMENT ON DATA VERIFICATION

The author is concerned about the accuracy of the USBRC analyses for silver, lead, and zinc. The USBRC analyses represent a significant portion of the analyses in the database and the USBRC silver, lead, and zinc values are consistently higher when compared to CMS, Hunter and Bondar-Clegg checks (the only exception is the CMS lead analyses which are 10% higher than the USBRC analyses). This is especially true for the 0.5 to 2.0 oz/ton range for silver, and the 0.02 to 1.0% range for lead and zinc.

Because the 0.5 to 2.0 oz Ag/ton range is critical for this deposit, it must be determined if the USBRC analyses are reliable for silver, lead and zinc. Since the pulps are no longer available, twinning of some of the holes from the early Borax drilling is required. Twelve drillhole locations have been identified outside, but in the vicinity of, the existing open-pit that would twin USBRC-analyzed holes and might provide a statistically viable data set. Additional holes may be required after the first phase of twinning.

## 12.5 ASSAY DATABASE AUDIT

The majority of the 1982 through 1992 drillhole analytical data and collar information was entered into a digital database by Renaissance prior to 2007. Approximately 10% of the silver values were checked against original lab reports and entries were accurate. However, several discrepancies for the lead and zinc values required that all lead and zinc values be checked against the lab reports. The information from the 2006 and 2007 drilling was in digital form and was imported into the project database.

## 13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

### 13.1 HISTORICAL TESTING AND MINERAL PROCESSING

In discussing possible processing costs and processes prior to mining at Trinity, Ganderup (1986) provided the following discussion of projected recoveries:

*"Oxide leach extractions are estimated to be 93% [for silver] based on an average of more than 40 intervals from ten holes. Surface oxide results which yielded lower extractions were excluded because they represent only a very minor portion of the reserves.*

*Flotation recovery of silver from sulfide ore is estimated to be 92% at a weight concentration ratio of 10:1. Leach extraction from the sulfide concentrate following a 48 hour atmospheric pre-aeration at 85°C is optimistically estimated to be 87%. However, this leach extraction requires a potentially difficult solid-liquid separation between the pre-aeration and leach stages and there are some indications of minor silver losses to the pre-aeration liquor.*

*It is estimated that an additional silver loss of roughly 1% can be expected in the 4-stage CCD [countercurrent decantation] circuit. Therefore the overall silver recovery is estimated to be 92% for oxide ore and 79% for sulfide ore."*

Whateley, et al. (2006) provided the following summary of mineral processing:

- / The oxide mineralization is amenable to direct leaching resulting in a 94 to 97% silver recovery.
- / Oxide material recoveries of silver by flotation were low at 50 to 60%.
- / Sulfide mineralization required fine grinding to give a 78 to 84% silver recovery; cyanide consumption was high.
- / Flotation tests on sulfide material liberated 90 to 95% of the silver and 90% of the lead and zinc, providing that the pH values of the collectors, which suppress Fe and As, were high.
- / Following production of the oxide deposit, it was estimated that 75% of the silver in the oxide material was recovered by heap leaching.

The following subsections describe specific metallurgical testing programs undertaken in 1983 to 1987, prior to production from the Trinity mine.

#### 13.1.1 1983 U. S. BORAX RESEARCH CORP. AND HAZEN RESEARCH INC.

In March-April 1983, USBRC conducted preliminary metallurgical testing on three predominantly sulfide samples and one oxidized sample of rotary drill cuttings, including cyanide leach and froth flotation testing (Smith, 1983). Silver recovery following grinding and rougher/scavenger froth flotation procedures was about 90%, with a maximum of 93% and a minimum of 72%. Cyanide leaching resulted in silver recovery of about 50%, with a high of 65% and a low of 24%; reagent consumption was high.

Later in 1983, Hazen Research, Inc. ("Hazen") reviewed the prior test results from USBRC and proceeded with preliminary testing and a mineralogical examination of the Trinity silver mineralization

(Gathje, 1983). Hazen's test work used a sample composite prepared by Borax from rotary drill holes S-8 and S-13. The following is a summary of their conclusions:

- / Bulk sulfide flotation achieved silver recoveries of 90 to 95% into a bulk concentrate assaying about 20 oz silver/ton and containing about 15-20% by weight of the feed. Corresponding lead and zinc recoveries ranged from the high 80s to low 90s percent. A typical cleaner concentrate contained 77% of the contained arsenic and assayed 1.90% arsenic.
- / Selective sulfide flotation achieved a silver recovery of 84.6% into a single-stage cleaner concentrate assaying 156.8 oz silver/ton and containing 2.07% by weight of the feed. This concentrate, assaying 0.60% arsenic and containing approximately 8% of the arsenic in the feed, demonstrated the possibility of producing concentrates low in arsenic.
- / Cyanide leach tests, using 2 to 10g NaCN/liter and grinds from 77 to 92% passing 270 mesh, demonstrated slow kinetics for silver dissolution. In all cases, the data show the need for leach times of at least 72 to 96 hours. At 96 hours, the cyanidation of whole sample gave silver dissolutions of 78 to 84%. The cyanidation of a bulk flotation concentrate, after regrinding, achieved a silver dissolution of 82.4% after 96 hours. Cyanide consumptions were high, presumably due to interactions between the sulfides and cyanide.

### 13.1.2 1983-1984 U. S. BORAX RESEARCH CORP.

Following the preliminary metallurgical work showing that flotation was feasible, whereas direct cyanidation of the test material was very slow with high cyanide consumption (described in Section 13.1.1), USBRC evaluated a wider selection of rotary drillhole composite samples with four objectives: depressing arsenic, iron, and zinc; analyzing rougher and cleaner concentrates to produce a concentrate acceptable to a smelter; investigating ways to make cyanidation feasible; and evaluating gravity separation (Woods and Smith, 1984). All of the samples contained a high amount of fines (rotary drill cuttings). Composites were developed from holes S-1, S-3, S-4, S-8, S-13, S-26, S-30, S-42, and S-45. The following is a summary of their conclusions:

- / Screen fractionation and analyses showed increased silver assay with decreasing particle size down to 325 mesh; the minus 325 mesh fraction had a normal silver content. Even the coarsest fraction could not be discarded without serious silver losses.
- / A bulk sample of oxidized surface rock gave Bond work index values of 7.5 kilowatt hours per short ton ("kwh/st") (ball mill) and 5.5 kwh/st (rod mill). Due to a lack of core, no work index data could be obtained on the sulfide mill feed.
- / Flotation of oxidized samples, representing about 20% of the deposit, gave 50-60% recovery of silver, even with use of a sulfidizer. However, these samples were amenable to direct cyanide leaching, providing 94-97% recovery of silver with moderate cyanide consumption.
- / Bulk flotation of sulfide mill feed using a strong collector gave silver recoveries of 87-96%. Lead and zinc recoveries generally also are high, followed by arsenic and iron. Maintaining a high pH (11) during bulk flotation substantially lowered the arsenic, zinc, and iron recoveries with only a slight effect on silver recovery. Changing to more selective collectors (Aero 325 and the Minerecs) gave much lower arsenic recovery, while silver recovery dropped to the 76-89% range.
- / Two stages of cleaning can give 87% recovery of silver, with a second increment of arsenic and iron depression (pH 11). Regrinding of the intermediate rougher concentrate is beneficial.

- / Cyanidation of sulfide mill feed directly is not practicable due to very slow leach rate, poor ultimate silver recovery, and high cyanide consumption. Finer grinding and higher initial cyanide concentration increased the leaching rate but pushed the cyanide consumption even higher.
- / Pre-aeration of bulk concentrates oxidizes iron minerals, and subsequent cyanide leaching gives greatly improved results. Addition of soda ash to maintain a pH near ten during the oxidation is beneficial.
- / Heavy media sink-float experiments were not encouraging.

### 13.1.3 1984-1985 U. S. BORAX RESEARCH CORP.

From February 1984 through December 1985, USBRC undertook extensive metallurgical testing of Trinity oxide and sulfide samples (Ganderup and Woods, 1986). In contrast to the 1983 USBRC and Hazen studies that primarily involved rotary cuttings (Section 13.1.1), this work studied core samples from holes SC-1 through SC-5 (only zones running 1.5 oz silver/ton and higher were composited). This work included a study of mineralogy and determination of specific gravity; gravity separation; rod and ball mill grinding studies; flotation; analysis of cyanidation vs. depth; and column leaching. They also studied agglomeration, percolation, acid extraction, oxidation-reduction potential, and Merrill-Crowe silver recovery. The following are some of their conclusions:

- / Rod mill and ball mill grindability tests on five sulfide core composites yielded moderate work indices, averaging ten kwh/st; rod mill testing of four oxide composites yielded an average work index of 8.7 kwh/st.
- / For five sulfide core composites, in general, silver recovery and the total weight of the flotation concentrate both increased at finer grinds; the ideal grind was around 60-65% minus 200 mesh, at finer grinds minimal improvement was seen in silver recovery.
- / At a grind of 60% minus 200 mesh, the core composites averaged 93.2% recovery of silver to a concentrate containing 10.7% of the initial feed weight.
- / In contrast to earlier results on rotary cuttings, three of the five sulfide core composites yielded poor rejection of arsenic (15-35%) under high pH conditions (pH 11).
- / Specific gravity determinations were made on core samples taken at 50ft intervals within each hole. The mineral specific gravity (without voids) for rhyolite tuff averaged 2.61, intrusive rhyolite averaged 2.65, and argillite averaged 2.66. The rock specific gravities showed more variance for the rhyolites and averaged 2.10 for the tuff, 2.24 for the intrusive rhyolites, and 2.56 for the argillite.
- / Tabling tests on ground core showed that sufficient grinding to relieve gangue locking gave unacceptable silver losses to slimes.
- / Column leaching of agglomerated oxide material showed that at depth, oxide should be amenable to heap leaching, but surface oxide would require finer grinding and give lower recovery.
- / Cyanidation as a function of depth for two rotary and two core holes showed a sharp break in silver recovery between oxide and sulfide material, also suggesting that direct cyanidation of upper sulfide material would not be feasible.

- / Column leach recoveries were low, and cyanide consumption was high on agglomerated sulfide material; air oxidation at ambient temperature prior to cyanidation gave only marginal improvement.
- / Mineralized material from RC drillhole SR-5 was floated to produce rougher concentrate for oxidation/cyanidation studies. Recovery of 93.6% of the silver was achieved in a concentrate with 12.2% of the initial feed weight; rejection of arsenic and iron was good.

#### 13.1.4 1986–1987 KAPPES, CASSIDAY & ASSOCIATES

In late 1986, Kappes, Cassiday & Associates (“KCA”) received five (5) core samples from the Trinity, Nevada property (SC-6 through SC-10). The core samples were assayed (gold and silver) and split to form two composites (upper and lower). These composites were utilized for head assay, bottle roll leach, NaCN/pH optimization, agglomeration and column leach test work. (Dix, 1987)

Results for the (SC-6 through SC-10) composites indicated:

- / Column leach tests on agglomerated material (minus 3/4 inch) indicated an average silver recovery of 84% based on 67 days of leaching and an average calculated head of 6.86 oz silver/ton. Chemical consumption averaged 2.18 pounds of NaCN and 0.23 pounds of hydrated lime per ton or mill feed.
- / Bottle roll leach tests on pulverized material (minus 100 mesh) indicated an average silver recovery of 94% in 24 hours of leaching, based on an average calculated head of 7.16 oz silver/ton. Chemical consumption averaged 1.14 pounds of NaCN and 1.10 pounds of hydrated lime per ton of mill feed.

In late 1987, KCA conducted additional test work on five additional samples from the Trinity property. These samples were identified as *Mn low-grade (5310’, 1.3 oz/ton Ag)*, *Drain from Cell 1&2 Composite*, *Drain from Cell 3 Composite*, *Drain from Cell 4 Composite* and *Drain from Cell 5 Composite*. The material (minus ten mesh) was utilized for bottle roll leach tests. Results for the bottle roll leach test work indicated silver recovery in the range of 27.3% – 52.5% based on calculated heads between 0.44 – 1.13 oz silver/ton of mill feed (Yernberg, 1987).



## 14.0 MINERAL RESOURCE ESTIMATES

No Mineral Resources are reported in this technical report.



## 15.0 MINERAL RESERVE ESTIMATES

No Mineral Reserves are reported in this technical report.



## 16.0 MINING METHODS

No mining methods are reported in this technical report.



## 17.0 RECOVERY METHODS

No recovery methods are reported in this technical report.



**RESPEC**

## **18.0 PROJECT INFRASTRUCTURE**

No project infrastructure is reported in this technical report.



## 19.0 MARKET STUDIES AND CONTRACTS

No market studies and contracts are reported in this technical report.



**RESPEC**

## **20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT**

No environmental studies, permitting, or social or community impacts are reported in this technical report.



**RESPEC**

## **21.0 CAPITAL AND OPERATING COSTS**

No capital or operating costs are reported in this technical report.



**RESPEC**

## **22.0 ECONOMIC ANALYSIS**

No economic analysis is reported in this technical report.



**RESPEC**

## **23.0 ADJACENT PROPERTIES**

The author is not aware of any activity on adjacent properties that is relevant to this report.



**RESPEC**

## **24.0 OTHER RELEVANT DATA AND INFORMATION**

There is no remaining infrastructure related to the 1987-1989 mining operations conducted on the property.

## 25.0 INTERPRETATION AND CONCLUSIONS

The author completed a comprehensive review of Liberty project data, including the drillhole database. The review included a field visit to the project site. Based on this review, two primary controls on the silver, lead, and zinc mineralization were identified and used to model the mineral domains.

Mineralization is controlled by (1) a northeast-trending zone of normal faults, the primary one being along the axis of the pit, and (2) the contact between the underlying Triassic sedimentary rocks and overlying Tertiary rhyolitic volcanic rocks, creating a tabular, slightly northwest-dipping zone. The bulk of this low-angle tabular zone is in the rhyolites.

Silver mineralization at Trinity occurs as oxidized and unoxidized sulfides in veinlets, as fracture-controlled mineralization, and as disseminated mineralization primarily within Tertiary rhyolite porphyry, aphanitic rhyolite, and volcanoclastic rocks. There is an oxide zone that overlies sulfide mineralization.

Between 1982 and 1985, 273 rotary percussion and reverse circulation holes were drilled by Borax to explore and define the Trinity mineralization. The bulk of the silver, lead, and zinc analyses were completed by Borax. A review of the QA/QC data collected between 1982 and 1985 identified inconsistencies with the Borax results and check analyses. The check analyses by commercial laboratories are not within acceptable ranges. Twinning of at least some of the early drill holes is required to compare verifiable analyses with the Borax values.

The oxide portion of the Trinity silver deposit was mined by a joint-venture of Borax and SFPM from September 1987 to August 1988, with leaching continuing into 1989. During that time, the mine produced about five million ounces of silver from about 1.1 million tons of oxidized ore grading six ounces of silver per ton. Mining ceased when sulfide mineralization was encountered. Although metallurgical testing on the sulfide mineralization indicated there was potential to recover silver, lead, and zinc, metal prices were too low at that time to support mining of the sulfide mineralization.

The author was provided with a project drillhole database consisting of information derived from 199 rotary percussion holes, 117 reverse circulation holes, and ten core holes. These holes were drilled by Borax, Santa Fe Pacific Mining, and Renaissance. Mr. Bickel completed and checked the analytical portion of the database and entered a standardized general lithologic coding into the database.

Trinity is considered to be a property of merit by the author. There is widespread mineralization at the property. In addition, the large area of geochemically anomalous lead and zinc encountered in the drilling is suggestive of an intrusive-related system that has not been systematically explored, especially to the south of Section 9.

## 26.0 RECOMMENDATIONS

The author recommends two phases of work to advance the project:

1. Comprehensive data compilation and evaluation,
2. Drilling approximately ten dry reverse circulation holes with a focus on twinning at least eight historical drillholes to confirm their results.

Phase 1 should focus on compilation, modernization, and organization of data to evaluate the known mineralization and generate targets for further exploration and understanding gaps and shortcomings of the historical work on the project. This work should include creating an inventory and re-logging available core/chips at the storage facility in Sparks, Nevada. The cost of this Phase 1 work program is approximately \$150,000 and is based primarily on consulting fees for geologic evaluations. Upon completion of this data review phase, a decision should be made as to whether or not to continue to advance the property. Given positive results, a phase 2 program should be enacted.

Phase 2 should focus on drilling, specifically to twin/validate historical holes, and test one or two exploration targets identified in Phase 1. The author estimates that for dry reverse circulation drilling, all-in costs including assaying, logging, and reclamation are approximately \$75/foot and that the total cost for this phase is approximately \$300,000.

Table 26-1 summarizes the above recommendations with estimated costs.

Table 26-1. Estimated Costs of the Trinity Program

Item	Estimated Cost
Phase 1: Data Compilation and Evaluation	\$ 150,000
Phase 2: RC Drilling and Assaying (~4,000 feet)	\$300,000
<i>Total</i>	<i>US \$450,000</i>

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## 28.0 DATE AND SIGNATURE PAGE

Effective Date of report:

October 16, 2025

Completion Date of report:

October 28, 2025

***"Jeffrey Bickel"***

\_\_\_\_\_  
Jeffery Bickel, C.P.G.

Date Signed:

***Oct. 28, 2025***

## 29.0 CERTIFICATE OF QUALIFIED PERSONS

### CERTIFICATE OF QUALIFIED PERSON

I, Jeffrey Bickel, C.P.G. (AIPG) and Registered Geologist (Arizona), do hereby certify that:

1. I am currently employed as a Senior Geologist at RESPEC Company LLC (formerly Mine Development Associates, Inc.) ("RESPEC"), at 210 South Rock Blvd, Reno, Nevada, 89502.
2. This certificate applies to the technical report titled "Technical Report for the Trinity Project, Pershing County, Nevada, USA", with an Effective Date of October 16, 2025 (the "Technical Report"), prepared for K9 Gold Corp. ("K9").
3. I graduated with a Bachelor of Science degree in Geological Sciences from Arizona State University in 2010. I am a Certified Professional Geologist (#12050) with the American Institute of Professional Geologists. I am also a Registered Geologist in the State of Arizona (#60863).
4. I have worked as a geologist continuously for over 15 years since graduating from university. During that time, I have been engaged in the exploration, definition, and modeling of precious and base metal mineral deposits in North America and have estimated the mineral resources for such deposits, including silver deposits in Nevada similar to Trinity.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I visited the Trinity Project site on October 16, 2025.
7. I am responsible for all sections of the Technical Report.
8. I am independent of K9 Gold Corp and all its subsidiaries as described in Section 1.5 of NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28<sup>th</sup> day of October, 2025.



**"Jeffrey Bickel" ("signed" and "sealed")**

Jeffrey Bickel, C.P.G. (# 12050)



# APPENDIX A

## LIST OF SIGNIFICANT INTERSECTIONS



# APPENDIX A: LIST OF SIGNIFICANT INTERSECTIONS

Hole ID	Collar						Significant Mineralized Intervals							High-Grade Subsets							
	Easting	Northing	Elevation ft	Azimuth	Dip	Hole Depth	From ft	To ft	Length ft	True Width	Ag ppm	Pb ppm	Zn ppm	From ft	To ft	Length ft	True Width ft	Ag ppm	Pb ppm	Zn ppm	
S-1	493891.44	2056321.59	5306.95	0	-90	200	0	140	140	140	72	3630	3094								
S-100	492648.53	2055820.08	5315.93	0	-90	505	40	195	155	89	26	280	905	90	120	30	17	953	8192	2675	
S-100							380	460	80	46	20	2394	2408								
S-106	492204.68	2055540.22	5256.71	0	-90	405	135	230	95	54	75	8476	9388								
S-108	492007.95	2055498	5258.37	0	-90	405	50	100	50	29	35	155	721								
S-108							305	365	60	34	47	1019	1272								
S-109	492746.48	2055845.95	5317.42	0	-90	445	0	75	75	43	114	1111	383								
S-109							260	410	150	86	22	2984	3044								
S-111	492265.34	2055363.64	5280.9	0	-90	405	110	240	130	75	20	1261	1540								
S-112	492298.48	2055556.5	5266.86	0	-90	385	135	260	125	72	47	2056	2426								
S-113	492107.93	2055515.48	5260.31	0	-90	380	10	160	150	86	87	847	531	125	130	5	3	1000	3770	65	
S-113							245	380	135	77	209	4114	4351	270	300	30	17	670	12787	11457	
S-114	492145.9	2055322.81	5273.88	0	-90	405	225	275	50	29	40	1991	2031								
S-115	492045.82	2055307.43	5268.42	0	-90	405	140	190	50	29	81	1044	1401								
S-115							270	330	60	34	40	3322	2836								
S-116	491950.13	2055281.69	5264.36	0	-90	385	10	150	140	80	38	655	524								
S-117	491969.64	2055185.89	5270.34	0	-90	405	45	110	65	37	35	320	185								
S-119	492187.23	2055639.17	5267.43	0	-90	385	125	245	120	69	112	3287	3567								
S-120	492269.39	2055661.55	5276.41	0	-90	405	125	185	60	34	43	591	411								
S-121	492474.61	2055700.76	5291.47	0	-90	455	385	450	65	37	28	5968	4595								
S-125	491043.216	2054491.566	5256.8	0	-90	405	315	385	70	40	32	137	102								
S-126	493249.65	2055869.44	5349.25	0	-90	280	175	225	50	50	21	462	637								
S-127	493114.26	2055520.97	5342.58	0	-90	340	100	260	160	160	29	9037	1923								
S-128	492735.04	2055946.07	5334.99	0	-90	325	25	240	215	123	311	3312	358	135	185	50	29	809	7515	240	
S-13	493425.35	2056565.3	5349.26	0	-90	405	150	405	255	146	134	1231	1244								
S-131	492223.25	2055438.1	5273.91	0	-90	400	15	110	95	54	42	613	437								
S-131							160	305	145	83	85	2981	3226								
S-133	492024.29	2055401.09	5262.32	0	-90	385	10	110	100	57	22	166	323								
S-133							285	385	100	57	409	7508	5359	320	340	20	11	1337	18450	10540	
S-136	494341.86	2055072.68	5520.21	0	-90	405	170	230	60	60	38	9402	3163								
S-140	492572.44	2055722.34	5294.37	0	-90	485	290	485	195	112	27	6904	6076								
S-141	492373.61	2055678.05	5286.48	0	-90	345	220	275	55	32	43	988	1531								
S-147	494169.85	2056123.84	5418.43	0	-90	250	0	110	110	110	49	8867									
S-148	494234.71	2055045.65	5491.95	0	-90	400	55	220	165	165	31	8506									
S-149	494182.63	2055325.85	5475.61	0	-90	400	140	390	250	250	22	5553									
S-153	492668.13	2055748.1	5301.3	0	-90	420	270	350	80	46	25	7273	5734								

Collar							Significant Mineralized Intervals							High-Grade Subsets							
Hole ID	Easting	Northing	Elevation ft	Azimuth	Dip	Hole Depth	From ft	To ft	Length ft	True Width	Ag ppm	Pb ppm	Zn ppm	From ft	To ft	Length ft	True Width ft	Ag ppm	Pb ppm	Zn ppm	
S-154	492822.36	2055971.75	5336.45	0	-90	400	190	400	210	120	26	328	306								
S-16	493899.27	2056116.89	5318.42	0	-90	245	0	115	115	115	41	3297	3144								
S-170	492396.19	2055584.37	5273.39	0	-90	440	120	270	150	86	24	4801	4171								
S-174	492918.39	2055996.64	5339.76	0	-90	440	280	320	40	23	34	1648	1726								
S-174							350	395	45	26	19	3800	3252								
S-175	493013.49	2056020.56	5343	0	-90	420	90	255	165	95	61	956	821								
S-178	491990.48	2055595.94	5247.9	0	-90	440	30	80	50	29	37	37	462								
S-179	491795.38	2055054.23	5268.03	0	-90	420	95	195	100	57	83	1428	1060								
S-198	493264.05	2055767.41	5367.19	0	-90	360	130	240	110	110	37	10451	6350								
S-199	493216.23	2055545.8	5358.28	0	-90	340	155	295	140	140	33	11414	7112								
S-2	493963.8	2055782.57	5340.75	0	-90	305	40	130	90	90	23	6669	8398								
S-200	493166.74	2055331.71	5358.08	0	-90	470	305	365	60	60	25	143	9243								
S-203	494219.71	2055183.34	5465.14	0	-90	300	110	160	50	50	22	4852	960								
S-207	493916.02	2055508.16	5359.38	0	-90	380	55	145	90	90	26	9588	6412								
S-208	492692.81	2055879.71	5328.43	0	-90	300	200	265	65	37	31	550	1043								
S-25	493449.71	2056389.15	5358.03	0	-90	445	185	445	260	149	40	2659	2687								
S-26	493450.48	2055915.47	5381.59	0	-90	375	110	270	160	160	31	6031	5806								
S-27	493281	2056165	5369.18	0	-90	425	300	400	100	57	27	3208	3426								
S-29	493651.2	2055531.64	5436.23	0	-90	445	300	355	55	55	27	3350	989								
S-3	493478.93	2056772.97	5274.12	0	-90	205	0	45	45	26	106	923	433								
S-30	493615.69	2056605.07	5277.6	0	-90	225	0	225	225	129	91	777	740								
S-31	493797.35	2056499.67	5289.88	0	-90	205	0	205	205	205	54	723	781								
S-32	493767.07	2055971.35	5335.09	0	-90	290	30	175	145	145	30	5792	5276								
S-34	493231.97	2056533.84	5339.58	0	-90	405	60	120	60	34	21	80	233								
S-34							295	405	110	63	61	689	653								
S-39	493250.38	2056369.17	5348.17	0	-90	485	215	470	255	146	59	1183	1274								
S-4	493473.99	2056218.81	5371.86	0	-90	340	230	340	110	110	91	2755	3418								
S-40	493277.96	2056732.78	5317.64	0	-90	405	145	230	85	49	25	194	329								
S-40							360	405	45	26	30	235	299								
S-41	493342.12	2056942.96	5242.75	0	-90	265	15	180	165	95	33	211	203								
S-42	493435.62	2056477.7	5351.11	0	-90	555	35	550	515	295	50	844	867								
S-43	493462.87	2056300.65	5364.46	0	-90	405	200	380	180	103	32	3240	3341								
S-44	493501.8	2056120.93	5383.62	0	-90	405	130	320	190	190	34	3871	3113								
S-45	493503.05	2056589.83	5318.06	0	-90	445	0	425	425	244	86	762	753								
S-46	493680.88	2055741.18	5398.63	0	-90	345	180	270	90	90	30	2668	11736								
S-47	493534.69	2056985.07	5236.51	0	-90	285	50	120	70	40	30	499	197								
S-48	493695.36	2056067.23	5374.17	0	-90	325	125	275	150	150	25	4395	5016								
S-49	493766.36	2056828.26	5252.26	0	-90	245	70	175	105	60	41	335	454								

Hole ID	Collar						Significant Mineralized Intervals							High-Grade Subsets							
	Easting	Northing	Elevation ft	Azimuth	Dip	Hole Depth	From ft	To ft	Length ft	True Width	Ag ppm	Pb ppm	Zn ppm	From ft	To ft	Length ft	True Width ft	Ag ppm	Pb ppm	Zn ppm	
S-5	493427.69	2055700.08	5404.41	0	-90	395	190	290	100	100	37	9710	9620								
S-50	493803.26	2056634.58	5264.99	0	-90	225	130	190	60	34	19	190	131								
S-51	493890.36	2056207.63	5310.24	0	-90	200	5	80	75	75	51	7933	7086								
S-52	493917.91	2056010.64	5320.5	0	-90	345	10	95	85	85	41	7890	5104								
S-53	493963.98	2056355.25	5322.25	0	-90	305	20	110	90	90	44	2220	1362								
S-54	493998.99	2056152.12	5356.97	0	-90	305	75	150	75	75	112	3186	3876	125	130	5	5	1041	10000	9000	
S-55	492978.18	2056214.79	5363.34	0	-90	485	340	415	75	43	59	506	543								
S-56	493127.23	2056509.75	5335.42	0	-90	505	320	505	185	106	72	622	633								
S-57	493140.83	2056344.08	5341.63	0	-90	485	225	470	245	141	48	567	611								
S-58	493208.11	2056620.44	5331.78	0	-90	425	280	420	140	80	49	525	538								
S-59	493322.49	2056544.13	5343.45	0	-90	505	50	505	455	261	55	612	628								
S-60	493390.09	2056108.33	5376.89	0	-90	425	210	325	115	115	43	5692	6643								
S-61	493403.8	2056654.63	5336.08	0	-90	425	0	405	405	232	47	481	450								
S-62	493457.29	2056870.8	5245.26	0	-90	305	20	305	285	163	54	446	444								
S-64	493636.3	2056445.56	5291.12	0	-90	405	120	205	85	85	48	748	767								
S-65	493648.58	2056344.41	5311.76	0	-90	345	65	200	135	135	62	1515	1077	155	160	5	5	667	5340	6000	
S-66	493795.93	2056170.43	5298.06	0	-90	285	75	170	95	95	29	1024	1470								
S-67	493667.46	2056807.72	5253.13	0	-90	275	60	215	155	89	71	498	715								
S-68	493716.86	2056616.3	5267.38	0	-90	245	30	135	105	60	67	608	965								
S-68							205	245	40	23	24	198	206								
S-69	493777.38	2056271.47	5292.35	0	-90	240	5	135	130	130	28	4006	4325								
S-7	493944.11	2055282.89	5380.52	0	-90	200	110	170	60	60	26	6902	2828								
S-72	493585.92	2055735.27	5423.33	0	-90	325	230	310	80	80	24	2635	11137								
S-75	493362.55	2056852.4	5261.98	0	-90	305	5	215	210	120	35	263	191								
S-79	492154.65	2055066.94	5290.16	0	-90	405	5	60	55	32	25	399	235								
S-8	493669.99	2056258.03	5316.58	0	-90	305	20	190	170	170	37	4229	3437								
S-80	492346.93	2055533.35	5274.53	0	-90	425	110	270	160	92	37	2099	2528								
S-81	493295.008	2055672.22	5374.85	0	-90	405	205	260	55	55	66	10456	15871								
S-82	493210.45	2056066.49	5356.62	0	-90	425	300	355	55	32	21	3890	3972								
S-84	493539.05	2055926.7	5400.4	0	-90	365	135	300	165	165	28	4839	5617								
S-85	493599.78	2056144.87	5378.59	0	-90	365	130	290	160	160	29	3652	3865								
S-86	493367.53	2056290.25	5360.29	0	-90	405	310	405	95	54	37	3858	3793								
S-87	493172.35	2056254.92	5357.77	0	-90	545	135	325	190	109	51	1050	1194								
S-87							360	545	185	106	36	679	737								
S-88	492932.4	2056453.23	5316.21	0	-90	485	345	385	40	23	26	138	225								
S-89	493031.29	2056979.76	5289.4	0	-90	405	185	295	110	63	38	233	252								
S-91	492104.59	2055586.47	5259.38	0	-90	385	10	225	215	123	141	3030	2259	175	180	5	3	608	6660	8830	
S-92	492128.48	2055419.7	5270.22	0	-90	405	80	170	90	52	58	918	371								

Collar							Significant Mineralized Intervals							High-Grade Subsets							
Hole ID	Easting	Northing	Elevation ft	Azimuth	Dip	Hole Depth	From ft	To ft	Length ft	True Width	Ag ppm	Pb ppm	Zn ppm	From ft	To ft	Length ft	True Width ft	Ag ppm	Pb ppm	Zn ppm	
S-92							250	375	125	72	37	1569	1850								
S-94	491931.6	2055383.78	5259.3	0	-90	365	40	165	125	72	47	473	248								
S-96	491875.02	2055159.13	5256.66	0	-90	405	20	135	115	66	105	1016	891	95	100	5	3	1081	9430	8330	
SA-1	493349.42	2056387.36	5354.18	349	-55	585	60	585	525	55	72	1020	953								
SA-10	492824.81	2055889.79	5321.02	301	-61.5	445	130	215	85	10	129	1931	251								
SA-11	492808.1	2056067.97	5293.64	153.5	-46	300	40	170	130	121	268			90	125	35	33	599			
SA-11							255	300	45	42	26										
SA-12	492630.1	2055932.6	5295	153	-45	280	50	185	135	127	183			115	120	5	5	709			
SA-2	493397.09	2056659.01	5336.83	79	-70	380	5	380	375	268	54	1048	748								
SA-3	493272.5	2056533.58	5341.37	79	-51.5	490	60	490	430	331	95	1073	942								
SA-4	493438.71	2056411.17	5356.5	81	-70	530	290	450	160	116	31	2211	2008								
SA-5	493414.33	2056592.24	5345.58	349	-54	425	25	380	355	32	91	879	598								
SA-6	493466.04	2056287.19	5363.93	349	-45	540	145	470	325	14	72	1205	972								
SA-8	492241.66	2055344.38	5279.58	304	-45	480	20	245	225	38	44	753	352								
SA-8							355	455	100	17	103	2431	2513								
SA-9	492820.89	2055894.54	5320.76	301	-45	345	140	215	75	12	273	3349	359	140	145	5	1	1021	6980	727	
SR-1	493344.68	2056435.54	5352.23	0	-90	540	45	540	495	284	52	1031	1346								
SR-10	493622.07	2056598.92	5277.87	0	-90	400	5	400	395	227	73	899	834								
SR-11	493347.38	2056393.13	5354.59	0	-90	540	60	225	165	95	34	645	644								
SR-11							340	540	200	115	91	2452	2540								
SR-12	493237.38	2056448.09	5344.39	0	-90	480	225	420	195	112	145	1951	2085								
SR-13	493138.43	2056425.09	5327.53	0	-90	500	25	135	110	63	18	194	266								
SR-13							290	370	80	46	32	388	318								
SR-14	493388.59	2056192.21	5368.36	0	-90	480	215	385	170	170	31	2151	2640								
SR-15	493272.36	2056272.14	5355.93	0	-90	540	85	445	360	206	30	1524	1614								
SR-16	493073.32	2056234.8	5359.25	0	-90	520	35	155	120	69	25	361	234								
SR-17	493182.12	2056715.41	5330.03	0	-90	480	245	425	180	103	72	705	585								
SR-19	493578.78	2056240.02	5357.13	0	-90	405	105	200	95	95	25	4166	2415								
SR-2	493366.72	2056551.68	5346.74	0	-90	500	10	485	475	272	99	935	834	25	35	10	6	733	3695	890	
SR-2										0				350	355	5	3	946	7010	6000	
SR-21	493091.01	2056693.95	5341.1	0	-90	505	265	430	165	95	82	599	548								
SR-22	493068.98	2056784.36	5326.05	0	-90	405	260	340	80	46	46	316	252								
SR-23	493058.18	2056874.31	5314.83	0	-90	405	235	390	155	89	54	364	257								
SR-27	493110.64	2056596.68	5329.21	0	-90	545	315	400	85	49	120	1166	1028								
SR-28	493103.93	2056141.79	5357.1	0	-90	520	135	295	160	92	28	1087	1480								
SR-3	493315.72	2056641.99	5339.75	0	-90	500	40	480	440	252	49	487	600								
SR-30	492994.6	2056675.75	5309.43	0	-90	545	255	335	80	46	84	411	348								
SR-31	493166.86	2056807.08	5317.55	0	-90	505	280	360	80	46	51	435	397								

Collar							Significant Mineralized Intervals							High-Grade Subsets							
Hole ID	Easting	Northing	Elevation ft	Azimuth	Dip	Hole Depth	From ft	To ft	Length ft	True Width	Ag ppm	Pb ppm	Zn ppm	From ft	To ft	Length ft	True Width ft	Ag ppm	Pb ppm	Zn ppm	
SR-32	493264.48	2056842.27	5283.7	0	-90	465	95	290	195	112	102	700	509								
SR-35	493624.36	2056529.98	5284.9	0	-90	340	5	215	210	120	58	1169	1108								
SR-36	493570.15	2056696.2	5276.6	0	-90	310	80	230	150	86	35	706	1168								
SR-4	493431.92	2056412.95	5356.97	0	-90	540	290	540	250	143	77	4856	4676								
SR-41	493290.53	2056730.93	5317.61	0	-90	480	365	460	95	54	36	432	313								
SR-5	493584	2056681.82	5276.62	0	-90	380	0	295	295	169	48	504	651								
SR-55	493740.26	2056476.69	5275	0	-90	300	25	140	115	115	143	1883	1965								
SR-56	493830.55	2055987.12	5315	0	-90	200	15	175	160	160	47	8295	5341								
SR-58	492072.36	2055577.52	5273.35	0	-90	250	15	205	190	109	106			155	160	5	3	755			
SR-59	492152.8	2055578.96	5280.88	0	-90	250	50	250	200	115	107										
SR-6	493546.88	2056889.14	5242.96	0	-90	300	40	180	140	80	51	387	351								
SR-60	492239.84	2055620.26	5289.57	0	-90	200	10	190	180	103	74										
SR-61	492260.68	2055506.73	5264.31	0	-90	200	45	200	155	89	46	1433	9								
SR-62	492337.98	2055626.19	5275.75	0	-90	240	20	205	185	106	48	808	324								
SR-64	492223.8	2055698.37	5277.14	0	-90	285	130	260	130	75	124	2501	815								
SR-65	492640.82	2055872.97	5295	0	-90	245	25	230	205	118	486	3355		95	175	80	46	1099	5347		
SR-67	492809.05	2056014.96	5295	0	-90	265	5	155	150	86	95	1723	437								
SR-68	492562.12	2055765.17	5296.04	0	-90	225	90	205	115	66	519	3800		120	170	50	29	1038	6464		
SR-69	492168.48	2055460.12	5268.54	0	-90	225	115	225	110	63	52	2025									
SR-7	493840.98	2056398	5301.68	0	-90	260	0	105	105	105	37	1096	1201								
SR-8	493765.02	2056378.89	5284.58	0	-90	320	45	120	75	75	45	2006	2362								
SR-9	493699.4	2056715.25	5259.43	0	-90	240	40	185	145	83	47	513	499								
TS07-15	492052.2014	2055228.065	5270	130	-50	665	85	130	45	43	32	497	226								
TS07-21	491882.6116	2055810.462	5275	134.4	-48.2	725	560	715	155	151	35	1799	2068								
TS07-22	492182.2593	2055918.215	5305	181.4	-47.9	765	245	480	235	181	70	1370	2098								
TS07-24	492429.2919	2056087.048	5330	130	-50	650	500	575	75	72	20	3735	4409								
TSD-002	492064.1723	2055450.114	5190	37	-70	300	5	117	112	59	33	319	1004								
TSD-002							207	257	50	26	88	2055	7608								
TSD-003	492589.4781	2055699.868	5130	315	-45	298	130	210	80	14	27	564	1728								
TSD-007	492351.9449	2055352.457	5284	307.4	-45.1	556	70	155	85	15	75	496	203	140	145	5	1	891	2390	431	
TSD-008	491907.8016	2055235.946	5263	38.6	-55	600	345	410	65	30	80	4396	5590								
TSD-009	492202.594	2055242.868	5282	304.6	-45.6	600	10	150	140	22	28	343	292								