Amended and Restated
NI 43-101 PRELIMINARY ECONOMIC ASSESSMENT FOR THE PINE GROVE PROJECT, LYON COUNTY, NEVADA

Prepared for Lincoln Mining Corporation
A British Columbia, Canada Corporation

And
Lincoln Gold US Corporation
A Nevada Corporation

Effective Date: December 8, 2011
Report Date: February 4, 2015

Prepared By
Patricia A. Maloney, SME-RM
Douglas W. Willis, CPG
Randall K. Martin, SME-RM
John D. Welsh, PE
Thom Seal, PE

DATE AND SIGNATURES PAGE

This amended and restated report entitled NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, dated February 4, 2014 with an effective date of December 8, 2011, was prepared and signed by the following qualified persons:

Prepared by:	**“Original document signed by Patricia A. Maloney”** **Patricia A. Maloney, SME-RM**	**Date: February 4, 2015**
Prepared by:	**“Original document signed by Douglas W. Willis”** **Douglas W. Willis, CPG**	**Date: February 4, 2015**
Prepared by:	**“Original document signed by Randall K. Martin”** **Randall K. Martin, SME-RM**	**Date: February 4, 2015**
Prepared by:	**“Original document signed by John D. Welsh”** **John D. Welsh, PE**	**Date: February 4, 2015**
Prepared by:	**“Original document signed by Thom Seal”** **Thom Seal Ph.D., P.E.**	**Date: February 4, 2015**

CERTIFICATE OF QUALIFIED PERSON

Patricia A. Maloney
Mining Engineer
Welsh Hagen Associates (formerly Telesto Nevada, Inc.)
5490 Longley Lane
Reno, Nevada 89511
Telephone: 775.853.7666
Fax: 775.853.9191
Email: pmaloney@custertel.net

I, Patricia A. Maloney, do hereby certify that:

1. I am an independent consultant working with Welsh Hagen Associates (formerly Telesto Nevada, Inc.), an engineering firm located in Reno, Nevada, USA. This certificate is part of the report titled “Amended & Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, dated February 4, 2015, prepared for Lincoln Mining Corporation and Lincoln Gold US Corporation.
2. I graduated from West Virginia University with a Bachelor of Science Degree in Mining Engineering in 1980.
3. I have practiced my profession as a mine engineer continuously since graduation for a total of 32 years. I have worked in open pit coal mines, copper mines, molybdenum mines and gold mines, evaluating reserves, mine economics and mine planning. My experience also includes equipment selection, production management and involvement in feasibility studies at all levels.
4. I am a Registered Professional Engineer in the states of Wyoming (License No. 5120), Montana (License No. 16929), and Idaho (License No. 6180). I am a Registered Member (# 2009570RM) in good standing with the Society of Mining, Metallurgy and Exploration (SME).
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that I do fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I visited the property on October 31, 2011 – all day.
7. I am responsible for the following sections or sub-sections of the report entitled, “Amended & Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, dated February 4, 2015, “(the “Technical Report”): Sections 1.0, 1.2, 1.3, 1.6, 1.7, 1.8, 2, 3, 5, 16, 19, 20, 21, 22, 25.5, 26.2, 26.4, 26.7 and 27.
8. The effective date of the Technical Report is December 8, 2011.
9. I am independent of Lincoln Gold, applying all of the tests in section 1.5 of NI 43-101.
10. I have had no prior involvement with the property that is subject to this Technical Report.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with the instrument.
12. As of the date of this certificate, 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.
13. I consent to the public filing of this Technical Report, only in its entirety, in a prospectus or any similar offering document, for presentation to any stock exchange or other regulatory authority, and for publication, including electronic publication accessible by the public. This consent extends as well to all other forms of written disclosure.

Dated this 4th day of February, 2015.

“Signed and Sealed”

Patricia A. Maloney

Patricia A. Maloney, PE

CERTIFICATE OF QUALIFIED PERSON

Douglas W. Willis
Senior Geologist
Welsh Hagen Associates (formerly Telesto Nevada, Inc.)
5490 Longley Lane
Reno, Nevada 89511
Telephone: 775.853.7776
Email: dwillis@welshhagen.com

I, Douglas W. Willis, do hereby certify that:

1. I am an independent consultant working with Welsh Hagen Associates (formerly Telesto Nevada, Inc.), an engineering firm located in Reno, Nevada, USA.
2. This certificate is part of the report titled "Amended and Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, Nevada, dated February 4, 2015, prepared for Lincoln Mining Corporation and Lincoln Gold US Corporation.
3. I graduated from California State University, Chico with a Bachelor of Science degree in Geology in 1987.
4. I have practiced my profession as a geologist for 11 years primarily focusing on gold exploration in Nevada, USA. I have managed numerous drill programs, overseen drill sampling programs and conducted geological investigations for numerous projects in the western United States. I have worked for a mining engineering firm focused on all aspects of mine permitting, mine planning and production for 3 years.
5. I am a Certified Professional Geologist (#11371) in good standing with the American Institute of Professional Geologists (AIPG).
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that I do fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I visited the property on January 28, 2011 – all day.
8. I am responsible for the following sections or sub-sections of the report entitled, "Amended and Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, Nevada, dated February 4, 2015, "(the "Technical Report"): Sections 1.1, 4, 6, 7, 8, 9, 10, 11, 12, 23, 24, 25.1, 25.4, 26.1, 26.3, 26.5.
9. The effective date of the Technical Report is December 8, 2011.
10. I am independent of Lincoln Gold, applying all of the tests in section 1.5 of NI 43-101.
11. I have had no prior involvement with the property that is subject to this Technical Report.
12. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with the instrument.
13. As of the date of this certificate, 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.
14. I consent to the public filing of this Technical Report, only in its entirety, in a prospectus or any similar offering document, for presentation to any stock exchange or other regulatory authority, and for publication, including electronic publication accessible by the public. This consent extends as well to all other forms of written disclosure.

Dated this 4th day of February, 2015.

"Signed and Sealed"

Douglas W. Willis

Douglas W. Willis, CPG

CERTIFICATE OF QUALIFIED PERSON

Randall K. Martin
Mineral Modeler/Mine Planner
Welsh Hagen Associates (formerly Telesto Nevada, Inc.)
5490 Longley Lane
Reno, Nevada 89511
Telephone: 775.853.7666
Fax: 775.853.9191
Email: rkmassoc@comcast.net

I, Randall K. Martin, do hereby certify that:

1. I am an independent consultant working with Welsh Hagen Associates (formerly Telesto Nevada, Inc.), an engineering firm located in Reno, Nevada, USA.
2. This certificate is part of the report titled "Amended and Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, Nevada, dated February 4, 2015, prepared for Lincoln Mining Corporation and Lincoln Gold US Corporation.
3. I graduated from Colorado School of Mines with a Bachelor of Science Degree in Metallurgical Engineering in 1977, and a Master of Science Degree in Mineral Economics in 1978.
4. I have practiced my profession as a mineral modeler and mine planner continuously since graduation for a total of 34 years. Although my BS degree is in Metallurgical Engineering, I have spent my entire career as a resource modeler, mine planner, and computer software engineer. My first ten years of employment were with the exploration division of a major mining company. I have also worked a total of 14 years as a full-time employee for three major consulting companies. In addition, I am president and owner of a software company that specializes in mineral modeling and mine planning software.
5. I am a Registered Member (# 4063888RM) in good standing with the Society of Mining, Metallurgy and Exploration (SME).
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that I do fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I have not visited the property but was informed of the geologic conditions and controls to mineralization by the Geological Setting and Mineralization Qualified Person.
8. I am responsible for the following sections of the report entitled, "Amended and Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, Nevada, dated February 4, 2015, prepared for Lincoln Mining Corporation and Lincoln Gold US Corporation, (the "Technical Report"): Sections 1.4, 14, 15, and 25.2.
9. The effective date of the Technical Report is December 8, 2011.
10. I am independent of the issuer, applying all of the tests in section 1.5 of NI 43-101.
11. I have had no prior involvement with the property that is subject to this Technical Report.
12. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with the instrument.
13. As of the date of this certificate, 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.
14. I consent to the public filing of this Technical Report, only in its entirety, with a prospectus or any similar offering document, for presentation to any stock exchange or other regulatory authority, and for publication, including electronic publication accessible by the public. This consent extends as well to all other forms of written disclosure.

Dated this 4th day of February, 2015

"Signed and Sealed"

Randall K. Martin

Randall K. Martin, SME-RM

CERTIFICATE OF QUALIFIED PERSON

John D. Welsh, P.E.
Senior Principal Engineer
Welsh Hagen Associates (formerly Telesto Nevada, Inc.)
5490 Longley Lane
Reno, Nevada 89511
Telephone: 775.853.7666
Fax: 775.853.9191
Email: jwelsh@telesto-inc.com

I, John D. Welsh, P.E. do hereby certify that:

1. I am a Senior Principal Engineer with Welsh Hagen Associates (formerly Telesto Nevada, Inc.), an engineering firm located in Reno, Nevada, USA.
2. I graduated from the Missouri School of Mines with a Bachelor of Science Degree in Civil Engineering in 1970 and from Colorado State University with a Masters' Degree in Geotechnical Engineering in 1979.
3. I am an active Registered Professional Civil Engineer in the State of Nevada, No. 6296, and the State of California, No. 35861, and I have voluntarily inactivated registrations in the following states: Colorado, Arizona, Alaska, Montana, Washington, Oregon, Idaho, Wyoming and Missouri.
4. My career as a professional engineer has primarily been involved with design and construction of mines and mining facilities. Since joining Telesto Nevada, I have personally participated in prefeasibility level designs and cost estimates for the Springer Mine (Golden Predator, unpublished), Isabella Mine (private owner) and the Borealis Mine.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, registration as a professional engineer and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101,
6. I am responsible for Sections 17.1, 17.2, 17.3, 17.4, and 18 of this report entitled, "Amended and Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, Nevada, dated February 4, 2015, prepared for Lincoln Mining Corporation and Lincoln Gold US Corporation, (the "PEA"). My last personal inspection of the Pine Grove property was on June 15, 2011 for the duration of one day.
7. The effective date of the Technical Report is December 8, 2011.
8. My involvement with the Pine Grove property has been in a consulting capacity to Lincoln Gold.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the study not misleading.
10. I am independent of the issuer applying all of the tests of Section 1.5 of NI 43-101,
11. I have read NI 43-101 and Form 43-101F1, and the PEA has been prepared in compliance with that instrument and that form.
12. I consent to the filing of the PEA with the stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Pre- Feasibility Study Update. This consent extends as well to all other forms of written disclosure.

Dated this 4th day of February, 2015.

"Signed and Sealed"

John D. Welsh

John D. Welsh, P.E.
Nevada Registered Professional Engineer
No. 6296

CERTIFICATE OF QUALIFIED PERSON

Thom Seal
Mining-Mineral Process Engineer
Differential Engineering Inc.
P.O. Box 8353
Spring Creek, Nevada 89815
Email: tseal@unr.edu

CERTIFICATE OF AUTHOR

I, Thom Seal, Ph.D., P.E. hereby certify that:

1. I am a mining-mineral process engineer and owner of Differential Engineering Inc. a consulting firm, a corporation registered in Nevada and Oregon, and located in, Spring Creek, Nevada 89815.

2. I graduated from the University of Idaho with a M.S. in Metallurgical Engineering in 1988 and a Ph. D. in Mining Engineering-Metallurgy in 2004.

3. I have practiced my profession for over 20 years and have been directly involved in working on metallurgical-mineral processes in North and South America, and South Africa. This work included deposit evaluation, process test work, process evaluation, design, construction, and operation for Newmont Mining Inc. (13 years) as a metallurgical manager for all Newmont's heap leach operations in Nevada.

4. I am registered as a professional mining-mineral process engineer in Nevada (PE No. 15291) and I am also a **founding registered member** of the Society of Mining and Metallurgical Engineers (SME), located in Denver, Colorado. Chapter 5 Rules and Policies, 5.1.1 National Instrument 43-101 – Standards of Disclosure for Mineral Projects: Appendix A: "Recognized Foreign Associations and Designations" "Any state in the United States: and "Licensed or certified as a professional engineer" adopted 12/30/05. Plus "The Society for Mining, Metallurgy and Exploration, Inc (SME)" "Registered Member" adopted on 6/30/11.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, registration as a professional engineer, SME registered member and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I have visited the property on June 15, 2011, but I have not handled any samples, supervised, or managed any of the test work included in reports cited in this Technical Report. My assessment of Section 13 "Mineral Processing and Metallurgical Testing" is based solely on copies of reports, test work, and analytical data provided to me by Telesto. I found the information, data, and test work results found in the documents I was provided to be acceptable as the reports appear to be from well respected metallurgical laboratories that serve the mining industry and I have, at this time, no reason to doubt the veracity of their results. These metallurgical testing laboratories specialize in the evaluation of precious metals extraction processes from heap leach projects, and are well recognized in the mining industry as producing quality test results and reports.

7. I am responsible for Sections: 1.5, 13, 17.5, 25.3, and 26.6, and assessment of the metallurgical test work conducted by several laboratories and consultants for the Pine Grove Property, and contained in Section 13 of this report entitled Amended and Restated NI 43-101 Preliminary Economic Assessment for the Pine Grove Project, Lyon County, Nevada", dated February 4, 2015 and prepared for Lincoln Gold US Corp., (the "PEA").

8. The effective date of the Technical Report is December 8, 2011.

9. I am independent of the issuer, applying all of the tests in Section 1.5 of NI 43-101.

10. My involvement with the Pine Grove property is to serve in a consulting capacity to Lincoln Gold Corporation assisting with the potential to open the property utilizing open pit, heap leach processing to produce gold and silver. This involvement has been from July, 2011 via verbal and written communications.

11. I have read NI 43-101 and Form 43-101F1 and certify that the sections of the PEA for which I am responsible have been prepared in compliance with the Instrument. As of the date of this certificate, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make Section 13 of the PEA and the other sections for which I am responsible not misleading.

12. I consent to the public filing of this written PEA, only in its entirety, in a prospectus or any similar offering document, for presentation to any stock exchange or other regulatory authority, and for publication, including electronic publication accessible by the public. This consent extends as well to all other forms of written disclosure.

Dated this 4th day of February, 2015.

"Signed and Sealed"

Thom Seal

Thom Seal, Ph.D., P.E.
Mining-Metallurgical Engineer

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1.0 EXECUTIVE SUMMARY

On June 14, 2011, Lincoln Mining Corporation and Lincoln Gold US Corporation (herein after referred to as "Lincoln", "Lincoln Gold" or the "Company") engaged Telesto Nevada, Inc. (Telesto) to undertake the preparation of a Preliminary Economic Assessment for gold (Au) on their Pine Grove Project (Project) in the Pine Grove District, Lyon County, Nevada, USA. The Project consists of two separate gold deposits: the Wilson and the Wheeler.

The work by Telesto consisted of updating and verifying an electronic database of drillhole data from logs, performing a statistical analysis on the drillhole data and creating a resource model. The project was also examined for potential economic viability relative to the stated resources. Telesto also presents their interpretations and conclusions in this report.

The preliminary economic assessment is preliminary in nature and includes some inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized. The reported mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.1 Data Limitations

Some of the historical records for underground channel sampling were not well documented. Because of this, the underground channel samples have been excluded from influence on the resource estimate contained in this PEA. Telesto reviewed 100% of the drillhole database and copies of corresponding assay certificates and found them to be a sufficient representation for determining the accuracy of the database. Drillhole collar locations reported on original sheets were also compared to the database information and corrected where necessary. No downhole survey information was available from drillhole records.

1.2 Property Description

The Project, which encompasses approximately 4,586 acres (1,856 hectares) of mineral rights, is located in Lyon County, about 21 miles southeast of Yerington, Nevada (See **Figure 1.1**). The approximate center of the project area is latitude 38° 40' 43" N, longitude -119° 07' 07" W. The property encompasses portions of the following sections; Sec. 36, T10N, R25E; Sec. 28, 29, 31, 32 and 33, T10N, R26E; Sec. 1 and 12, T09N, R25E; and Sec. 1, 4, 5, 6, 7 and 8, T09N, R26E, Mount Diablo Baseline and Meridian (MDB&M).

The Project is accessed via Interstate 80 by traveling approximately 33 miles east from Reno. Exit Interstate 80 at Exit 46 (U.S. Highway 95 Alternate) and turn south (right). Follow the road south and then east for approximately 1.5 miles until reaching the center of Fernley. Turn south (right) onto U.S. Highway 95 Alternate South. Continue on Highway 95 Alternate for 45 miles. Turn east (left) to stay on Highway 95 Alternate at the designated intersection. Yerington is one mile from the intersection. Turn south (right) onto N. Main Street in Yerington, which doubles as Nevada Highway 208. Stay on Nevada Highway 208 for 11 miles. Where Nevada Highway 208 makes a 90° right turn toward Smith Valley (west), continue south onto a dirt road (East Walker Road) which immediately turns southeast. East Walker Road is maintained by the county and is very well-graded. Follow the dirt road for 10 miles until reaching Pine Grove Road. Turn right (west) onto Pine Grove Road and travel approximately 4 miles to reach the Project.

1.2.1 Climate and Physiography

The Pine Grove Project lies on the western edge of the Basin and Range province, a major physiographic region of the western United States. The region is typified by north-northeast trending mountain ranges separated by broad, flat, alluvium filled valleys. The Pine Grove Project is located in the Pine Grove Hills. Elevation of the project ranges from approximately 5,680 to 7,870 feet.

At Yerington, Nevada, the nearest town to the Project area, the average annual precipitation is 5.07 inches, the average maximum annual temperature is 68.8° F, and the average minimum annual temperature is 37.6° F (Western Regional Climate Center data).

1.2.2 Local Resources and Infrastructure

Yerington, Nevada, is approximately 21 miles (34 kilometers) north of the Project. The population of Yerington is 3,048 according to the 2010 Census. The community of Yerington is equipped to provide housing, shopping and schools for mine personnel and their families. Skilled mining personnel are expected to be available in Yerington and from nearby communities such as Reno, Carson City, Fallon, Fernley, and Hawthorne. Reno, a city with a 200,000+ population, is 80 miles northwest of Yerington.

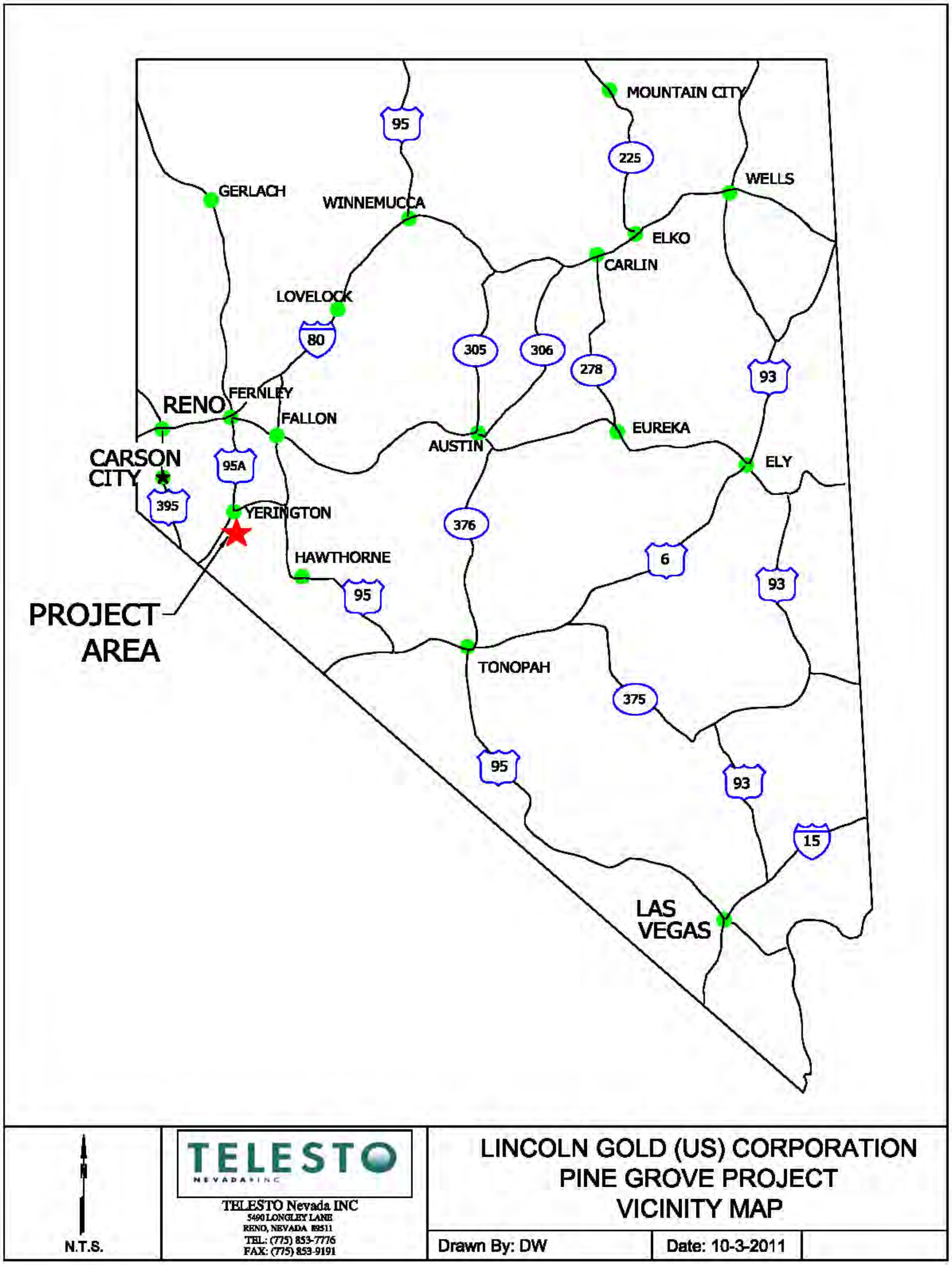


Figure 1.1: Location Map of the Pine Grove Project

1.3 Ownership

A Title Review by G.I.S. Land Services in Reno, Nevada, states that Lincoln controls 243 unpatented mining claims (lode, placer and millsite) and 12 patented mining claims which total ±4,586 acres (1,856 hectares) of land. Lincoln maintains two mining lease agreements on patented claims, the Wheeler Lease and the Wilson Lease. Annual payments on the Wheeler Lease are a fixed $30,000 per year with a sliding scale NSR production royalty (3 to 7%) based on the price of gold. Annual payments on the Wilson Lease are a fixed $25,000 per year with a fixed NSR production royalty of 2.5% and a 5% NSR on all claims staked by Lincoln within a 6 square mile Area of Interest surrounding the Wilson patented claims. In addition, Lincoln purchased eight lode claims, one placer claim, and one millsite claim ("Cavanaugh Group") which carries a fixed 1.5% NSR production royalty. Also, Lincoln purchased three lode claims ("Harvest Group") which carry a 5% NSR production royalty with an option to buy-down 2.5% of the royalty for $100,000 per point. Lincoln Gold US Corp. has staked and controls 100% in 221 lode claims and nine placer claims in the district (G.I.S. Land Services, 2010; Tetra Tech, 2011).

1.4 Resources

The resulting resources reported herein for Pine Grove were classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definitions. Resources are reported as measured, indicated and inferred.

In order to comply with the CIM definition of "reasonable prospects for economic extraction," the following tables report only those blocks contained within a designed pit shell as Mineral Resources.

Gold values were carried in troy ounces per short ton (opt) in the database. The resource is reported in terms of ounces per short ton (opt) and grams per metric tonne (g/t). Total measured and indicated resources are shown in **Table 1.1**. Inferred resources are shown in **Table 1.2**.

Table 1.1: Total Measured and Indicated Gold Resources at Pine Grove

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Measured	2,356	2,137	0.007	0.240	0.041	1.42	97,300	3,026,300
Indicated	1,017	923	0.007	0.240	0.037	1.25	37,200	1,155,600
Measured + Indicated	3,373	3,060	0.007	0.240	0.040	1.37	134,500	4,182,000

At 0.014 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Measured	1,580	1,433	0.014	0.480	0.057	1.95	89,700	2,788,500
Indicated	647	587	0.014	0.480	0.052	1.78	33,600	1,044,900
Measured + Indicated	2,227	2,020	0.014	0.480	0.055	1.90	123,300	3,833,400

At 0.019 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Measured	1,256	1,139	0.019	0.651	0.067	2.30	84,200	2,619,800
Indicated	509	462	0.019	0.651	0.062	2.11	31,300	974,700
Measured + Indicated	1,765	1,601	0.019	0.651	0.065	2.24	115,600	3,594,600

Notes:
- Mineral Resources are reported using a long term gold price of $1425/oz.
- Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
- Cutoff Grade 0.014 opt Au. This is considered “best case”.
- Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
- Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

Table 1.2: Total Inferred Gold Resources at Pine Grove

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	160	145	0.007	0.240	0.041	1.41	6,600	204,100

At 0.014 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	88	80	0.014	0.480	0.067	2.29	5,900	183,000

At 0.019 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	64	58	0.019	0.651	0.086	2.94	5,500	170,500

Notes:
- Mineral Resources are reported using a long term gold price of $1425/oz.
- Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
- Cutoff Grade 0.014 opt Au
- Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
- Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

1.5 Metallurgy and Processing

Differential Engineering Inc. was retained to review the available metallurgical reports on their Pine Grove property, for a heap leach process and estimate the potential precious metal recovery based upon the provided metallurgical reports. The recent cyanide leach test work of five column and 45 bottle roll tests from the Wheeler and Wilson deposit provide the bulk of the metallurgical test data used, with a weighted average gold recovery value of 77%, if crushed to 80% passing $^3/_8$ inch and heap leached for 150 days. In this report, a gold recovery value of 75% was used for all modeling and mine planning. As a general rule, a feed requiring crushing to a nominal ¾ inch (19 mm) or finer will need agglomeration, even if clayey constituents are not present." (McClelland, 1988)

1.6 Environmental Studies and Permitting

Lincoln Gold has retained the services of JBR Environmental Consultants, Inc. (JBR) to assist with the environmental permitting of the Pine Grove Project.

The USFS administers exploration and mining on NFS lands under mining regulations defined in Chapter 36 of the Code of Federal Regulations, Part 228, Subpart A (36 CFR 228 Subpart A). In accordance with 36 CFR 228 Subpart A, future exploration and mining on the project unpatented claims will require Lincoln Gold to submit a Plan of Operations (PoO) for review by the USFS, Bridgeport Ranger District. The PoO will include the activities proposed on the unpatented and patented claims, and will serve as an overall plan for the entire project. Following their review, the USFS will determine whether an Environmental Assessment (EA) or an Environmental Impact Statement (EIS) is required for compliance with the National Environmental Policy Act of 1969 (NEPA). Preliminary discussions with the Bridgeport Ranger District indicate that an EA will likely be required for the project. Since the EA will analyze the activities proposed in the PoO, the NEPA analysis will include the activities proposed on the unpatented claims and the activities occurring or proposed on the patent claims. The anticipated timeline for completion of an EA is 9 to 12 months after development of the PoO.

A full discussion of environmental considerations and permitting activities may be found in Section 20 of this report. At this time, JBR does not foresee any sensitive plant or wildlife constraints. JBR also does not anticipate any reason why other required permits cannot be obtained.

1.7 Project Economics

A detailed economic analysis was completed for the Pine Grove Project utilizing information from Cost Mine for labor rates and some capital items, quotations from vendors and Telesto experience with projects of a similar size and nature. This Preliminary Economic Assessment indicates free cash flow total of $23 million dollars at a gold price of $1425. This total included all pre-production costs, capital, operating costs and such items as royalties and the Nevada Net Proceeds tax. The operating cash cost per ounce is $843. The economics indicate that this project deserves further study.

The reader is reminded that this PEA is preliminary in nature, and is based on technical and economic assumptions which will be evaluated in more advanced studies. The PEA is based on the Project resource model which consists of material in Measured, Indicated and Inferred classifications. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in this PEA will be realized. The PEA results are only intended as an initial, first-pass review of the potential project economics based on preliminary information.

1.8 Annual Gold Production

The current conceptual mine plan calls for a total mine life of 6 years: one year of pre-production (Year 0), four years of active mining with associated gold production, and one year of post-mining rinse down of the heap leach pad with residual gold production. **Table 1.3** shows the conceptual gold production by year.

Table 1.3: Conceptual Gold Production by Year

Year	Gold Produced (troy ounces)
Year 0 (pre-production)	0
Year 1	26,124
Year 2	23,831
Year 3	27,293
Year 4	23,375
Year 5	0
Total	**100,623**

2.0 INTRODUCTION

On June 14, 2011, Lincoln engaged Telesto Nevada, Inc. (Telesto) to undertake the preparation of a NI 43-101-compliant Preliminary Economic Assessment (PEA) for the Pine Grove property in Nevada, USA. The work by Telesto consisted of reviewing historical reports prepared by earlier workers/companies on the project, preparing a resource model that includes all of Lincoln's drilling results, performing a preliminary economic analysis and offering interpretations and conclusions in this report. This report has been prepared with the guidelines provided in NI 43-101 Standards of Disclosure for Mineral Projects, and conforms to Form 43-101F1 for technical reports. The Resource and Reserves definitions are as set forth in the Appendix to Companion Policy 43-101CP, CIM – Definitions Adopted by CIM Council, June 30, 2011.

2.1 Sources of Information

The sources of information include data and reports supplied by Lincoln personnel and documents referenced in Section 27. The contractor team used its experience to determine if the information from previous reports was suitable for inclusion in this report and adjusted information that required amending. Revisions to previous data were based on research, recalculations, and information from other projects. The level of detail utilized was appropriate for this level of study.

This preliminary economic assessment is based on the following sources of information:

- Personal inspection of the Pine Grove site and surrounding area
- Technical information provided by Lincoln through various reports
- Drill hole and assay data collected by Lincoln and previous owners
- Technical and cost information provided by Lincoln and other vendors and suppliers
- Technical and economic information subsequently developed by the contractor team
- Information provided by other experts with specific knowledge and expertise in their fields as described in Section 3 of this report, Reliance on Other Experts
- Additional information obtained from public domain sources.

The qualified persons responsible for the preparation of this Report are:

Table 2.1: Qualified Persons Areas of Responsibility

QP Name	Company	Qualification	Site Visit Date	Area of Responsibility
Patricia Maloney	Telesto	SME RM	31 October 2011	Sections 1.0, 1.2, 1.3, 1.6, 1.7, 1.8, 2, 3, 5, 16, 19, 20, 21, 22, 25.5, 26.2, 26.4, 26.7 and 27.
John D. Welsh	Telesto	P.E.	15 June 2011	Sections 17.1, 17.2, 17.3, 17.4, and 18
Douglas W. Willis	Telesto	C.P.G	15 June 2011 31 October 2011	Sections 1.1, 4, 6, 7, 8, 9, 10, 11, 12, 23, 24, 25.1, 25.4, 26.1, 26.3 and 26.5.
Randall Martin	Telesto	SME RM	None	Sections 1.4, 14, 15, and 25.2.
Thom Seal	Differential Engineering Inc.	PhD, P.E.	15 June 2011	Sections 1.5, 13, 17.5, 25.3 and 26.6.

Contributors John Welsh, Douglas Willis, and Thom Seal visited the Pine Grove Project area on June 15, 2011. Telesto personnel were accompanied by Jeffrey Wilson and Micheal Attaway of Lincoln. On October 31, 2011, Patricia Maloney and Douglas Willis visited the site for the duration of one day. Randall Martin has not visited the site, but has been informed by Douglas Willis on the geological conditions and controls to mineralization at Pine Grove.

This report has been prepared using data obtained from field observations taken during a site visit, drillhole logs and assay certificates which were supplied by Lincoln, and from data obtained from numerous prior reports, as detailed throughout this report.

2.2 Terms of Reference and Units of Measure

Both imperial units (American System) and Metric units are used in this report. All monetary values are in U.S. dollars ($) unless otherwise noted. Tonnages are reported in short tons (2,000 lbs).

2.2.1 Acronyms and Abbreviations

A.A.: Atomic Absorption
ABA: Acid-base accounting
ADR: Adsorption, Desorption and Refining
Ag: Silver
AMC: Antecedent Moisture Condition
ASTM: American Society for Testing and Materials
Au: Gold
BAPC: Bureau of Air Pollution Control
BLM: United States Bureau of Land Management
BMRR: Bureau of Mining Regulation and Reclamation
BWPC: Bureau of Water Pollution Control
CAA: Clean Air Act
CEQ: Council on Environmental Quality
cf: cubic foot or cubic feet
cfm: cubic feet per minute
CFO: Chief Financial Officer
CIC: Carbon-in-Column
C.P.G.: Certified Professional Geologist
CPT: Corrugated polyethylene tubing
Cu: Copper
CWA: Clean Water Act
EA: Environmental Assessment
EIS: Environmental Impact Statement
ESA: Endangered Species Act
G & A: General and Administrative
gpm: Gallons per minute
HDPE: High-density polyethylene
HLP: Heap leach pad
HMI: Human machine interfaces
HOA: Hand-off-auto
I.D.: Inside diameter
IBC: International Building Code
JBR: JBR Environmental Consultants, Inc.
kt: Kilo tons = 1,000 tons
kV: Kilovolts = 1,000 volts
kVA: Kilovolt Ampere = 1,000 volt-amperes
kWh: Kilowatt-hour = 1,000 watt-hours
LCRS: Leachate Collection and Removal System
MCC: Motor Control Center
MCP: Motor Circuit Protector
MDB&M: Mount Diablo Base and Meridian
MSHA: U.S. Department of Labor Mine Safety and Health Administration
MVA: Megavolt Ampere = 1,000,000 volt-amperes
MWMP: Meteoric Water Mobility Procedure

NAC: Nevada Administrative Code
NaCN: Sodium cyanide
NAGPRA: Native American Graves Protection and Repatriation Act
NDEP: Nevada Division of Environmental Protection
NDOW: Nevada Department of Wildlife
NEPA: National Environmental Policy Act of 1969
NFPA: National Fire Protection Association
NFS: National Forest System
NHPA: National Historic Preservation Act
NI 43-101: Canadian Institute of Mining's National Instrument 43-101
NNHP: Nevada Natural Heritage Program
NOI: Notice of Intent
NPDES: National Pollutant Discharge Elimination System
NRS: Nevada Revised Statutes
NSR: Net smelter return
opt: Troy ounces per short ton
oz: Troy ounces
P.E.: Professional Engineer
pcf: Pound-force per Cubic Foot (unit of material density)
PCMS: Process component monitoring system
PLC: Programmable logic controller
PoO: Plan of Operations
ppb: Parts per billion
ppm: Parts per million
psi: Per square inch
QA/QC: Quality Assurance and Quality Control
RC: Reverse circulation
ROM: Run-of-mine
RTU: Remote terminal unit
sf: Square foot or square feet
SHPO: Nevada State Historical Preservation Office
SPCC: Spill Prevention, Control, and Countermeasure
SRCE: Nevada Standardized Reclamation Cost Estimator
SUP: Special Use Permit
SWPPP: Storm Water Pollution Prevention Plan
Teck: Teck Resources
ton: Dry short ton of 2,000 pounds
T: Metric ton = tonne = 1,000 kg
µm: micron
USD: U.S. Dollars
USFS: United States Forest Service
USFWS: United States Fish and Wildlife Service

2.2.2 Unit Conversion Factors

1 ounce (oz) [troy] = 31.1034768 grams (g)
1 short ton = 0.90718474 metric tonnes
1 troy ounce per short ton = 34.2857 grams per metric tonne = 34.2857 ppm
1 gram per metric tonne = 0.0292 troy ounces per short ton

1 foot (ft) = 0.3048 meters (m)
1 mile (mi) = 1.6093 kilometers (km) = 5280 feet
1 meter = 39.370 inches (in) = 3.28083 feet
1 kilometer = 0.621371 miles = 3280 feet

1 acre (ac) = 0.4047 hectares
1 square kilometer (sq km) = 247.1 acres = 100 hectares = 0.3861 square miles
1 square miles (sq mi) = 640 acres = 258.99 hectares = 2.59 square kilometers

Degrees Fahrenheit (^{o}F) – 32 x 5/9 = Degrees Celsius (^{o}C)

3.0 RELIANCE ON OTHER EXPERTS

The Pine Grove PEA relies on reports and statements from legal and technical experts who are not Qualified Persons as defined by NI 43-101. The Qualified Persons responsible for preparation of this report have reviewed the information and conclusions provided and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this report.

Telesto made no attempt to verify the ownership documents or title to the Pine Grove Project. We have relied on the information contained in *G.I.S. Land Services, 2010, Pine Grove Title Review, Lode Patents and Lode, Placer and Millsite Claims, Lyon County, Nevada, Prepared for Lincoln Gold US Corp. NI 43-101 Executive Summary, Report 2010-24-TR*. This information is referenced in Section 4 of the Pine Grove PEA.

Lincoln Gold U.S. Corporation (Lincoln Gold) has retained the services of JBR Environmental Consultants, Inc. (JBR) to assist with the environmental permitting of the Pine Grove Project. Telesto has no expertise in this area and has utilized information prepared by JBR for insertion into Section 20 of this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Pine Grove Project, which encompasses approximately 7.2 square miles (18.1 square kilometers) of mineral rights, is located in Lyon County, County, Nevada, about 80 miles southeast of Reno, Nevada. The approximate center of the project area is latitude 38° 40’ 43” N, longitude -119° 7’ 7” W. Elevation of the project ranges from approximately 5,680 to 7,870 feet. The general location is depicted in **Figure 1.1**. Figure 5.1 is a more detailed property location map.

The Project area lies in the sections listed in **Table 4.1** (See **Figure 4.1**).

Table 4.1: Township, Range and Sections of the Pine Grove Project

Section (s)	Township	Range
36	10 North	25 East
31, 32, 33	10 North	26 East
1	9 North	25 East
5, 6	9 North	26 East
Mount Diablo Baseline and Meridian (MDB&M)		

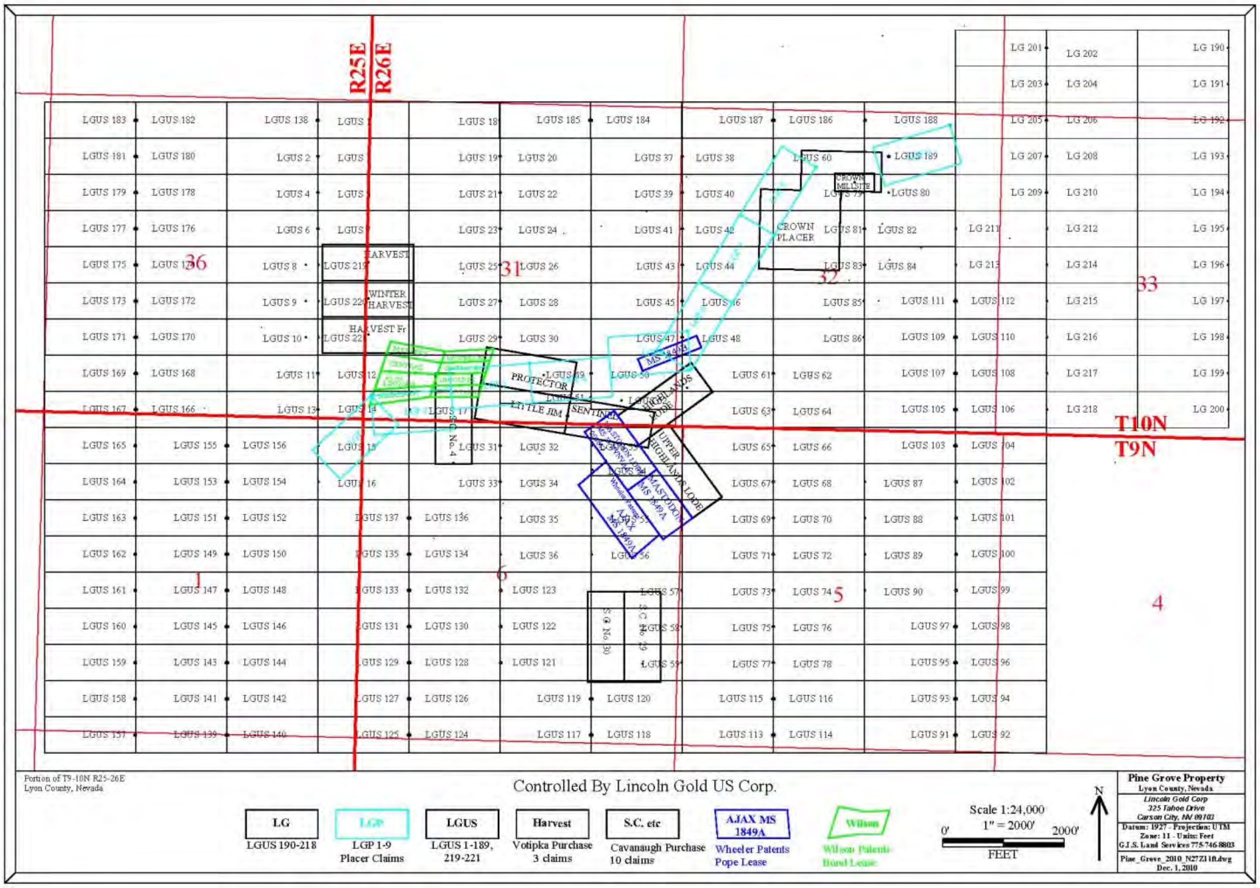


Figure 4.1: Lincoln Gold US Claim Map for Pine Grove

4.2 Environmental Studies and Permitting

The project consists of several patented and numerous unpatented mining claims. The unpatented claims are located on USFS land and therefore any proposed mining activities will be subject to Federal land use regulations as well as State of Nevada environmental regulations. Although the Pine Grove property is located in a very dry area of Nevada and has a mining history, environmental considerations that will need to be addressed in future applications for operating permits include an evaluation of potential impacts on these key resources:

- Air
- Water
- Biological
  - o Threatened and Endangered Species
- Impacts on conflicting land usage
- Cultural resources

Other evaluations needed will include potential impacts on: wild horses, existing grazing allotments, water rights, Native Americans, wilderness areas (if present), and community impacts through a special use permit in Lyon County.

A full discussion of environmental studies and permitting activities may be found in Section 20 of this report. At this time, JBR does not foresee any sensitive plant or wildlife constraints. JBR also does not anticipate any reason why other required permits cannot be obtained.

4.3 Ownership

This discussion of Lincoln's property holdings at Pine Grove refers to certain legal issues and proceedings. The authors are not qualified persons with respect to legal matters. Telesto believes that Lincoln's property holdings are as stated herein, but this is not a legal opinion.

4.3.1 Mineral Rights

The USFS controls all of the land around the Pine Grove Project. The 12 patented claims cover 88.45 acres (35.8 hectares) and are shown on **Figure 4.1**.

A report on the status of the mining claims at Pine Grove was completed on November 25, 2010 by the G.I.S. Land Services of Reno, Nevada (G.I.S. Land Services, 2010). A list of the core Lincoln claims is shown in **Table 4.2**.

Table 4.2: Summary List of Lincoln's Unpatented Mining Claims

Name of Claim	Location Date	BLM Serial No.	BLM Filing Date	Lyon County Document No.	County Filing Date
Cavanaugh Claim Group					
Highlands	10/08/2003	NMC 858438	12/22/2003	311369	01/05/2004
Upper Highlands	10/08/2003	NMC 858439	12/22/2003	311370	01/05/2004
Little Jim	04/24/2004	NMC 868934	05/25/2004	321313	04/24/2004
Protector	05/01/2004	NMC 868933	05/25/2004	321312	05/24/2004
Sentinel	04/24/2004	NMC 868935	05/25/2004	321314	04/24/2004
Southern Cross No. 4	04/24/2004	NMC 868936	05/25/2004	321315	04/24/2004
Southern Cross No. 29	09/18/2004	NMC 880068	10/19/2004	333515	10/18/2004
Southern Cross No. 30	10/08/2003	NMC 858437	12/22/2003	308146	11/12/2003
Crown Placer	04/25/2006	NMC 927125	05/30/2006	383018	05/26/2006
Crown Millsite	10/08/2003	NMC 858436	12/22/2003	308144	11/12/2003
"Harvest" Lode Claims					
Harvest	09/17/1998	NMC 793071	10/14/1998	223746	10/12/2010
Harvest Fraction	01/10/1999	NMC 800356	02/05/1999	228692	10/12/2010
Winter Harvest	01/10/1999	NMC 800355	02/05/1999	228692	10/12/2010
"LGUS" Lode Claims					
LGUS 1	05/01/2010	NMC 1024429	06/18/2010	460609	10/12/2010
LGUS 2	05/01/2010	NMC 1024430	06/18/2010	460610	10/12/2010
LGUS 3	05/01/2010	NMC 1024431	06/18/2010	460611	10/12/2010
LGUS 4	05/01/2010	NMC 1024432	06/18/2010	460612	10/12/2010
LGUS 5	05/01/2010	NMC 1024433	06/18/2010	460613	10/12/2010
LGUS 6	05/01/2010	NMC 1024434	06/18/2010	460614	10/12/2010
LGUS 7	05/01/2010	NMC 1024435	06/18/2010	460615	10/12/2010
LGUS 8	05/01/2010	NMC 1024436	06/18/2010	460616	10/12/2010
LGUS 9	05/01/2010	NMC 1024437	06/18/2010	460617	10/12/2010
LGUS 10	05/01/2010	NMC 1024438	06/18/2010	460618	10/12/2010
LGUS 11	05/01/2010	NMC 1024439	06/18/2010	460619	10/12/2010

Table 4.2: Summary List of Lincoln's Unpatented Mining Claims

Name of Claim	Location Date	BLM Serial No.	BLM Filing Date	Lyon County Document No.	County Filing Date
LGUS 12	05/01/2010	NMC 1024440	06/18/2010	460620	10/12/2010
LGUS 13	05/01/2010	NMC 1024441	06/18/2010	460621	10/12/2010
LGUS 14	05/01/2010	NMC 1024442	06/18/2010	460622	10/12/2010
LGUS 15	05/01/2010	NMC 1024443	06/18/2010	460623	10/12/2010
LGUS 16	05/01/2010	NMC 1024444	06/18/2010	460624	10/12/2010
LGUS 17	05/02/2010	NMC 1024445	06/18/2010	460625	10/12/2010
LGUS 18	05/02/2010	NMC 1024446	06/18/2010	460626	10/12/2010
LGUS 19	05/02/2010	NMC 1024447	06/18/2010	460627	10/12/2010
LGUS 20	05/02/2010	NMC 1024448	06/18/2010	460628	10/12/2010
LGUS 21	05/02/2010	NMC 1024449	06/18/2010	460629	10/12/2010
LGUS 22	05/02/2010	NMC 1024450	06/18/2010	460630	10/12/2010
LGUS 23	05/02/2010	NMC 1024451	06/18/2010	460631	10/12/2010
LGUS 24	05/02/2010	NMC 1024452	06/18/2010	460632	10/12/2010
LGUS 25	05/02/2010	NMC 1024453	06/18/2010	460633	10/12/2010
LGUS 26	05/02/2010	NMC 1024454	06/18/2010	460634	10/12/2010
LGUS 27	05/02/2010	NMC 1024455	06/18/2010	460635	10/12/2010
LGUS 28	05/02/2010	NMC 1024456	06/18/2010	460636	10/12/2010
LGUS 29	05/02/2010	NMC 1024457	06/18/2010	460637	10/12/2010
LGUS 30	05/02/2010	NMC 1024458	06/18/2010	460638	10/12/2010
LGUS 31	04/30/2010	NMC 1024459	06/18/2010	460639	10/12/2010
LGUS 32	04/30/2010	NMC 1024460	06/18/2010	460640	10/12/2010
LGUS 33	04/30/2010	NMC 1024461	06/18/2010	460641	10/12/2010
LGUS 34	04/30/2010	NMC 1024462	06/18/2010	460642	10/12/2010
LGUS 35	04/30/2010	NMC 1024463	06/18/2010	460643	10/12/2010
LGUS 36	04/30/2010	NMC 1024464	06/18/2010	460644	10/12/2010
LGUS 37	05/02/2010	NMC 1024465	06/18/2010	460645	10/12/2010
LGUS 38	05/02/2010	NMC 1024466	06/18/2010	460646	10/12/2010
LGUS 39	05/02/2010	NMC 1024467	06/18/2010	460647	10/12/2010
LGUS 40	05/02/2010	NMC 1024468	06/18/2010	460648	10/12/2010
LGUS 41	05/02/2010	NMC 1024469	06/18/2010	460649	10/12/2010
LGUS 42	05/02/2010	NMC 1024470	06/18/2010	460650	10/12/2010
LGUS 43	05/02/2010	NMC 1024471	06/18/2010	460651	10/12/2010
LGUS 44	05/02/2010	NMC 1024472	06/18/2010	460652	10/12/2010
LGUS 45	05/02/2010	NMC 1024473	06/18/2010	460653	10/12/2010
LGUS 46	05/02/2010	NMC 1024474	06/18/2010	460654	10/12/2010
LGUS 47	05/02/2010	NMC 1024475	06/18/2010	460655	10/12/2010
LGUS 48	05/02/2010	NMC 1024476	06/18/2010	460656	10/12/2010
LGUS 49	05/03/2010	NMC 1024477	06/18/2010	460657	10/12/2010
LGUS 50	05/03/2010	NMC 1024478	06/18/2010	460658	10/12/2010
LGUS 51	05/03/2010	NMC 1024479	06/18/2010	460659	10/12/2010
LGUS 52	05/03/2010	NMC 1024480	06/18/2010	460660	10/12/2010
LGUS 53	05/02/2010	NMC 1024481	06/18/2010	460661	10/12/2010
LGUS 54	05/02/2010	NMC 1024482	06/18/2010	460662	10/12/2010
LGUS 54 Amended*	7/02/2010			461539	
LGUS 55	05/02/2010	NMC 1024483	06/18/2010	460663	10/12/2010

Table 4.2: Summary List of Lincoln's Unpatented Mining Claims

Name of Claim	Location Date	BLM Serial No.	BLM Filing Date	Lyon County Document No.	County Filing Date
LGUS 56	05/02/2010	NMC 1024484	06/18/2010	460664	10/12/2010
LGUS 57	05/02/2010	NMC 1024485	06/18/2010	460665	10/12/2010
LGUS 58	05/02/2010	NMC 1024486	06/18/2010	460666	10/12/2010
LGUS 59	05/02/2010	NMC 1024487	06/18/2010	460667	10/12/2010
LGUS 60	05/03/2010	NMC 1024488	06/18/2010	460668	10/12/2010
LGUS 61	05/03/2010	NMC 1024489	06/18/2010	460669	10/12/2010
LGUS 62	05/03/2010	NMC 1024490	06/18/2010	460670	10/12/2010
LGUS 63	05/03/2010	NMC 1024491	06/18/2010	460671	10/12/2010
LGUS 63 Amended*	7/02/2010			461540	
LGUS 64	05/03/2010	NMC 1024492	06/18/2010	460672	10/12/2010
LGUS 65	05/03/2010	NMC 1024493	06/18/2010	460673	10/12/2010
LGUS 65 Amended*	7/02/2010			461541	
LGUS 66	05/03/2010	NMC 1024494	06/18/2010	460674	10/12/2010
LGUS 67	05/03/2010	NMC 1024495	06/18/2010	460675	10/12/2010
LGUS 67 Amended*	7/02/2010			461542	
LGUS 68	05/03/2010	NMC 1024496	06/18/2010	460676	10/12/2010
LGUS 69	05/03/2010	NMC 1024497	06/18/2010	460677	10/12/2010
LGUS 70	05/03/2010	NMC 1024498	06/18/2010	460678	10/12/2010
LGUS 71	05/03/2010	NMC 1024499	06/18/2010	460679	10/12/2010
LGUS 72	05/03/2010	NMC 1024500	06/18/2010	460680	10/12/2010
LGUS 73	05/03/2010	NMC 1024501	06/18/2010	460681	10/12/2010
LGUS 74	05/03/2010	NMC 1024502	06/18/2010	460682	10/12/2010
LGUS 75	05/03/2010	NMC 1024503	06/18/2010	460683	10/12/2010
LGUS 76	05/03/2010	NMC 1024504	06/18/2010	460684	10/12/2010
LGUS 77	05/03/2010	NMC 1024505	06/18/2010	460685	10/12/2010
LGUS 78	05/03/2010	NMC 1024506	06/18/2010	460686	10/12/2010
LGUS 79	05/02/2010	NMC 1024507	06/18/2010	460687	10/12/2010
LGUS 80	05/02/2010	NMC 1024508	06/18/2010	460688	10/12/2010
LGUS 81	05/02/2010	NMC 1024509	06/18/2010	460689	10/12/2010
LGUS 82	05/02/2010	NMC 1024510	06/18/2010	460690	10/12/2010
LGUS 83	05/02/2010	NMC 1024511	06/18/2010	460691	10/12/2010
LGUS 84	05/02/2010	NMC 1024512	06/18/2010	460692	10/12/2010
LGUS 85	05/02/2010	NMC 1024513	06/18/2010	460693	10/12/2010
LGUS 86	05/02/2010	NMC 1024514	06/18/2010	460694	10/12/2010
LGUS 87	05/03/2010	NMC 1024515	06/18/2010	460695	10/12/2010
LGUS 88	05/03/2010	NMC 1024516	06/18/2010	460696	10/12/2010
LGUS 89	05/03/2010	NMC 1024517	06/18/2010	460697	10/12/2010
LGUS 90	05/03/2010	NMC 1024518	06/18/2010	460698	10/12/2010
LGUS 91	05/03/2010	NMC 1024519	06/18/2010	460699	10/12/2010
LGUS 92	05/03/2010	NMC 1024520	06/18/2010	460700	10/12/2010
LGUS 93	05/03/2010	NMC 1024521	06/18/2010	460701	10/12/2010
LGUS 94	05/03/2010	NMC 1024522	06/18/2010	460702	10/12/2010
LGUS 95	05/03/2010	NMC 1024523	06/18/2010	460703	10/12/2010
LGUS 96	05/03/2010	NMC 1024524	06/18/2010	460704	10/12/2010
LGUS 97	05/03/2010	NMC 1024525	06/18/2010	460705	10/12/2010

Table 4.2: Summary List of Lincoln's Unpatented Mining Claims

Name of Claim	Location Date	BLM Serial No.	BLM Filing Date	Lyon County Document No.	County Filing Date
LGUS 98	05/03/2010	NMC 1024526	06/18/2010	460706	10/12/2010
LGUS 99	05/03/2010	NMC 1024527	06/18/2010	460707	10/12/2010
LGUS 100	05/03/2010	NMC 1024528	06/18/2010	460708	10/12/2010
LGUS 101	05/03/2010	NMC 1024529	06/18/2010	460709	10/12/2010
LGUS 102	05/03/2010	NMC 1024530	06/18/2010	460710	10/12/2010
LGUS 103	05/03/2010	NMC 1024531	06/18/2010	460711	10/12/2010
LGUS 104	05/03/2010	NMC 1024532	06/18/2010	460712	10/12/2010
LGUS 105	05/03/2010	NMC 1024533	06/18/2010	460713	10/12/2010
LGUS 106	05/03/2010	NMC 1024534	06/18/2010	460714	10/12/2010
LGUS 107	05/03/2010	NMC 1024535	06/18/2010	460715	10/12/2010
LGUS 108	05/03/2010	NMC 1024536	06/18/2010	460716	10/12/2010
LGUS 109	05/03/2010	NMC 1024537	06/18/2010	460717	10/12/2010
LGUS 110	05/03/2010	NMC 1024538	06/18/2010	460718	10/12/2010
LGUS 111	05/03/2010	NMC 1024539	06/18/2010	460719	10/12/2010
LGUS 112	05/03/2010	NMC 1024540	06/18/2010	460720	10/12/2010
LGUS 113	05/03/2010	NMC 1024541	06/18/2010	460721	10/12/2010
LGUS 114	05/03/2010	NMC 1024542	06/18/2010	460722	10/12/2010
LGUS 115	05/03/2010	NMC 1024543	06/18/2010	460723	10/12/2010
LGUS 116	05/03/2010	NMC 1024544	06/18/2010	460724	10/12/2010
LGUS 117	05/02/2010	NMC 1024545	06/18/2010	460725	10/12/2010
LGUS 118	05/02/2010	NMC 1024546	06/18/2010	460726	10/12/2010
LGUS 119	05/02/2010	NMC 1024547	06/18/2010	460727	10/12/2010
LGUS 120	05/02/2010	NMC 1024548	06/18/2010	460728	10/12/2010
LGUS 121	04/30/2010	NMC 1024549	06/18/2010	460729	10/12/2010
LGUS 122	04/30/2010	NMC 1024550	06/18/2010	460730	10/12/2010
LGUS 123	04/30/2010	NMC 1024551	06/18/2010	460731	10/12/2010
LGUS 124	04/30/2010	NMC 1024552	06/18/2010	460732	10/12/2010
LGUS 125	04/30/2010	NMC 1024553	06/18/2010	460733	10/12/2010
LGUS 126	04/30/2010	NMC 1024554	06/18/2010	460734	10/12/2010
LGUS 127	04/30/2010	NMC 1024555	06/18/2010	460735	10/12/2010
LGUS 128	04/30/2010	NMC 1024556	06/18/2010	460736	10/12/2010
LGUS 129	04/30/2010	NMC 1024557	06/18/2010	460737	10/12/2010
LGUS 130	04/30/2010	NMC 1024558	06/18/2010	460738	10/12/2010
LGUS 131	04/30/2010	NMC 1024559	06/18/2010	460739	10/12/2010
LGUS 132	04/30/2010	NMC 1024560	06/18/2010	460740	10/12/2010
LGUS 133	04/30/2010	NMC 1024561	06/18/2010	460741	10/12/2010
LGUS 133 Amended*	7/02/2010			461543	
LGUS 134	04/30/2010	NMC 1024562	06/18/2010	460742	10/12/2010
LGUS 135	04/30/2010	NMC 1024563	06/18/2010	460743	10/12/2010
LGUS 136	04/30/2010	NMC 1024564	06/18/2010	460744	10/12/2010
LGUS 137	04/30/2010	NMC 1024565	06/18/2010	460745	10/12/2010
LGUS 138	05/01/2010	NMC 1024566	06/18/2010	460746	10/12/2010
LGUS 139	05/01/2010	NMC 1024567	06/18/2010	460747	10/12/2010
LGUS 140	05/01/2010	NMC 1024568	06/18/2010	460748	10/12/2010
LGUS 141	05/01/2010	NMC 1024569	06/18/2010	460749	10/12/2010

Table 4.2: Summary List of Lincoln's Unpatented Mining Claims

Name of Claim	Location Date	BLM Serial No.	BLM Filing Date	Lyon County Document No.	County Filing Date
LGUS 142	05/01/2010	NMC 1024570	06/18/2010	460750	10/12/2010
LGUS 143	05/01/2010	NMC 1024571	06/18/2010	460751	10/12/2010
LGUS 144	05/01/2010	NMC 1024572	06/18/2010	460752	10/12/2010
LGUS 145	05/01/2010	NMC 1024573	06/18/2010	460753	10/12/2010
LGUS 146	05/01/2010	NMC 1024574	06/18/2010	460754	10/12/2010
LGUS 147	05/01/2010	NMC 1024575	06/18/2010	460755	10/12/2010
LGUS 148	05/01/2010	NMC 1024576	06/18/2010	460756	10/12/2010
LGUS 149	05/01/2010	NMC 1024577	06/18/2010	460757	10/12/2010
LGUS 150	05/01/2010	NMC 1024578	06/18/2010	460758	10/12/2010
LGUS 151	05/01/2010	NMC 1024579	06/18/2010	460759	10/12/2010
LGUS 152	05/01/2010	NMC 1024580	06/18/2010	460760	10/12/2010
LGUS 153	05/01/2010	NMC 1024581	06/18/2010	460761	10/12/2010
LGUS 154	05/01/2010	NMC 1024582	06/18/2010	460762	10/12/2010
LGUS 155	05/01/2010	NMC 1024583	06/18/2010	460763	10/12/2010
LGUS 156	05/01/2010	NMC 1024584	06/18/2010	460764	10/12/2010
LGUS 157	05/01/2010	NMC 1024585	06/18/2010	460765	10/12/2010
LGUS 158	05/01/2010	NMC 1024586	06/18/2010	460766	10/12/2010
LGUS 159	05/01/2010	NMC 1024587	06/18/2010	460767	10/12/2010
LGUS 160	05/01/2010	NMC 1024588	06/18/2010	460768	10/12/2010
LGUS 161	05/01/2010	NMC 1024589	06/18/2010	460769	10/12/2010
LGUS 162	05/01/2010	NMC 1024590	06/18/2010	460770	10/12/2010
LGUS 163	05/01/2010	NMC 1024591	06/18/2010	460771	10/12/2010
LGUS 164	05/01/2010	NMC 1024592	06/18/2010	460772	10/12/2010
LGUS 165	05/01/2010	NMC 1024593	06/18/2010	460773	10/12/2010
LGUS 166	05/01/2010	NMC 1024594	06/18/2010	460774	10/12/2010
LGUS 167	05/01/2010	NMC 1024595	06/18/2010	460775	10/12/2010
LGUS 168	05/01/2010	NMC 1024596	06/18/2010	460776	10/12/2010
LGUS 169	05/01/2010	NMC 1024597	06/18/2010	460777	10/12/2010
LGUS 170	05/01/2010	NMC 1024598	06/18/2010	460778	10/12/2010
LGUS 171	05/01/2010	NMC 1024599	06/18/2010	460779	10/12/2010
LGUS 172	05/01/2010	NMC 1024600	06/18/2010	460780	10/12/2010
LGUS 173	05/01/2010	NMC 1024601	06/18/2010	460781	10/12/2010
LGUS 174	05/01/2010	NMC 1024602	06/18/2010	460782	10/12/2010
LGUS 175	05/01/2010	NMC 1024603	06/18/2010	460783	10/12/2010
LGUS 176	05/01/2010	NMC 1024604	06/18/2010	460784	10/12/2010
LGUS 177	05/01/2010	NMC 1024605	06/18/2010	460785	10/12/2010
LGUS 178	05/01/2010	NMC 1024606	06/18/2010	460786	10/12/2010
LGUS 179	05/01/2010	NMC 1024607	06/18/2010	460787	10/12/2010
LGUS 180	05/01/2010	NMC 1024608	06/18/2010	460788	10/12/2010
LGUS 181	05/01/2010	NMC 1024609	06/18/2010	460789	10/12/2010
LGUS 182	05/01/2010	NMC 1024610	06/18/2010	460790	10/12/2010
LGUS 183	05/01/2010	NMC 1024611	06/18/2010	460791	10/12/2010
LGUS 184	05/02/2010	NMC 1024612	06/18/2010	460792	10/12/2010
LGUS 185	05/02/2010	NMC 1024613	06/18/2010	460793	10/12/2010
LGUS 186	05/03/2010	NMC 1024614	06/18/2010	460794	10/12/2010

Table 4.2: Summary List of Lincoln's Unpatented Mining Claims

Name of Claim	Location Date	BLM Serial No.	BLM Filing Date	Lyon County Document No.	County Filing Date
LGUS 187	05/03/2010	NMC 1024615	06/18/2010	460795	10/12/2010
LGUS 188	05/03/2010	NMC 1024616	06/18/2010	460796	10/12/2010
LGUS 189	05/02/2010	NMC 1024617	06/18/2010	460797	10/12/2010
LGUS 219	05/18/2010	NMC 1024618	06/18/2010	460798	10/12/2010
LGUS 220	05/18/2010	NMC 1024619	06/18/2010	460799	10/12/2010
LGUS 221	05/18/2010	NMC 1024620	06/18/2010	460800	10/12/2010
"LG" Lode Claims					
LG 190	10/12/2009	NMC 1011622	11/02/2009	450440	10/12/2010
LG 191	10/12/2009	NMC 1011623	11/02/2009	450441	10/12/2010
LG 192	10/12/2009	NMC 1011624	11/02/2009	450442	10/12/2010
LG 193	10/12/2009	NMC 1011625	11/02/2009	450443	10/12/2010
LG 194	10/12/2009	NMC 1011626	11/02/2009	450444	10/12/2010
LG 195	10/12/2009	NMC 1011627	11/02/2009	450445	10/12/2010
LG 196	10/12/2009	NMC 1011628	11/02/2009	450446	10/12/2010
LG 197	10/12/2009	NMC 1011629	11/02/2009	450447	10/12/2010
LG 198	10/12/2009	NMC 1011630	11/02/2009	450448	10/12/2010
LG 199	10/12/2009	NMC 1011631	11/02/2009	450449	10/12/2010
LG 200	10/12/2009	NMC 1011632	11/02/2009	450450	10/12/2010
LG 201	10/12/2009	NMC 1011633	11/02/2009	450451	10/12/2010
LG 202	10/12/2009	NMC 1011634	11/02/2009	450452	10/12/2010
LG 203	10/12/2009	NMC 1011635	11/02/2009	450453	10/12/2010
LG 204	10/12/2009	NMC 1011636	11/02/2009	450454	10/12/2010
LG 205	10/12/2009	NMC 1011637	11/02/2009	450455	10/12/2010
LG 206	10/12/2009	NMC 1011638	11/02/2009	450456	10/12/2010
LG 207	10/12/2009	NMC 1011639	11/02/2009	450457	10/12/2010
LG 208	10/12/2009	NMC 1011640	11/02/2009	450458	10/12/2010
LG 209	10/12/2009	NMC 1011641	11/02/2009	450459	10/12/2010
LG 210	10/12/2009	NMC 1011642	11/02/2009	450460	10/12/2010
LG 211	10/12/2009	NMC 1011643	11/02/2009	450461	10/12/2010
LG 212	10/12/2009	NMC 1011644	11/02/2009	450462	10/12/2010
LG 213	10/12/2009	NMC 1011645	11/02/2009	450463	10/12/2010
LG 214	10/12/2009	NMC 1011646	11/02/2009	450464	10/12/2010
LG 215	10/12/2009	NMC 1011647	11/02/2009	450465	10/12/2010
LG 216	10/12/2009	NMC 1011648	11/02/2009	450466	10/12/2010
LG 217	10/12/2009	NMC 1011649	11/02/2009	450467	10/12/2010
LG 218	10/12/2009	NMC 1011650	11/02/2009	450468	10/12/2010
"LGP" Placer Claims					
LGP 1	09/01/2010	NMC 1029177	11/15/2010	467863	11/15/2010
LGP 2	08/26/2010	NMC 1029178	11/15/2010	467864	11/15/2010
LPG 3	09/01/2010	NMC 1029179	11/15/2010	467865	11/15/2010
LGP 4	09/01/2010	NMC 1029180	11/15/2010	467866	11/15/2010
LGP 5	09/01/2010	NMC 1029181	11/15/2010	467867	11/15/2010
LGP 6	09/01/2010	NMC 1029182	11/15/2010	467868	11/15/2010
LGP 7	09/01/2010	NMC 1029183	11/15/2010	467869	11/15/2010
LGP 8	09/01/2010	NMC 1029184	11/15/2010	467870	11/15/2010

Table 4.2: Summary List of Lincoln's Unpatented Mining Claims

Name of Claim	Location Date	BLM Serial No.	BLM Filing Date	Lyon County Document No.	County Filing Date
LGP 9	09/01/2010	NMC 1029185	11/15/2010	467871	11/15/2010

Note: Table 4.2 is adapted from Table 4-1 in Tetra Tech (2011)

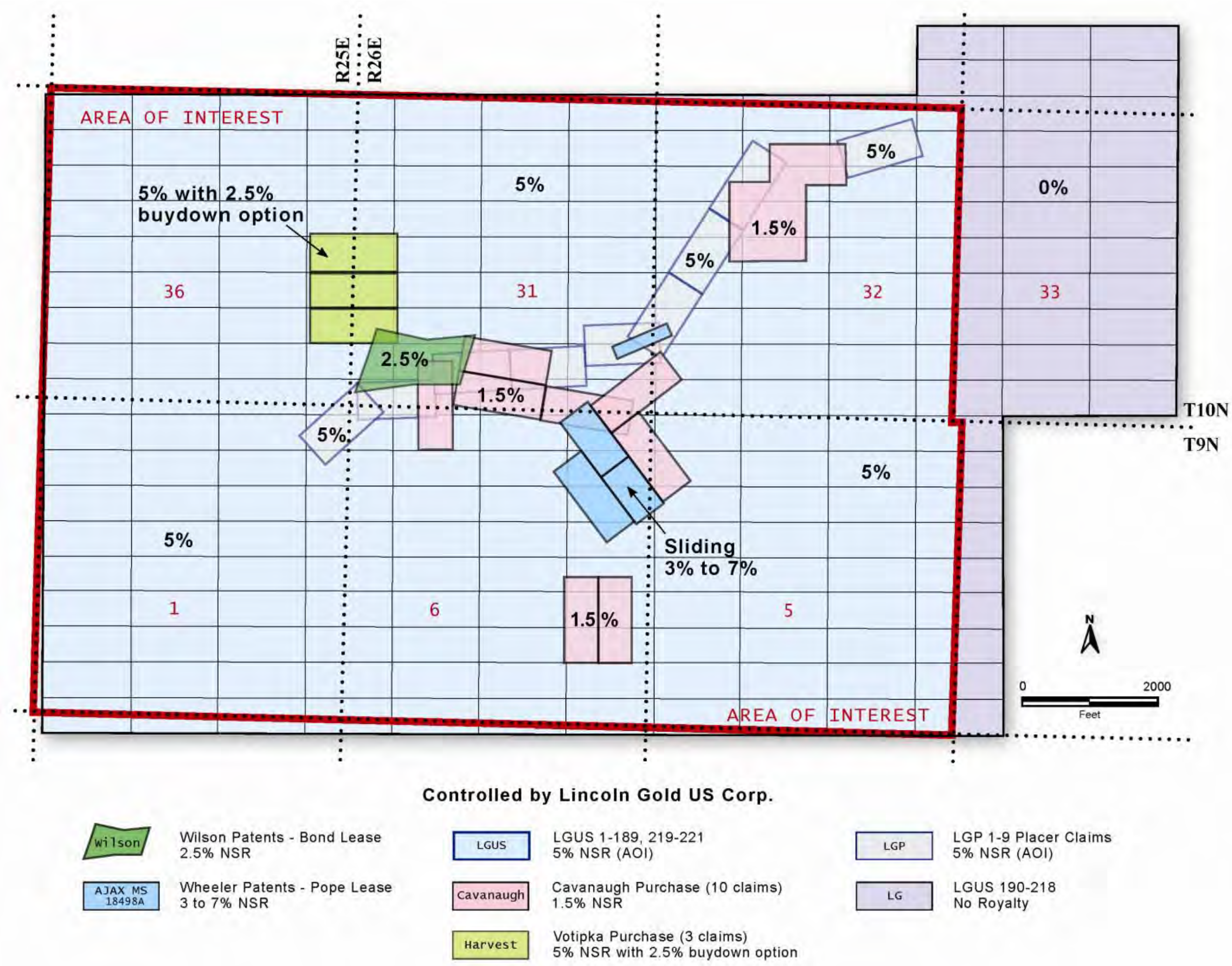
Lincoln also controls 12 patented mining claims. The patented claims encompass 88.45 acres (35.8 hectares), for a project total of ±4,586 acres (1,856 hectares). The patented claims, which are listed in **Table 4.3**, are shown on **Figure 4.2**.

Table 4.3: Summary of Patented Claims Held Under Lincoln Mining Lease

Patented Claim Name	Patent No.	Mineral Survey No.	Acres	Lyon Co. Tax ID No.	Owner
Wheeler Patents					
Ajax	#32624 06/02/1900	MS 1849A	20.09	012-521-01	Wheeler Mining Company
Mastodon		MS 1849A	13.32	012-521-01	
Mastodon Lode		MS 37/1696	16.48	012-521-01	
Wheeler Millsite		MS 1849B	5.01	012-501-02	
		Subtotal:	54.90		
Wilson Patents					
Mystery	#37585 12/11/1903	MS 1953	4.29	012-521-01	Lyon Grove LLC
Mystery 1st E. Ext.		MS 1953	3.48	012-521-01	
Central		MS 1953	5.12	012-521-01	
Central 1st E. Ext.		MS 1953	3.47	012-521-01	
Lincoln		MS 1953	5.11	012-521-01	
Lincoln 1st E. Ext.		MS 1953	3.48	012-521-01	
Himalaya		MS 1953	5.08	012-521-01	
Himalaya 1st E. Ext.		MS 1953	3.52	012-521-01	
		Subtotal:	33.55		
		Grand Total:	**88.45**		

Note: Table 4.3 is adapted from TABLE 4-2 in Tetra Tech (2011)

Telesto's preliminary review of current claim ownership at Pine Grove using the U.S. Bureau of Land Management's (BLM) LR-2000 online database system indicates that, as of the effective date of this report, all of the claims listed herein are valid and in good standing in regards to federal claim maintenance fee requirements. A search of Lyon County, Nevada records was not performed. See Section 4.2 for a more detailed discussion of mineral rights and ownership.



Note: Figure 4.2 is adapted from Figure 4-3 in Tetra Tech (2011).

Figure 4.2: Pine Grove Project Claim Blocks and Royalty Status

4.4 Terms of Agreement Between Lincoln and Other Entities

4.4.1 Terms of Agreement with Wheeler Mining Company (Wheeler Patented Claims)

From Tetra Tech (2011):

Lincoln leases the Wheeler patented claims from the Wheeler Mining Company ("Wheeler Mining") through a mining lease option agreement dated July 13, 2007 and effective through December 31, 2022, with an option to renew for additional successive terms. The terms of this agreement included advance royalty payments of $10,000 in the first year and $30,000 per year in subsequent years, along with a sliding scale NSR royalty ranging from 3% at a gold price of $450 to 7% at a gold price exceeding $700. The agreement also stipulated that Lincoln would use its best efforts to produce a positive feasibility study within 24 months of the date of the agreement, but by subsequent agreement dated January 2, 2009, the parties extended the deadline to three months after all permits have been received but no later than December 31, 2010.

Lincoln has since received an extension from Wheeler Mining to allow for negotiation concerning the issue of a Preliminary Economic Assessment.

4.4.2 Terms of Agreement with Lyon Grove LLC (Wilson Patented Claims)

From Tetra Tech (2011):

Lincoln leases the Wilson patented claims from Lyon Grove, LLC through a mining lease-option agreement dated August 1, 2007. The initial term is 15 years with the right to extend the term for up to 10 additional one-year extensions. The terms of this agreement included advance royalty payments of $10,000 in the first year and $25,000 per year in subsequent years, along with a sliding scale NSR royalty ranging from 3% at a gold price of $450 to 7% at a gold price exceeding $700. The agreement provides that the owner may require the Lessee to purchase the property for $1,000 at any time after applications have been made to permit and develop a mine on the property. There is also an annual work commitment that now stands at $50,000.

The agreement includes a 6 square mile Area of Interest that includes a 5% NSR royalty on any new claims put into production within the following area:

- *All of Section 36, T10N, R25E*
- *All of Section 1, T9N, R25E*
- *All of Section 31, T10N, R26E*
- *All of Section 32, T10N, R26E*
- *All of Section 5, T9N, R26E*
- *All of Section 6, T9N, R26E*

The original agreement was amended effective July 21, 2010 to reduce the sliding scale NSR to a fixed 2.5% on the Wilson patented claims. The 5% NSR on the area of interest was modified to exclude the Harvest claims, Cavanaugh claims, and Wheeler patented claims. The amendment required the payment of US$300,000 in two equal payments and the issuance of 500,000 common shares of Lincoln. The first payment of US$150,000 and the issuance of all 500,000 shares was completed in 2010. A single payment of US$150,000 is due to Lyon Grove, LLC on the effective date in 2011.

4.4.3 Terms of Agreement with Cavanaugh (Cavanaugh Claim Group)

Effective August 23, 2010, Lincoln purchased 100% of ten unpatented claims (eight lodes, one placer, one millsite) from the Estelle D. Cavanaugh Trust and Lynn R. Shelley (“Cavanaugh”) of Newbury Park, CA, whereby Cavanaugh retains a fixed 1.5% NSR production royalty on the ten claims (See **Table 4.4**). The purchase agreement requires Lincoln to make payments totaling US$650,000 and the issuance of 400,000 common shares of Lincoln over a period of three years. In 2010, Lincoln paid Cavanaugh US$250,000 and issued 150,000 shares. In 2011, Lincoln’s payment obligation is US$150,000 and 150,000 shares.

4.4.4 Terms of Agreement with Votipka (Harvest Claim Group)

From Tetra Tech (2011):

Effective September 6, 2007, Lincoln purchased three unpatented "Harvest" lode claims from Harold Votipka of Carson City, NV. The purchase price was US$12,000 and included a 5% NSR production royalty. Lincoln retains the option to buy-down up to 2.5% of the NSR royalty by paying to Votipka US$100,000 per full point.

Table 4.4: Summary of Current Royalties at Pine Grove

NSR Recipient	Property	Area	NSR	Remarks
Estelle D. Cavanaugh Trust & Lynn R. Shelley	Cavanaugh Claims	8 Lode Claims 1 Placer Claim 1 Millsite Claim	1.5%	Incorrectly reported as 2.5% by Tetra Tech (2011)
Harold Votipka	Harvest Claims	3 Lode Claims	5.0%	Buy-down option for 2.5%
Wheeler Mining Company	Wheeler Patented Claims	54.90 acres	3% to 7% sliding scale	Lincoln intends to buydown NSR
Lyon Grove, LLC	Wilson Patented Claims	33.55 acres	2.5%	
	Area of Interest	6 square miles	5.0%	Covers most Lincoln claims

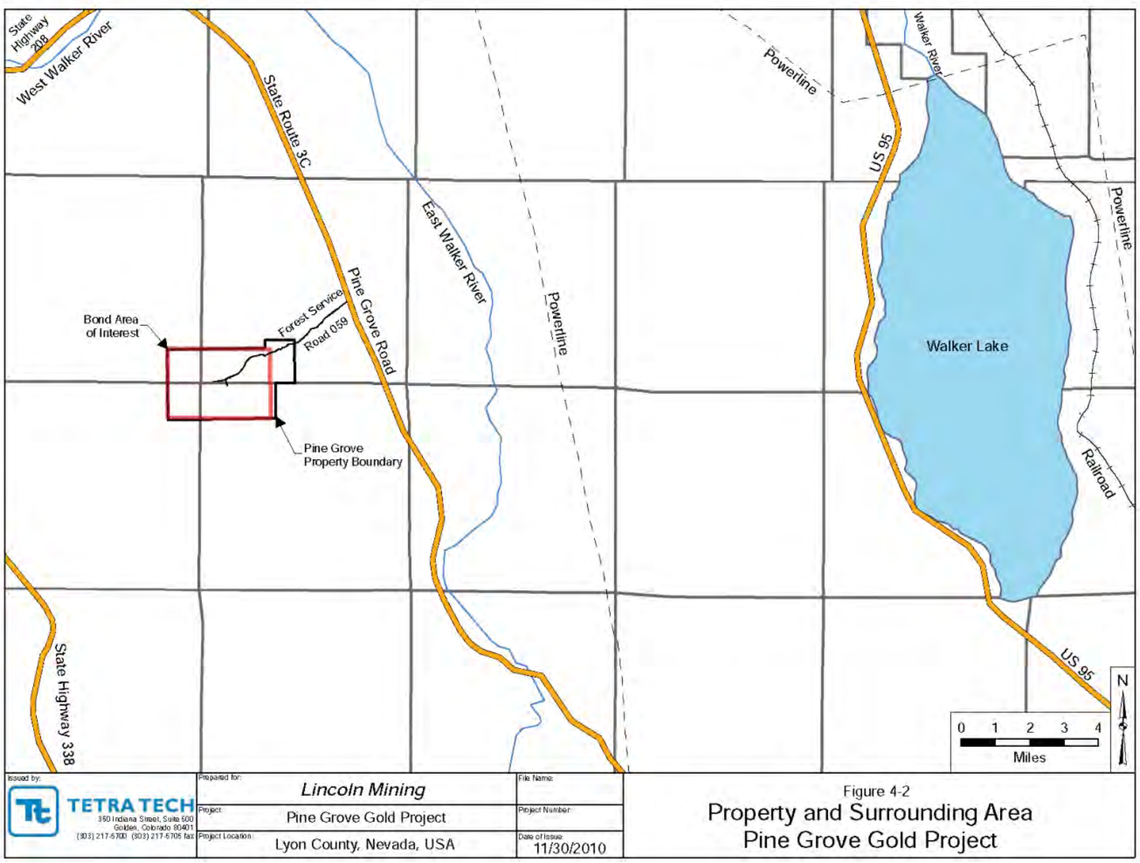
Note: Table 4.4 is adapted from TABLE 4-3 in Tetra Tech (2011).

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access to the Property

The Project is accessed via Interstate 80 by traveling approximately 33 miles east from Reno. Exit Interstate 80 at Exit 46 (U.S. Highway 95 Alternate) and turn south (right). Follow the road south and then east for approximately 1.5 miles until reaching the center of Fernley. Turn south (right) onto U.S. Highway 95 Alternate South. Continue on Highway 95 Alternate for 45 miles. Turn east (left) to stay on Highway 95 Alternate at the designated intersection. Yerington is one mile from the intersection. Turn south (right) onto N. Main Street in Yerington, which doubles as Nevada Highway 208. Stay on Nevada Highway 208 for 11 miles. Where Nevada Highway 208 makes a 90° right turn toward Smith Valley (west), continue south onto a dirt road (East Walker Road) which immediately turns southeast. East Walker Road is maintained by the county and is very well-graded. Follow the dirt road for 10 miles until reaching Pine Grove Road. Turn right (west) onto Pine Grove Road and travel approximately 4 miles to reach the Project. **Figure 5.1** shows the regional location of the Pine Grove property.

A two-wheel drive vehicle is sufficient to get up Pine Grove Road into the property; however, in order to access the property fully on the numerous drill roads, a four-wheel drive vehicle is necessary.



Note: Figure 5.1 is adapted from Figure 4-2 in Tetra Tech (2011)

Figure 5.1: Pine Grove Vicinity Map

5.2 Climate and Physiography

The Pine Grove Project lies in the Basin and Range province, a major physiographic region of the western United States. The region is typified by north-northeast trending mountain ranges separated by broad, flat, alluvium filled valleys.

The project is located in the eastern Pine Grove Hills and includes Pine Grove Canyon and a portion of Scotts Canyon (Tetra Tech, 2011). Pine Grove Canyon is an ephemeral channel and drains the majority of the area. The Pine Grove Hills trend N25°W and are a southern continuation of the Singatse Range. Both constitute a west-tilted fault block (Dircksen, 1975). The topography is generally moderate to locally steep terrain. Elevations range from about 5,680 feet on Pine Grove Creek in the northeastern part of the project area to 7,870 feet on slopes in the south-central part of the project area (JBR, 2009b). The elevation in the vicinity of the Wheeler and Wilson mines is about 6,700 feet; relief is about 500 feet (Gray, 1968).

Lyon County contains productive, irrigated farmland surrounded by high-desert terrain and produces 23% of Nevada's agricultural products. Main crops are alfalfa, onion, garlic, grains, and potatoes. Livestock production includes beef, sheep, dairy operations, and llama breeding. The great majority of the Pine Grove project area is composed of mixed pinyon-juniper woodland (JBR, 2009b).

The climate is dry, with Yerington only receiving an average of 5.07 inches of precipitation per year (from Western Regional Climate Center data, www.wrcc.dri.edu). Precipitation primarily falls between November and May each year with peak precipitation falling in January. Yerington generally does not receive any snowfall.

The hottest month of the year is July when the average daily high and low temperatures are 92.2 °F and 52.5 °F respectively. January is the coldest month, when the average daily high temperature is 46.1 °F and the average low temperature is 17.7 °F (from WRCC website).

Exploration and mining can be conducted on the property year round (Tetra Tech, 2011).

5.3 Local Resources and Infrastructure

Yerington, Nevada, is approximately 21 miles (34 kilometers) north of the Project. The population of Yerington is 3,048 according to the 2010 Census. The community of Yerington is equipped to provide housing, shopping and schools for mine personnel and their families. Skilled mining personnel are expected to be available in Yerington and from nearby communities such as Reno, Carson City, Fallon, Fernley, and Hawthorne. Reno, a city with a 200,000+ population, is 80 miles northwest of Yerington.

5.3.1 Power Supply

The region is supported by grid power and other infrastructure. In November 2010, Lincoln contracted NV Energy to evaluate the availability of electricity to the property (Tetra Tech, 2011). Telesto also contacted NV Energy in October of 2011. The nearest power lines appear to be approximately seven miles away from Pine Grove and NV Energy has no plans to construct new transmission lines that would bring grid power closer to the property. Because of this, power will be supplied the by on-site generators.

5.3.2 Water Supply

Lincoln owns three small water rights on the property. These water rights were acquired with the Cavanaugh claim group purchase. Lincoln will also source additional and alternate supplies. The writers have not verified the availability of water rights from groundwater or surface water.

Table 5.1: Pine Grove Water Rights

Application No.	Certificate No.	Date Granted	Comment
24812	9312	Feb. 1, 1979	Not to exceed 15 million gallons/year
24518	9313	Feb. 1, 1979	Not to exceed 15 million gallons/year
24520	9314	Feb. 1, 1979	Not to exceed 0.944 million gallons/year

Note: Table 5.1 is adapted from Table 5-1 from Tetra Tech (2011).

5.3.3 Transportation Facilities

Lyon County is host to a main north-south U.S. highway (US-95A), an interstate highway (I-80), and two railway lines (Union Pacific Railway). The Amtrak train which runs from San Francisco to Chicago via Salt Lake City goes through Lyon County (Tetra Tech, 2011).

5.3.4 Buildings and Ancillary Facilities

The property area offers adequate available land for project development, including several large, gently sloping sites for a processing plant site, heap leach pads or tailings ponds, as needed. In 2009, Lincoln expanded their claim block to the east to include low-relief areas that could be used for mineral processing (Tetra Tech, 2011). Lincoln also maintains a field office and warehouse in Yerington.

6.0 HISTORY

6.1 Introduction

Telesto has reviewed the section on history from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Telesto Qualified Person agrees with the description of the history of the Pine Grove Project.

The following sections (in *italics*) on history are taken from Tetra Tech (2011).

The following information on the mining and exploration history of the Pine Grove project is largely taken from an article by Jackson (1996) and from the 2007 and 2008 technical reports (Stone, 2007, 2008), with additional information provided by Lincoln and taken from other references as cited.

6.2 Exploration and Mining History

6.2.1 Pre-1930 Production History

The Pine Grove district, also referred to as the Wilson district, is a former gold-producer with several underground mines. Gold was first discovered at Pine Grove in 1866, and within a year or so, the nearby town of Pine Grove had grown to over 300 people. By the late 1880s, the district hosted three mills producing $10,000 in gold bullion each week and the town of Pine Grove grew to over 1,000 people. The two principal mines were the Wilson (FIGURE 6-1), located on the north side of Pine Grove Canyon, and the Wheeler (FIGURE 6-2), on the south side.

Historic mining at Pine Grove produced roughly 240,000 ounces in gold from selected high-grade veins (Jackson, 1996). The Wilson and Wheeler mines were largely worked by lessees (Hill, 1915b). Some 150,000 ounces were produced from the Wilson mine, with about 100,000 ounces produced by the Wheeler mine (Stone, 2007, 2008). Grades reportedly averaged 1.4 oz Au/t at Wilson (104,046 tons of ore), and 1.3 oz Au/t at Wheeler (74,531 tons of ore). During this period, some 10,000 ft of underground workings were developed, along with a number of winzes, shafts, and adits. The Wilson deposit was mined to a depth of 140 ft, whereas Wheeler was mined to a depth of 120 ft. The historic cutoff grade, estimated from the remaining pillars, appears to be on the order of 0.35 to 0.50oz/t (Jackson, 1996). McKinstry (1941b) noted that sulfide ores could not be handled in the former operations. According to Hill (1915b), prior to 1896 none of the ore was concentrated and only 33% of the precious-metals value in the sulfide ore was free milling.

The boom ended in 1887; however, sporadic mining continued until 1915, and the town of Pine Grove was eventually abandoned in 1930 to become a ghost town. The underground workings are no longer accessible, and very few maps exist showing the locations of the workings. The extent of the historic underground mining can be estimated on the basis of the volume of waste, tailings, and historic maps. A letter dated 1935 opined that the three dumps on the property contain 150,000 tons; the tailing pond *has 15,000 tons; and under the largest of the dumps is about 15,000 tons of tailings (Courtney, 1935).*

Subsequent work at Pine Grove consisted of re-processing of the old mine dumps and tailings piles. This work has continued sporadically until modern times.

6.2.2 Modern Exploration

Pine Grove was essentially idle from the turn of the 20th century until the end of the 1960s.

In 1969, Quintana Minerals of Houston, Texas, reportedly was interested in the copper potential of the property. They undertook a program of surface mapping and completed one drill hole. The results of that program are not known, and the log/assays from the one drill hole were not available to Lincoln.

In 1981, Lacana Mining Corporation of Toronto, Ontario, explored the property for gold. This work consisted primarily of surface mapping. No further details on Lacana's work program or results are available.

In 1988, the property was optioned to Teck Resources ("Teck") of Reno, Nevada, a wholly owned U.S. subsidiary of Teck Corporation of Vancouver, B. C. Teck undertook the most extensive exploration program to date. The program included detailed geologic mapping, surface and underground geochemical sampling, biogeochemical sampling, geophysical surveying, and the drilling of 160 holes for a total of 53,000 feet. The geophysical work consisted of magnetic surveying by Quantec Consulting Inc. in May 1988 (Pawluk, 1990). The survey was conducted using the Scintrex IGS total field magnetometer. Nominal line spacing was 200 feet, with 50 feet survey station intervals. Lines ran north-south. Lincoln has copies of much of the original Teck data, including rock-chip and stream-sediment sample data and assays and various Teck maps. Teck dropped their option in 1992.

Silver Standard Resources Inc. ("Silver Standard") briefly explored the property in 1994, but they too subsequently dropped their option.

Lincoln acquired the property in 2007 as described in Section 4.2. Lincoln's exploration at Pine Grove is described in Section 10.0.

6.3 Prior Resource and Reserve Estimates

A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves and the issuer is not treating the historical estimate as current mineral resources or mineral reserves. The current mineral resource estimate contained in Section 14 of this report supersedes all reported historical estimates.

Teck estimated what they called "geologic reserves," "preliminary mineable reserves," and "diluted minable reserves (20% at Zero Grade)" in 1991 (Jackson, 1991). Those calculations, using the polygonal method, are shown on TABLE 6-1 (**Table 6.1** in this report)*. The parameters included a 0.015 oz Au/t cutoff, a density factor of 13 ft^3/t, and a 10 ft minimum thickness with no more than 5 ft of internal waste; assays were cut to 0.5 oz Au/t (2.6% of Wheeler ore-grade samples, <1% of Wilson), and voids (stopes) were given zero grade. "Minable reserves" were calculated based on a preliminary pit design with a maximum pit slope of 45°. Jackson (1991) notes that no geostatistical analysis was performed on the data, so no quantitative measure of the continuity of the mineralization was known. He also noted that no density tests had been performed on the "ore." These estimates do not comply with current NI 43-101 classifications and reporting requirements, and these estimates should not be relied upon.*

Regarding the 1991 estimates, Jackson (1991) stated that tonnages removed by historic mining had not been subtracted from the reserve figures. He stated that a maximum of 75,000 tons of high-grade material (>0.5 oz Au/t) were thought to have been removed from the Wheeler, with an additional tonnage of ore-grade rock (0.015 to 0.50 oz Au/t) removed by access tunnels and haulageways. Regarding the latter, he indicated no figure for this tonnage had been calculated but that it "should not significantly affect the reserve numbers." He stated that approximately 100,000 tons of high-grade ore was taken from the Wilson mine but that the vast bulk of those operations lie outside of the proposed pit and should have little or no effect on the tonnages.

Table 6.1: Teck's 1991 "Reserves" Estimates for Wilson and Wheeler

	Short Tons	Grade (opt)	Contained Ounces
Wilson	877,154	0.055	48,179
Wheeler	1,380,028	0.065	89,897
Total	2,257,182	0.061	138,076

(From Jackson, 1991; tonnages removed by historic mining have not been subtracted from these figures.)
Note: Table 6.1 is adapted from Table 6-1 from Tetra Tech (2011).

In 1992, Teck calculated a polygonal resource estimate for gold mineralization at the Wilson and Wheeler mine areas. This estimate was based on an assay top cut of 0.496 oz Au/t, and a cutoff grade of 0.015 oz Au/t. No estimate was made of the copper resources. The 1992 Teck resource estimate pre-dates the implementation of National Instrument 43-101 and is not compliant. The published resources were not classified into Measured, Indicated and Inferred. The Teck estimate of the remaining geologic resource is shown in TABLE 6-2 (**Table 6.2** in this report)*. According to Jackson (1996), the district originally contained, including the historic production and the in situ resources, roughly 2.54 million tons at an average grade of 0.15 oz Au/t or about 390,000 oz of gold.*

Table 6.2: Teck's 1992 Resource Estimate for Wilson and Wheeler

	Short Tons	Grade (opt)	Contained Ounces
Wilson	912,250	0.055	50,174
Wheeler	1,435,250	0.065	93,290
Total	2,347,500	0.061	143,464

(From Teck, 1992; 0.015 ozAu/t cutoff)
Note: Table 6.2 is adapted from Table 6-2 from Tetra Tech (2011).

Stone (2008) also reported that the old mine dumps, considered to be un-economic during underground mining in the 1880s, were thought to contain recoverable gold, and an estimate of the mineral potential of the dumps was made in 2006 (TABLE 6-3) (**Table 6.3** in this report)*. According to Stone (2008), this estimate was not compliant with the reporting requirements of NI 43-101.*

Table 6.3: Estimated Material in Mine Dumps and Tailings at Wilson and Wheeler

	Short Tons	Grade (opt)	Contained Ounces
Wilson	80,000	0.060	4,800
Wheeler	20,000	0.060	1,200
Total	100,000	0.060	6,000

(From Stone, 2007, 2008)
Note: Table 6.3 is adapted from Table 6-3 from Tetra Tech (2011).

As part of the 2007 and 2008 technical reports, Stone (2007, 2008) prepared resource estimates for the Wilson and Wheeler deposits using a block modeling technique and based only on the historic data. ....

A Qualified Person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves and the issuer is not treating the historical estimate as current mineral resources or mineral reserves**.** The current mineral resource estimate contained in Section 14 of this report supersedes all reported historical estimates.

6.4 Historic Production

PHOTO 6-1 (**Figure 6.1** in this report) *pictures a historical marker that has been placed near the Pine Grove project site. As seen, it details what is believed to be the historic production of gold from the area in US dollars. Applying a historic gold price of US$16.00 per ounce and using the reported approximately US$8,000,000 of value (US$5,000,000 from the Wheeler Mine and US$3,000,000 from the Wilson Mine), it appears that approximately 500,000 ounces of gold were historically produced from the property in the past. However, Teck Resources reported that historic production was on the order of 240,000 ounces gold. Since none of these figures can be independently validated, they are not either neither* (sic) *CIM nor NI 43-101 compliant.*

The Qualified Person has not verified the information contained in these publically disclosed reports and therefore the information in the reports is not necessarily indicative of the mineralization on the Pine Grove Project*.*

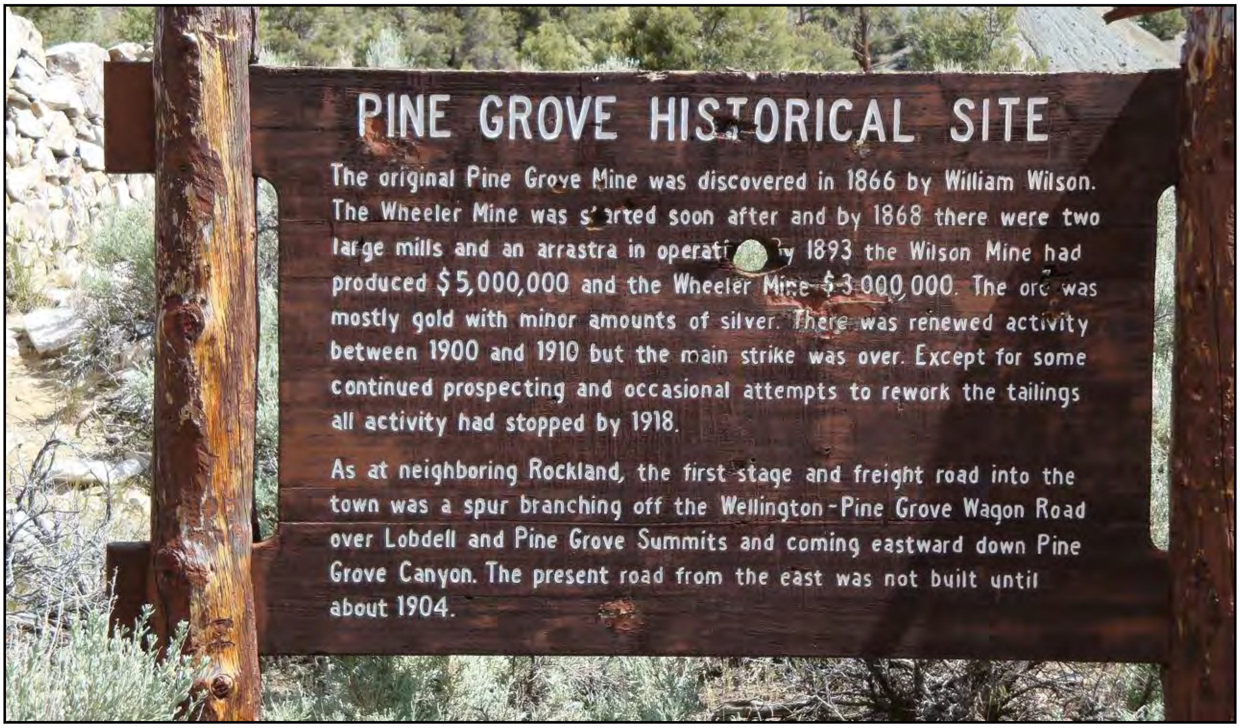


Figure 6.1: Historic Marker at the Pine Grove Project Site

6.5 Historic Reclamation

Very little, modern reclamation has occurred. Teck Resources re-contoured and re-seeded drill roads on USFS land with the exception of roads created prior to 1981. Teck did not reclaim any drill roads on the Wheeler and Wilson patented claims.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting Introduction

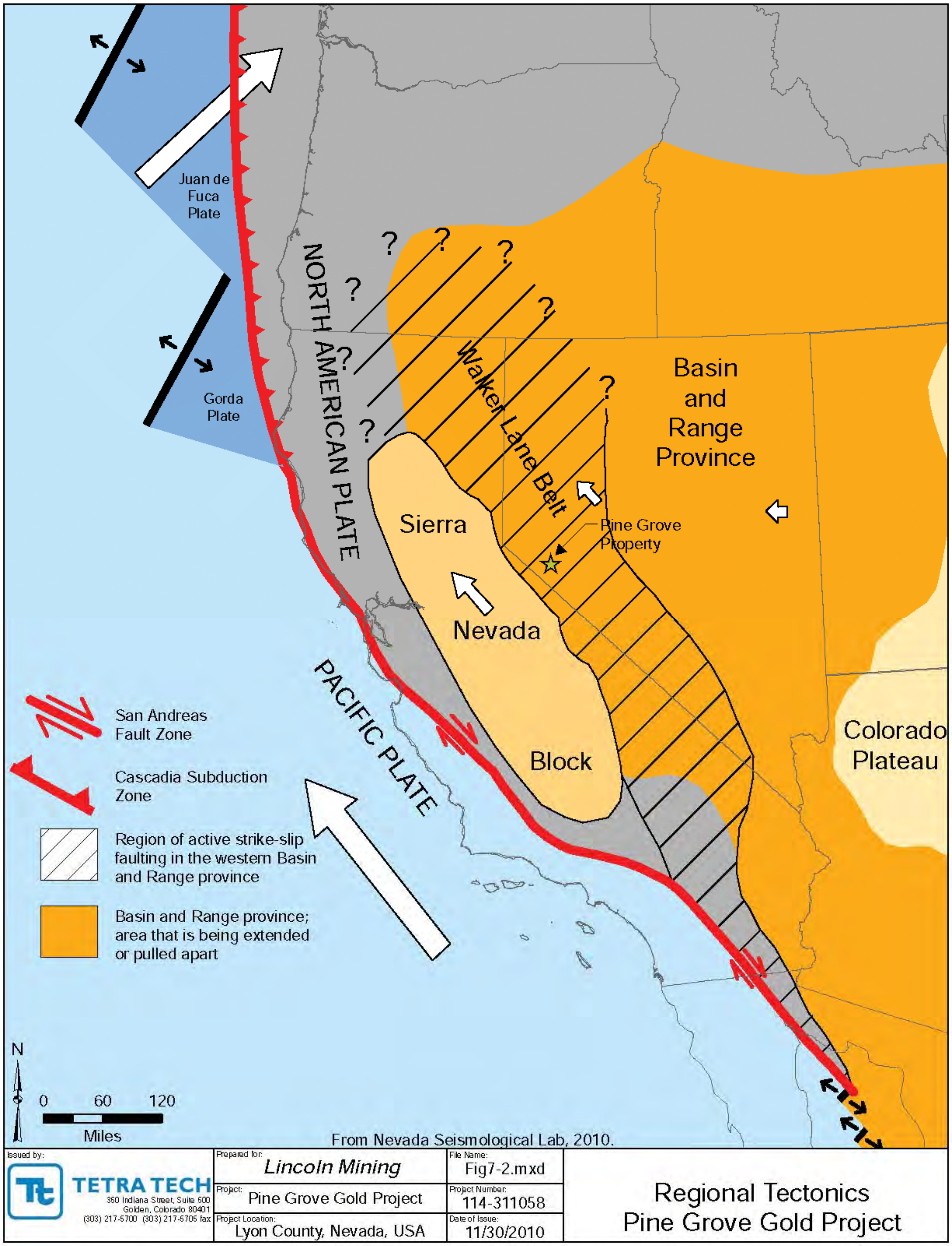
Telesto has reviewed the section on geological setting from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Telesto Qualified Person agrees with the description of the geological setting of the Pine Grove Project.

The following sections (in *italics*) on geological setting are taken from Tetra Tech (2011).

The information collected on the regional, district, and property geology has been summarized from papers by Dircksen (1975) and Princehouse (1993), a technical report by Lincoln Gold US Corp (Stone, 2007), and articles on the Pine Grove mining district in Lyon County, Nevada in Coyner and Fahey (1995).

The Pine Grove property lies within the central portion of the Walker Lane geologic province near its western margin (FIGURE 7-1) (**Figure 7.1** in this report)*. The Walker Lane is host to numerous mineral deposits including eipthermal* (sic) *gold-silver deposits related to Tertiary volcanics, sediment hosted-skarn related precious and base metal deposits and porphyry copper deposits.*

The Walker Lane is a geologic trough roughly aligned with the California/Nevada border southward to where Death Valley intersects the Garlock Fault, a major left-lateral strike-slip fault. The north-northwest end of the Walker Lane is between Pyramid Lake in Nevada and California's Mount Lassen where the Honey Lake Fault meets the transverse tectonic zone forming the southern boundary of the Modoc Plateau and Columbia Plateau provinces. The Walker Lane takes up 15 to 25 percent of the boundary motion between the Pacific Plate and the North American Plate, the other 75 percent being taken up by the San Andreas Fault system to the west. The Walker Lane may represent an incipient major transform fault zone which could replace the San Andreas as the plate boundary in the future. The Walker Lane deformation belt accommodates nearly 12 mm/yr of dextral shear between the Sierra Nevada-Great Valley Block and North America. The belt is characterized by the northwest-striking trans-current faults and co-evolutionary high-angle and low-angle dip-slip faults formed as result of a spatially segregated displacement field.



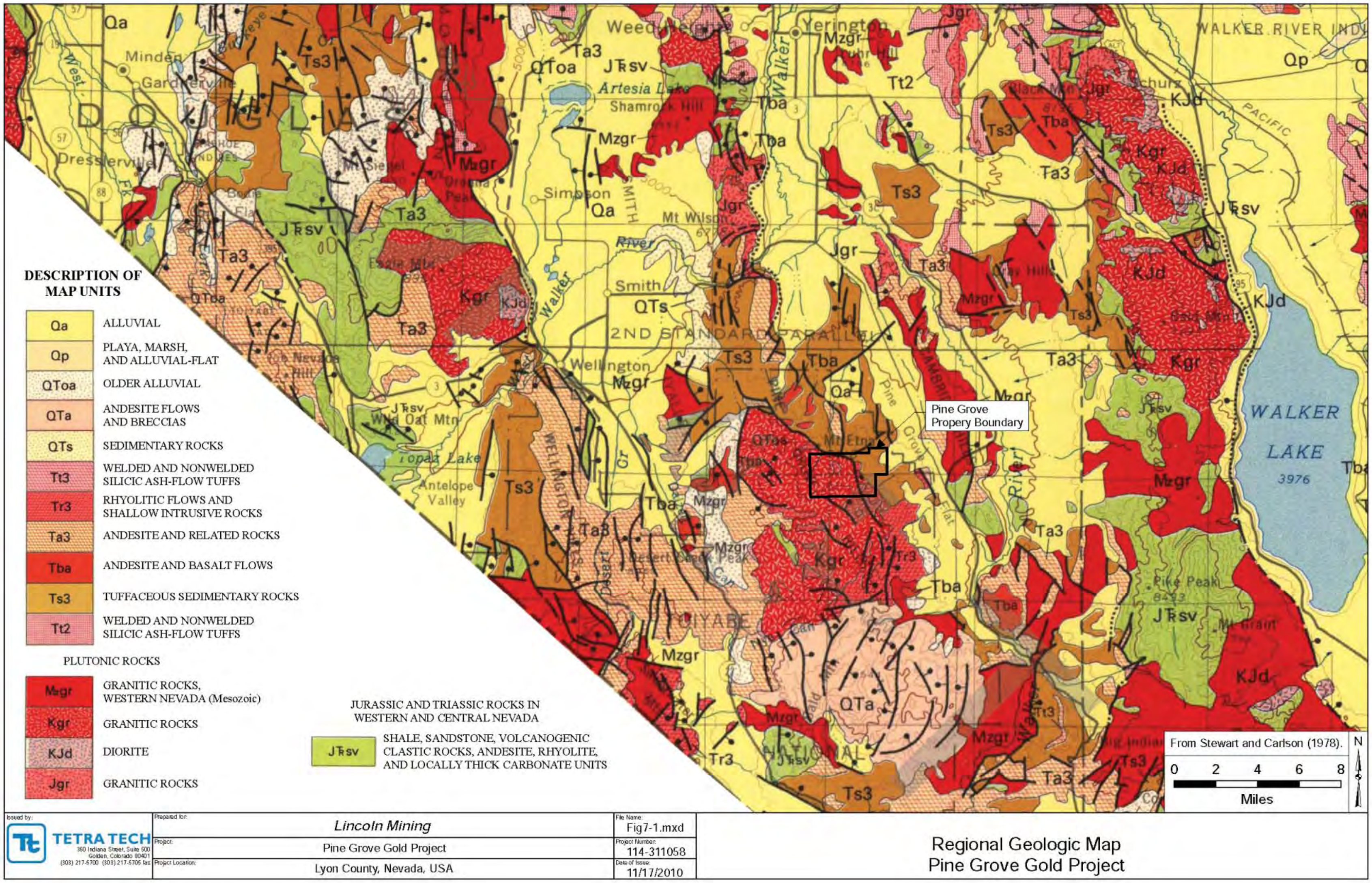
Note: Adapted from FIGURE 7-1, Tetra Tech (2011).

Figure 7.1: Regional Tectonics

7.2 Regional Geology

The oldest rocks in the region are volcanic, sedimentary and intrusive rocks of early Mesozoic age. These older rocks are part of a west-facing continental magmatic arc that extended along the western margin of the North America at the time and are now exposed along the western margin, and locally throughout the southern central portion, of the Walker Lane. These rock units have been highly deformed and metamorphosed.

In the Pine Grove region these deformed early Mesozoic rocks have been intruded by the Early Jurassic Lobdell Summit pluton, a multi-phase complex granodiorite to granitic intrusive dated at 187 Ma. The Lobdell Summit pluton is unconformably overlain by upper Tertiary and Quaternary age rocks, including Oligocene-lower Miocene silicic tuffs, Miocene andesite lavas, upper Miocene clastic sedimentary rocks with local basalt lavas. These sedimentary and volcanic rocks are part of the Wassuk Group, which includes the Morgan Ranch formation near Pine Grove. Normal faulting and extension began within the Walker Lane as early as 27 Ma and in the Pine Grove area extensional faulting started at about 12 Ma and continued through to about 7 Ma. The sedimentary and volcanic rocks that unconformably overlie the Lobdell Summit pluton were deposited in fault-bounded basins during the period of extensional tectonics of 12 Ma to 7 Ma. Intrusive rhyolite bodies, as small plugs and dikes, intruded along the high angle extensional faults at about 7.6 to 5.7 Ma. The youngest rock units in the area are pediment gravels and stream fill sands, silts and gravels of Quaternary age (FIGURE 7-2) (**Figure 7.2** in this report).



Note: Adapted from FIGURE 7-2, Tetra Tech (2011).

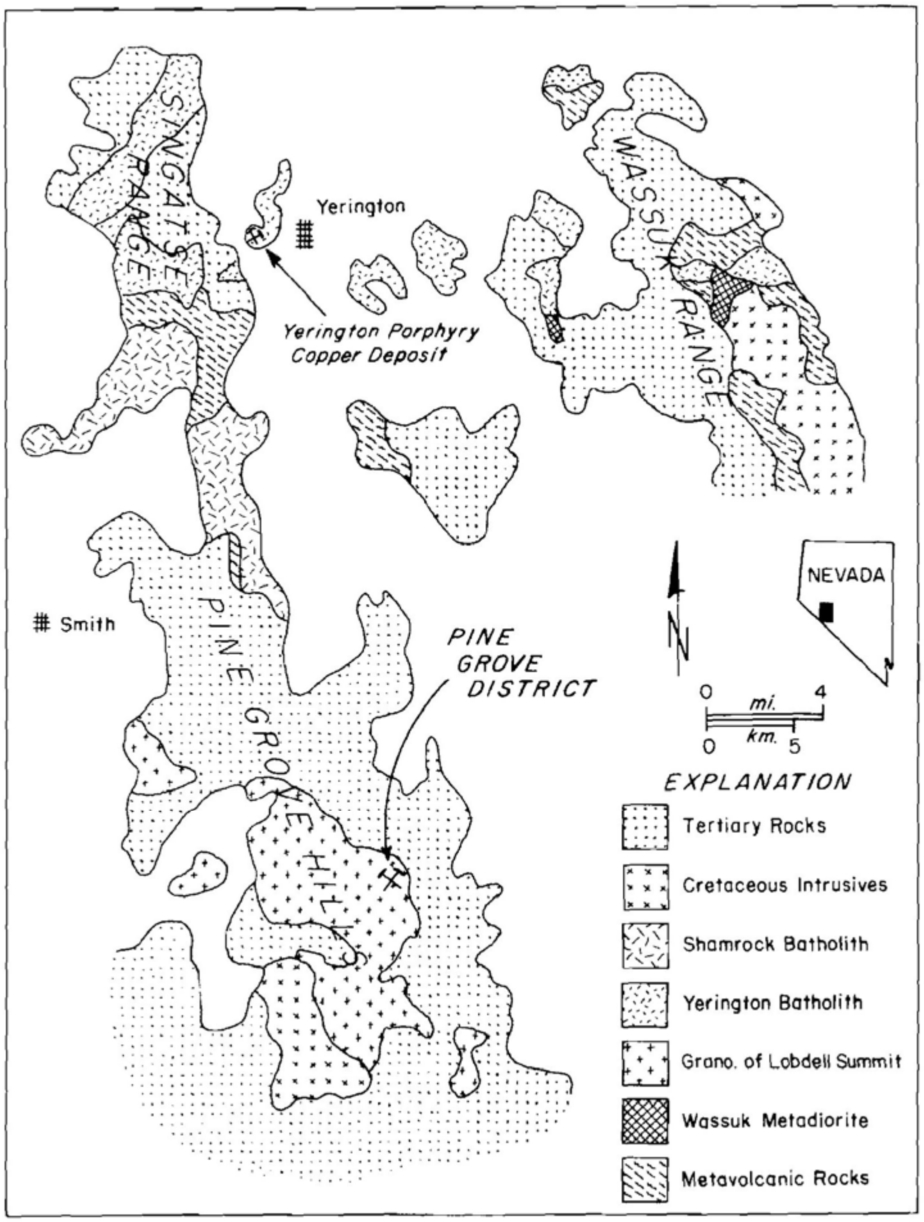
Figure 7.2: Regional Geologic Map

7.3 Local Geology

The Pine Grove project lies in the Pine Grove Hills, a north-trending, extensional fault-block mountain range of the western portion of the Basin and Range Province. The range is composed of a core of Mesozoic volcanic, sedimentary, and intrusive rocks that are in turn overlain by Tertiary sedimentary and volcanic rocks (FIGURE 7-3) (**Figure 7.3** in this report)*. In general, the Pine Grove Hills are a west-tilted fault block, bounded on the east by a series of faults, some of which transect the Pine Grove district. The northern and southern ends of the hills best demonstrate the westward tilting, but toward the center, the structure is more horst-like, with a central plateau of granitic rocks flanked to the east and west by Tertiary sedimentary and volcanic rocks (Moore, 1969). The most significant geologic feature in the Pine Grove project area is a northwest-striking, northeast-dipping normal fault that juxtaposes Mesozoic intrusive rocks in the footwall against intrusive rocks capped by Tertiary sedimentary rocks in the hanging wall. This structure, termed the Pine Grove fault, is a diffuse, 600-foot wide extensional shear zone that forms part of the eastern boundary of the Pine Grove Hills structural block. The fault originally had a steep dip but has been rotated to nearly flat by regional extension. Numerous sub-parallel dikes occur within the fault, and the structure served as the locus for mineralization in the area.*

The oldest rocks in the Pine Grove project area are metamorphosed volcanic and hypabyssal intrusive rocks of the Mesozoic metavolcanic sequence that occur as roof pendants in the Lobdell Summit granodiorite. Compositions include andesite, dacite, and fine- to medium-grained diorite. These rocks are only exposed south of Pine Grove Canyon in the hanging wall of the Pine Grove fault. The rocks are typically greenish as a result of lower greenschist grade metamorphism. Small pods of magnetite-bearing skarn developed locally in the andesitic portions of the rock.

The complex and multi-phase Granodiorite of Lobdell Summit intrusive forms the basement rock in the area and is host to all of the known mineralization. Granodiorite predominates, but quartz monzonite, monzonite, diorite, and granite phases can be found as well. Intruding the granodiorite are dikes and plugs of a leucocratic rock that was given the field term "microgranite." Dikes of microgranite a few meters or less in thickness occur west and southeast of the Wheeler mine and in the eastern portion of the Wilson mine; a larger body intermingled with granodiorite occurs north of the Wilson mine in the hanging wall of the Pine Grove fault. A thick sequence of younger (upper Miocene) Tertiary conglomerate and sedimentary breccia occurs in the hanging wall of the Pine Grove fault. The conglomerate is heterolithic, poorly sorted, and weakly indurated. Clasts range in size from less than 0.5 in. to over 20 ft in diameter and are angular to sub-rounded in a matrix of iron-stained, sand-sized particles. The most common clast lithology is "microgranite." This sequence is thought to correlate with the Morgan Ranch Formation of the Wassuk Group.



Note: Adapted from FIGURE 7-3, Tetra Tech (2011).

Figure 7.3: Local Geologic Map

Following the onset of faulting and extensional rotation, intrusions of rhyolite were emplaced along structures. Small plugs and dikes form steep, resistive outcrops in Pine Grove Canyon and follow two predominant structural orientations, a west-northwest-striking set and a north- to northeast-striking set. Conical-shaped intrusive plugs form several of the distinctive topographic features in the area, including Sugarloaf and Mt. Etna. The rock is distinctly flowbanded and often highly contorted.

There are three basic ages of structural change within the Pine Grove District; Mesozoic granitic dikes, metamorphic foliation and ductile deformation, and Tertiary brittle deformation. Mesozoic dikes are related to the hydrothermal alteration and *mineralization. The dikes strike north-northwest and were originally vertical; however, they tilted westward, giving them a dip of 30° to 40˚ east.*

The next structural change relates to the metamorphic event between 233 Ma and 169 Ma. The metamorphism aligned the igneous and hydrothermal biotite grains within the granodiorite, forming a weak foliation within the rock. The quartz in the granodiorite is commonly polygonalized, and the quartz grains in the folded quartz veins have triple junctions (Princehouse, 1993).

The last change is related to the brittle, cataclastic deformation that is characterized by faulting between 15 Ma and 7.5 Ma. The largest of the faults is the Pine Grove Fault. This fault separates the Mesozoic granitic rocks from the Tertiary Morgan Ranch formation. The fault strikes north-northwest and has a dip of 22° to 25˚ to the east, and is displaced approximately 4 miles. A poorly defined syncline was formed in the Morgan Range formation with an axis that runs parallel to the Pine Grove Fault. The formation of this syncline was most likely due to drag during faulting.

7.4 Property Geology

The Wheeler and Wilson deposits are the focus of this report and comprise the resource area. All of the mineralization found to date is hosted within the Lobdell Summit granodiorite intrusive or its associated complex dikes of rhyolite porphyry and granite porphyry. The dikes have intruded the Lobdell Summit Pluton along low-angle faults and shears sub-parallel to the Pine Grove fault. In the Wheeler Mine area the fault trends northerly and dips 15° to 35° to the east. At the Wilson Mine the Pine Grove fault is not present but a footwall splay that separates mineralized granodiorite with overlying Tertiary rhyolite strikes in a northwest to west-northwest direction and dips 15° to 35° to the northeast. The mineralized granodiorite at Wilson is separated by several tabular dikes of granite porphyry, rhyolite porphyry and dacite.

See **Figure 7.4** for a generalized map of local geology and see **Figure 7.5** for a detailed map showing property geology.

7.4.1 Lithology

<u>Lobdell Summit Granodiorite (gd)</u>

The granodiorite of Lobdell Summit is the oldest rock unit exposed in the property area. It is a green to gray-green medium-grained hornblende-biotite granodiorite containing microcline and accessory magnetite, ilmenite, titanite (sphene), epidote, allanite, apatite, and zircon. Based on textural relationships, much of the hornblende has been replaced by fine-grained biotite accompanied by epidote and locally by chlorite. This biotite forms a weak regional foliation that is interpreted to metamorphic in origin. The Lobdell Summit Pluton has been dated at 186.5±7.7 Ma and 186.9±8.5 Ma.

<u>Wheeler Granite Porphyry (Kp)</u>

The Wheeler granite porphyry is a pink to light gray with large, conspicuous pink orthoclase (up to 8mm), white plagioclase and green biotite phenocrysts. Accessory minerals include titanite, allanite and opaque minerals. It has a groundmass consisting of quartz, alkali feldspar and trace amounts of biotite. The granite porphyry occurs as low-angle dikes of 10 to 50 ft in thickness filling faults that are sub-parallel to the main Pine Grove fault trends within the Lobdell Summit intrusive. It is the oldest of the dike events and volumetrically the second most abundant.

<u>Rhyolite Porphyry (Rp)</u>

The rhyolite porphyry is light gray to pink in color and has distinct large quartz phenocrysts up to 3 mm. This porphyry has been altered and deformed to some degree. The groundmass is estimated to have been originally about equal amounts of quartz and alkali feldspar but secondary albite has totally replaced the alkali feldspar. The rhyolite porphyry occurs as low-angle dikes of 10 to 50 ft in thickness filling faults and shears in the Lobdell Summit intrusive.

The rhyolite porphyry dikes locally have distinctive glassy chilled margins, are younger than the granite porphyry and are volumetrically the most abundant.

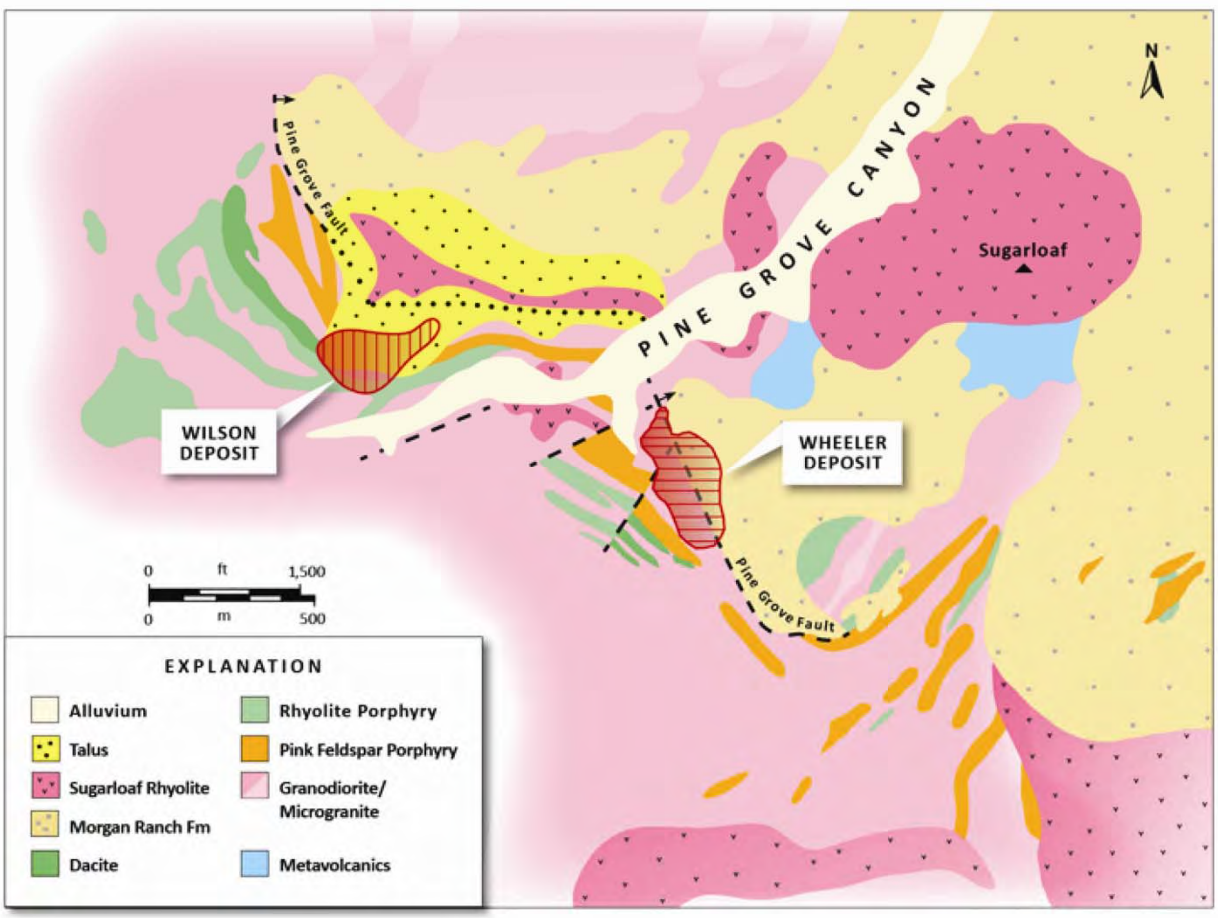
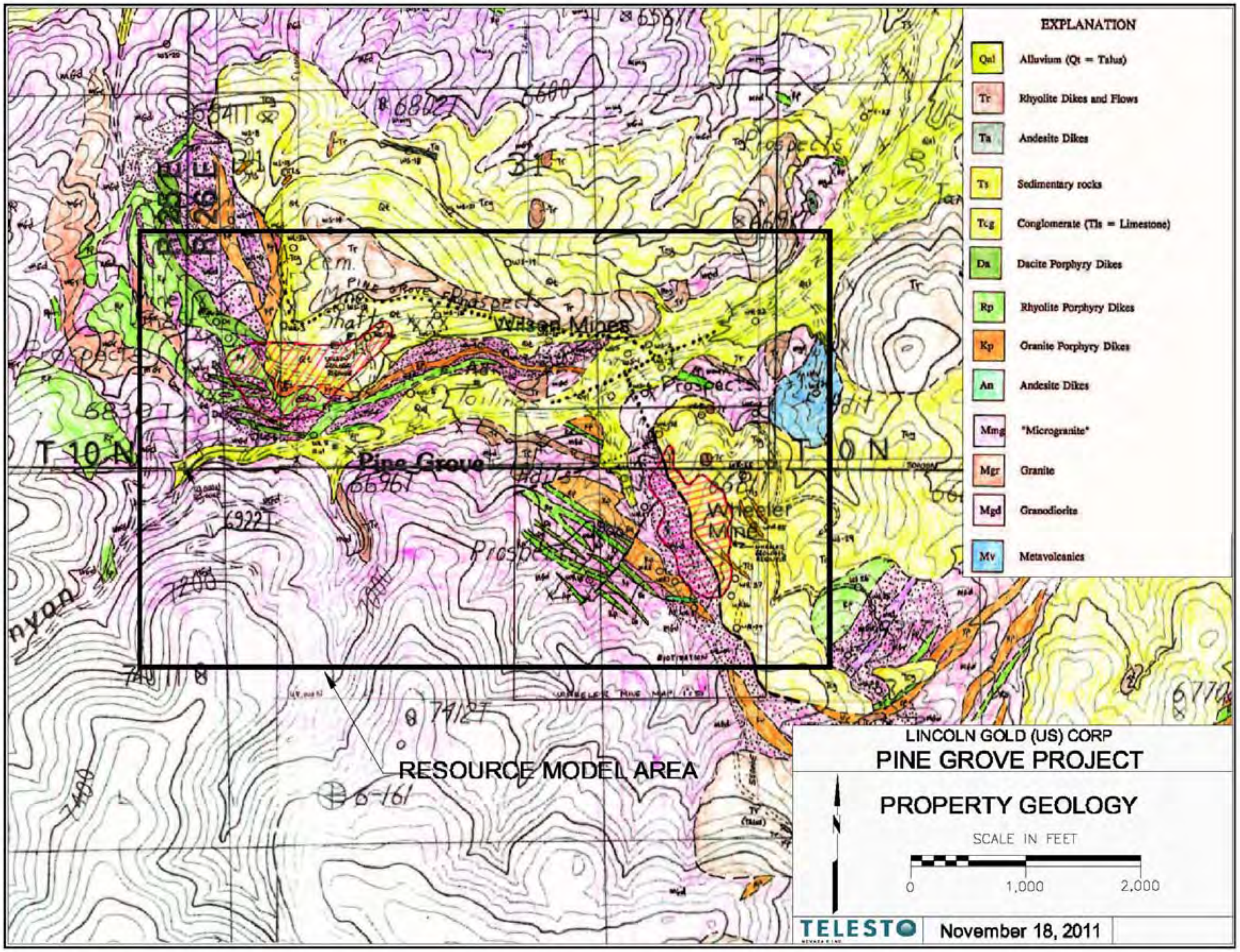


Figure 7.4: Local Geology

Figure 7.5: Property Geologic Map



Dikes of Dacite Porphyry (Da), Andesite (An) and Microgranite (Mg)

In the Wilson and Wheeler mine areas, reverse circulation drilling by Teck Resources and Lincoln has encountered only minor amounts of dacite, andesite and microgranite dikes. Andesite dikes are more common in the Wheeler area and occur as small discontinuous bodies filling low angle fractures and locally may occur along cross cutting high angle fractures. The andesite is dark green to nearly black in color owing to the abundant very fine grained biotite. These dikes are soft due to their sheared and altered nature.

The dacite dikes are more common in the Wilson area and occur as narrow bodies of 5 to 20 ft feet in thickness and have intruded along faults and fractures that are parallel to the rhyolite porphyry and granite porphyry dikes. Where seen in outcrop at the Wilson mine, the dacite is a light to medium gray color with fine-grained groundmass of feldspar and quartz and hornblende laths up to 3 mm. Locally the dacite may be irregular in shape.

The term “microgranite” has been applied to dikes and small irregular bodies of fine-grained leucocratic intrusive rocks that occur in the Wheeler area. The rock is fine-to medium-grained and equigranular with graphic intergrowths of quartz and feldspar.

Morgan Ranch Conglomerate and Breccia (Tcg)

The Morgan Ranch Formation is a thick sequence of Tertiary conglomerate and sedimentary breccia in the hanging wall of the Pine Grove fault. The conglomerate is poorly sorted and weakly indurated with angular to sub-rounded clasts up to 15 ft in size in a matrix of clay and sand. Clasts are weathered products of pre-Tertiary intrusive rocks, mainly the microgranite, granodiorite and associated dikes.

Sugarloaf Rhyolite (Tr)

Dikes and small plugs of white to red-brown flow banded rhyolite form steep, resistive outcrops in the Pine Grove Canyon. Sugarloaf Peak is a conical-shaped intrusive plug west of the Wheeler mine and forms a distinctive topographic feature in the area. The Sugarloaf rhyolite has intruded along two predominate structural trends, a west-northwest-striking set and a north to northeast-striking set.

Quaternary Alluvium (Qal) and Colluvial (Qcol) Slope Cover

The stream channel of Pine Grove Canyon is filled with alluvial deposits composed of silts, sands, cobbles and boulders derived from weathering of the various rock units in the district. The slopes of hills with gentle to moderate relief are covered with locally thick colluvial deposits of coarse bedrock fragments that are weathering in place and being moved down-slope, mainly by gravity. Pine Grove Creek contains various placer concentrations of gold derived from the Wilson and Wheeler lode deposits. These placers have been worked in the past.

7.4.2 Structure

In the vicinity of the Wheeler mine, the Pine Grove fault zone strikes N30°W and dips 25 to 35° northeast. The eastern edge of the fault zone is marked by the Pine Grove fault, which juxtaposes Tertiary conglomerate in the hanging wall against granodiorite of the footwall. According to Jackson (1996), the bulk of the displacement probably occurred along this structure. The Pine Grove fault contains several centimeters of gouge and breccia, often with slickensides, and in places hosts thin, shattered quartz veins that contain gold. Based on offset of Miocene volcanic rocks, the Wheeler fault (Pine Grove Fault) *has had about 3.75-4.37mi of normal displacement (Princehouse, 1993).*

The mineralized block of granodiorite is bounded on the west and bottom by the northwest-striking Stonehouse fault, which has been offset by northeast- and west-northwest-striking faults. Most of the significant gold mineralization in the immediate Wheeler area appears to lie within the 100 m-wide, highly sheared block between the Stonehouse and Pine Grove faults. West of the Stonehouse fault, the rocks are not sheared, except along some dike contacts, and represent deeper, less-mineralized parts of the hydrothermal system. The Stonehouse fault dips 70° northeast with no more than 75 m of displacement, according to Jackson (1996), although Princehouse (1993) reported that the Stonehouse fault had over 500 m of offset.

Northwest-striking, northeast-dipping breccia and gouge zones occur as lenses ranging from a few centimeters or less to several meters in thickness separated by blocks of sheared rock. Dips range from 40° to 75°. These gouge zones are cut by northeast-striking normal faults with displacement up to several tens of meters. There are still younger west-northwest-striking and north- to northwest-striking faults.

The originally steeply dipping, northwest-striking porphyry dikes were rotated to 30° northeast dips, with some of the westward tilt predating 15 Ma but most due to normal-oblique slip movement along the 7 Ma-old Pine Grove fault system (Princehouse, 1993; Princehouse and Dilles, 1996). The Wheeler fault (Pine Grove Fault) *has 6-7km of normal displacement; the Stonehouse fault is a footwall splay.*

Like the Wheeler deposit, the Wilson deposit is located within the Pine Grove fault zone. However, in contrast to the Wheeler deposit, Wilson's mineralization is confined to several slices of granodiorite that lie sandwiched between rhyolite porphyry and dacite dikes, below the granite porphyry dike that underlies the Wheeler mineralization. This setting for the Wilson mineralization is below that of the Wheeler deposit and corresponds to the granodiorite and dike package that lies west of the Stonehouse fault and below the Wheeler mine.

The package of dikes within the Pine Grove fault zone at Wilson strikes east and dips 0° to 15° north. The fault contact between the footwall intrusions and the hanging wall of sedimentary rocks is covered by talus shed from a Tertiary rhyolite dike on the ridge above the Wilson deposit. In contrast to Wheeler, at the Wilson mine strong biotite foliation is only sporadically developed, and evidence of brittle shearing is minimal.

7.5 Mineralization Introduction

Telesto has reviewed the section on mineralization from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Telesto Qualified Person agrees with the description of the mineralization at the Pine Grove Project. Moreover, the Qualified Person has visited the site and observed mineralization consistent with Tetra Tech's description. **Figure 7.6** presents a photograph of the Wheeler Deposit area.

The following sections (in *italics*) on the Pine Grove mineralization are taken from Tetra Tech (2011).

The following information is taken from the 2007 and 2008 technical reports (Stone, 2007, 2008), which were based on the work by Jackson (1996).

Known gold mineralization at the Pine Grove project is found at the Wheeler and the Wilson mines. The two areas show similar alteration and mineralization characteristics but differ in their structural signatures due to differing locations relative to the Pine Grove fault. Gold is found in transitional quartz veins and in thin, crosscutting pyrite-chalcopyrite stockwork veinlets; the transitional quartz veins occurred between prograde potassic and albitic alteration and retrograde sericite-pyrite-quartz alteration (Jackson, 1996). Dilles (1990) reports that sulfide mineralization is also disseminated.

7.6 Wheeler Mine

The Wheeler mine is situated in a fault-bounded block of granodiorite adjacent to the hanging wall of the Pine Grove fault at the contact with the Tertiary conglomerate and above the granite porphyry and other dikes that intrude the granodiorite. Princehouse and Dilles (1996) noted that the hydrothermal alteration and mineralization are spatially and temporally associated with the granite porphyry dikes. Gold and copper mineralization within the sheared block of granodiorite is exposed in outcrop, roadcuts, and underground workings, where it occurs with quartz veining and minor stockwork sulfide veinlets.

7.6.1 Structure

The approximately 330 ft-wide block of mineralized granodiorite is confined on the east by the hanging-wall structure of the Pine Grove fault and on the west by a parallel fault that was termed the Stonehouse fault. The Wheeler fault (Tetra Tech incorrectly identifies the Pine Grove Fault as the Wheeler Fault) *dips about 30° to the east, and the Stonehouse fault dips roughly 70° to the east. The block of mineralized granodiorite between the faults is strongly sheared and brecciated, with textures ranging from early, shallow-dipping, brittle-ductile smearing of foliated biotite to more steeply-dipping brittle, cataclastic breccia and gouge zones that parallel the Pine Grove fault.*

Note: From Photo 9-1, Tetra Tech, 2011.
Figure 7.6: Wheeler Deposit

Post-mineral shearing has disrupted the internal structure at the Wheeler mine veins system such that sizable volumes of gold-bearing gouge are typically encountered. This shearing has disrupted the veins and produced zones of crushed and pulverized material containing tiny blebs of silica that were probably once portions of discrete veins.

7.6.2 Alteration

The mineralization is accompanied by strong hydrothermal alteration that post-dated the metamorphic foliation (Jackson, 1996). In general, the alteration increases in intensity to the northeast, reaching a maximum at the contact between the granodiorite with the hanging wall Morgan Ranch conglomerate.

Hydrothermal alteration consists of early, prograde, high-temperature potassic alteration (biotization and potassium feldspar replacement), followed by an albititc (sic) *alteration event, then a transitional chlorite-actinolite event that hosts the gold mineralization. The chlorite-actinolite alteration is confined to the mineralized block between the Stonehouse and Pine Grove faults. Mineralization was followed by retrograde quartz-sericite-pyrite alteration. The alteration events are telescoped and overlap each other, and for the most part are restricted to the mineralized block of granodiorite.*

Jackson (1996) reports that much of the mineralized rock at Wheeler is steel bluish-gray in color and, in places, is very hard with bluish, glittery chalcedonic coatings on fractures. This alteration is not well studied but may be the result of a silica-clay event.

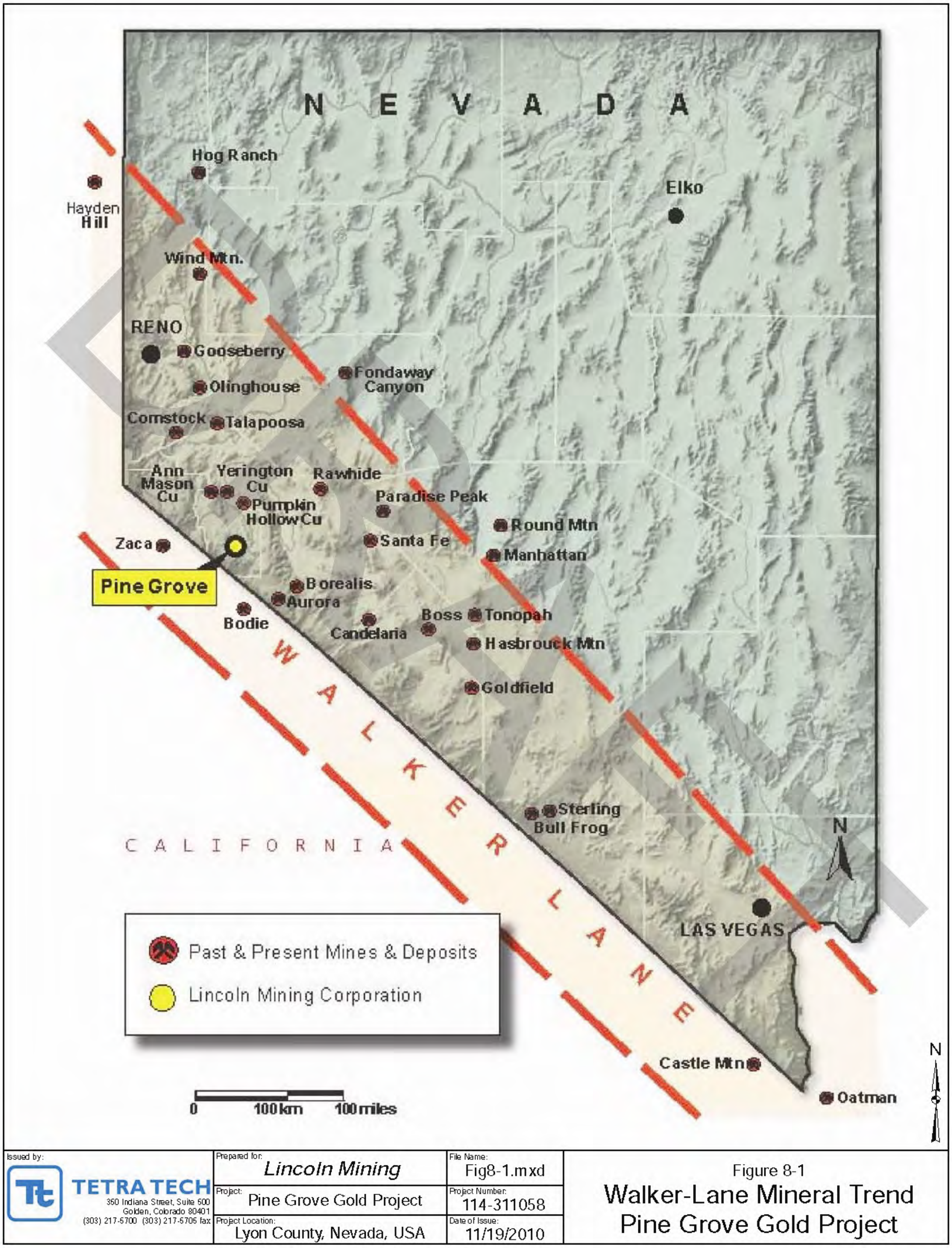
8.0 DEPOSIT TYPE

8.1 Introduction

Telesto has reviewed the section on deposit type from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Telesto Qualified Person agrees with the description of the deposit type of the Pine Grove Project. Moreover, the Telesto Qualified Person visited the site and observed alteration and mineralization consistent with Tetra Tech's description.

The following section (in *italics*) on the Pine Grove mineralized bodies are taken from Tetra Tech (2011).

> *The following information on deposit types has been taken from the 2008 technical report (Stone, 2008), with additional information from other sources as cited.*
>
> *The Pine Grove Property is located within the Walker-Lane mineral trend (FIGURE 8-1)* (**Figure 8.1** in this report)*. According to Stone (2008), the style of mineralization encountered at the Pine Grove project most closely resembles the "Shear Zone" sub-type of the "Plutonic-Related Au Quartz Veins and Veinlets L02" deposit type as described by Lefebure and Hart (2005). In particular, the gold mineralization at the Pine Grove project has the following features in common with the "Plutonic-Related Au Quartz Veins and Veinlets L02" deposit type:*
>
> *Commonly found in tectonic settings of continental margin sedimentary assemblages where intruded by plutons behind margin arcs. Typically developed late in the orogeny or post-collisional settings.*
>
> *Host rocks are equigranular granodiorite with associated, highly differentiated, porphyritic dikes.*
>
> *Mineralization can be divided into intrusion-related, epizonal, and shear veins. Intrusion-related mineralization typically occurs in widespread sheeted vein arrays parallel to the major structural trends. Veins are commonly just hairline fractures to a few centimeters wide and hosted by extensional shears. Veins contain native gold, pyrite, chalcopyrite, and pyrrhotite. Gangue consists of quartz, and sulfides comprise less than 3 percent of the veins. Epizonal mineralization is typically less focused and may be disseminated or occur as replacements. The shear-vein style of mineralization may occur in fault zones outside of the pluton.*



Note: From Tetra Tech, 2011

Figure 8.1: Walker Lane Mineral Trend

Alteration consists of biotite, albite, and sericite, and is spatially restricted to the mineralized zone.

Veins occur close to the associated granite dikes.

Mineralization within the quartz vein and stockwork zones occurs in relatively small tonnage but at relatively higher (2.042 opt) grades. Epizonal deposits have gold grades of 0.058 to 0.146 opt. Combined, these two styles of mineralization can form deposits of ten to hundreds of millions of tons.

A geochemical indicator for these types of deposits is the presence of gold placers in streams draining the plutons. Gold to silver ratios are typically less than one.

9.0 EXPLORATION

9.1 Introduction

Telesto has reviewed the section on exploration from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Telesto Qualified Person agrees with the description of the exploration at the Pine Grove Project. Moreover, the Qualified Person reviewed the references listed herein and found Tetra Tech's description to be accurate.

These sections (in *italics*), taken from Tetra Tech (2011), will briefly summarize the significant historic exploration on the property.

9.2 Rock Chip Geochemical Targets

A review of rock-chip sampling conducted by Teck Resources in 1989 (?) has revealed several targets worthy of further work.

Southern Cross Target: *Ten rock-chip samples are reported from the north-facing slope on a ridge approximately 1500 ft south of the Wheeler deposit. Six of the samples* (bold numbers) *contain significant gold with corresponding elevated copper. TABLE 10-1* (**Table 9.1** in this report) *summarizes the rock-chip assays.*

Table 9.1: Summary of Southern Cross Rock-Chip Assays

Sample Number	Gold Assay	Copper Assay
3606	0.022 ppm	30.2 ppm
3608	0.001 ppm	80.6 ppm
3609	**8.72 ppm**	2,794 ppm
PDS-4	0.030 ppm	1,900 ppm
PDS-5	**8.43 ppm**	2,400 ppm
PDS-6	**2.013 opt**	1,420 ppm
PDS-7	**0.535 ppm**	51 ppm
PDS-8	0.100 ppm	1,580 ppm
PDS-9	**6.12 ppm**	1,580 ppm
PGR-133	**1.61 ppm**	No Data

Lincoln plans to further develop the Southern Cross target by geologic mapping, rock-chip sampling, and a soil survey.

Wilson Long Tunnel Target: *Teck identified this old mine working which is located approximately 4000 ft northeast of the Wilson deposit. The collapsed adit is in favorable granodiorite host rock. Teck took four rock samples from the finger dump and one sample returned 1.58 ppm gold and 99 ppm copper. Lincoln plans to conduct follow-up exploration in this area.*

WS-6 Target: *This target is located on the north side of Pine Grove Creek in a wedge of prospective granodiorite, approximately 1700 ft east of the Wilson deposit. Teck drill hole WS-6 and encountered near-surface gold mineralization from 0 to 45 ft grading 0.0434 opt gold and 87 ppm copper. A single 8-ft horizontal rock-chip sampled collected by a Kinross geologist in 2009 near the drill site contained 2.37 ppm gold. This target warrants further surface sampling and follow-up drilling.*

9.3 Soil Geochemical Targets

Savage Area Targets: *In July and August of 2010, Lincoln conducted a soil geochemical survey for gold and copper. The survey was largely conducted along the western portion of the Wilson patented claims. Twenty-one (21) soil sample lines were oriented in a north-south direction and spaced 100 ft apart. A total of 857 soil samples were collected at 50-ft sample stations from the "B" horizon, where possible. Assay results show six discrete gold-in-soil anomalies which are oriented in a north-south direction (FIGURE 10-1)* (Figure not included in this report)*. Gold anomalies #1 and #2 are in an area of historic underground mining with several Lincoln drill holes that contain narrow, high-grade gold intercepts. These anomalies have coincident copper anomalies. Gold anomalies #3 and #4 are directly above old underground workings and have not been dilled. Gold anomalies #5 and #6 have coincident copper anomalies and are associated with surface prospects. These anomalies may reflect the up-dip extension of low-angle gold mineralization. There are also two, low-amplitude, linear, NW-trending, gold anomalies, #7 and #8, which may reflect structural gold zones. Lincoln intends to drill test the gold anomalies. (see FIGRURE* (sic) *10-1)* (Figure not included in this report)

9.4 Geologic Targets

Scott's Canyon Target: *This target is an area of past mining activity and is located approximately 4200 ft north of the Wilson deposit (FIGURE 10-1)* (Figure not included in this report)*. Several levels of collapsed adits are present that head southward towards the Wilson deposit. No surface sample data are available at this time. Teck drilled one vertical hole in the general area with no significant gold intercepts. The local geology consists of prospective granodiorite with copper-stained material on the local dumps. Owing to the significant amount of past workings and favorable geology, Lincoln believes that additional exploration work is warranted for this area.*

9.5 Step-Out Target

Wilson Step-Out Target: *Gold mineralization remains open in the northeastern portion of the Wilson deposit. Vertical drill hole WS-17, approximately 500 ft from the last row of holes on the Wilson, contains 45 ft grading 0.030 opt gold from 205 to 250 ft. This area has potential for a significant extension of gold mineralization northward towards drill hole WS-17 and beyond. Lincoln plans to drill test this area. Lincoln drilling will also offset vertical drill hole WL-68 which contains 5 ft grading 12.95 ppm gold from 180 to 185 ft and a contiguous 15 ft grading 0.177 opt gold from 185 to 200 ft.*

9.6 Generative Exploration Work

Lincoln is continuing to identify prospective areas at Pine Grove and plans to conduct additional detail geologic mapping, rock-chip sampling, soil sampling, and photo interpretation. Lincoln geologists believe that 80% of the property remains prospective to discovery of new resources.

Lincoln plans to conduct sampling, geologic mapping, and a soil geochemical survey on the ridge to further develop the target.

10.0 DRILLING

10.1 Introduction

Telesto has reviewed the sections on drilling and sampling approach from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Qualified Person agrees with the description of the drilling at the Pine Grove Project. Moreover, the Qualified Person reviewed the references listed herein and found Tetra Tech's description to be accurate.

The following sections (in *italics*) on drilling are taken from Tetra Tech (2011).

10.2 Drilling Summary

Since modern exploration of the Pine Grove Hills began in the late 1960s, Quintana, Teck and Lincoln are known to have drilled on the property. Tt has no information on the single hole drilled by Quintana, as reported in Section 6.1.2 (Section 6.2.2 in this report)*. A total of 273 holes (87,977 ft) have been drilled on the Pine Grove property. TABLE 11-1* (**Table 10.1** in this report) *summarizes the drilling by Teck and Lincoln since 1989.*

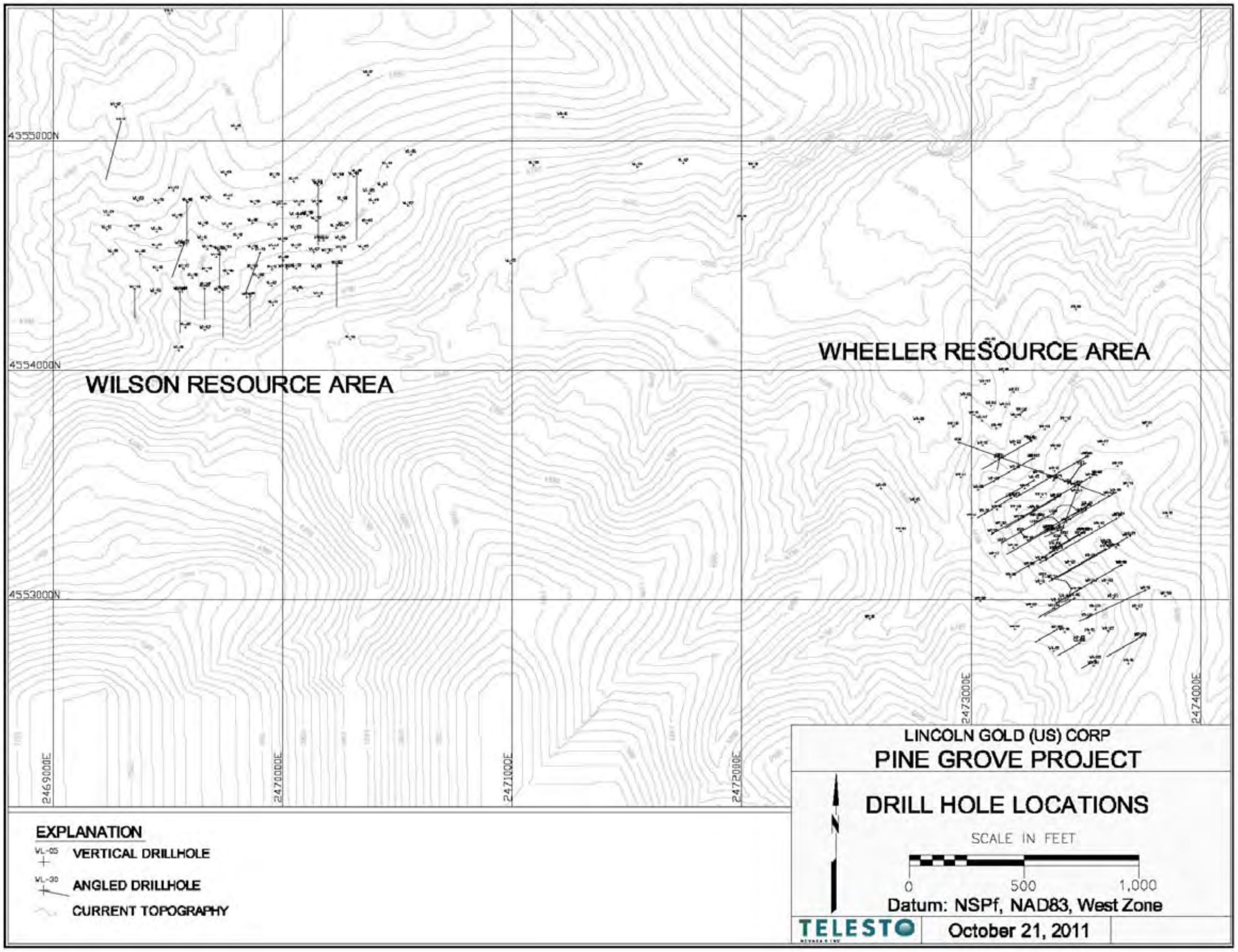
Table 10.1: Summary of Known Drilling at Pine Grove

Location	Type	No. of Holes	Total Footage
Quintana			
Wilson Mine	RC?	1	400
Teck Resources (1991 -			
Wheeler Mine	RC	97	33,608
	Core	2	614
Wilson Mine	RC	62	18,775
District Exploration	RC	29	15,105
Total		190	68,102
Lincoln Gold US Corp. (2008 – 2010)			
Wheeler Mine	RC	33	7,295
	Core*	4	769
Wilson Mine	RC	41	11,061
	Core*	4	740
Total		82	19,865

* Two holes in each deposit not yet analyzed for metallurgical testing
Note: Table 10.1 is adapted from TABLE 11-1 in Tetra Tech (2011)

Figure 10.1 shows the drillhole collar locations and the drill hole trace projections within the project area at Pine Grove.

Figure 10.1: Drill Hole Location Map



10.3 Drilling by Prior Operators

The following information on drilling prior to 2008 was largely taken from the 2007 and 2008 technical reports (Stone, 2007, 2008) with additional information provided by Lincoln.

The only historical records of drilling on the property relate to an exploration program undertaken by Teck from 1988 to 1992. During this period, Teck drilled 188 RC holes and two core holes in the district, as outlined in TABLE 11-1 (**Table 10.1** in this report)*. The bulk of the drilling was concentrated at the Wheeler and Wilson mines, where vertical RC holes and some angle holes were collared on roughly 115 foot-spaced grids that cover the two mineralized areas. Holes were also drilled around the grids to attempt to identify the margins of the mineralization. Lincoln reports that they have no details on Teck's drilling contractors or the type of equipment used, except that they are aware that a track drill was used.*

Following completion of vertical grid drilling, RC angle holes were drilled through the mineralized zones. At the Wheeler mine, the angle holes were situated at roughly 115-foot intervals in the hanging wall and drilled to the southwest, covering about 980 feet of the strike of the mineralization. The holes were angled at 60° to intersect the mineralized zones at 90°. A few angle holes were drilled down-dip to the northeast from the footwall side as well, to test for steeper mineralization controls. At the Wilson mine, the mineralization is essentially flat, however, six angle holes were drilled at 230 foot intervals along the deposit to test for the presence of steeper mineralization controls.

In addition, 29 district exploration holes were drilled between and around the two mineralized areas for exploration purposes.

Drilling was carried out by professional drilling contractors using industry-standard drilling equipment. The drilling and sampling were supervised by Teck professional personnel. Down-the-hole hammer bits were used throughout. Drilling was conducted dry when possible; however, water was occasionally injected, when conditions required, in order to avoid sample contamination. The bulk of the drilling was done dry, and water injection typically occurred only at depth in the holes. Sample recovery from the RC drilling was considered good. Lincoln reports that Teck did not conduct any down-hole surveys of their drill holes. No other details on the Teck drilling were available at the time of writing this report.

10.4 Drilling by Lincoln Gold US Corp.

10.4.1 Metallurgical Drilling

The following information has been taken from the 2008 technical report (Stone, 2008) with additional information provided by Lincoln and from other sources as cited.

10.4.1.1 *Core Drilling – 2008*

In January through February 2008, Lincoln drilled four core holes to acquire mineralized material for metallurgical testing. Major Drilling America Inc. ("Major") of Carlin, Nevada, was the drilling contractor, using a truck-mounted LF140 core-rig. Large-diameter PQ (85 mm diameter) core and HQ (63.5 mm diameter) core were recovered. Two core holes (WL10A, WL34A) were drilled on the Wilson deposit, and two core holes (WR2A, WR82A) were drilled on the Wheeler deposit for a total of 799 feet. Drilling conditions were extremely difficult due to zones of shattered rock and clays. Mine workings (voids 5 to 7 ft) were encountered in both holes on the Wilson deposit. The core was logged on site, and all core was assayed. Lincoln reports that all of the mineralized core was consumed in five column-leach tests at McClelland Laboratories in Sparks, Nevada.

An effort was made to position the core holes in mineralized zones adjacent (±10 ft) to existing RC drill holes completed by Teck. Core hole numbers reflect the adjacent Teck drill hole number with the addition of the letter "A".

The first phase of RC drilling by Lincoln was conducted using a track-mounted Drill Tech Model D25K with 4.0-inch pipe and 4.75-inch to 5.25-inch drill bits. The air compressor was 900cfm/350psi. The mast was capable of handling 20 ft rods. A second phase of RC drilling was conducted using a DLD 1000 mounted on a Cat E-70E with a separate carriage for the compressor and rotary splitter. The rod diameter was 4.0 inches and the bit diameter was 4.5-inch to 5.25-inch. The air compressor was 900cfm/350psi.

10.4.1.2 *Core Drilling – 2010*

An additional four, shallow, vertical HQ core holes were completed in December 2010 for metallurgical samples. The drilling contractor was KB Drilling Company, Inc. of Virginia City, NV using a KMB 1.4 Versa Drill mounted on a Hatachi (sic) *CG70 rubber track chasis* (sic) *and rated at 2,100 ft for PQ core. Two holes were drilled on the Wheeler (WR-131c, WR-132c) and two holes were drilled on the Wilson (WL-104c, WL-105c) for a total footage of 710 ft.* Data from these holes were not available at the effective date of this Technical Report. (Geologic data from these holes is included in the 2011 Telesto evaluation, however, assay data is pending).

10.4.2 Confirmation and Edge Drilling

Lincoln initiated RC drilling in November 2009 to confirm past RC drilling by Teck Resources on both the Wheeler and Wilson deposits and to test the edges of the two deposits on the patented claims. Initial drilling commenced in November 2009 and was completed in February 2010. Drilling was resumed in July 2010 but was shut down shortly thereafter due to poor driller performance. Diversified Drilling LLC ("Diversified Drilling") of Missoula, MT was contracted for both phases of drilling. All drilling was "wet" (water injected) owing to State of Nevada requirements. A face-return RC hammer bit was used as the primary bit and a Tricone bit with skirt was used occasionally when poor ground conditions were encountered. All holes were collared using a 15-ft length of casing. At the completion of each drill hole, the hole was plugged with a 10-ft cement cap and a 12-inch wooden stake with a scribed hole number of a metal label placed into

the cement. All holes were surveyed by Summit Engineering of Reno, NV utilizing Nevada State Plane Coordinates (in feet) (Telesto notes that the coordinates are in Nevada State Plane West Zone NAD83)*. Owing to the shallow nature of the RC holes, no down-hole surveys were conducted. A total of 74 RC holes were drilled in 2010 for 18,356 ft with an average hole depth of 248 ft. Forty-one holes were drilled on the Wilson and 33 holes were drilled on the Wheeler (two holes lost).*
To confirm Teck RC drilling on the Wilson deposit, 11 RC holes were drilled largely as "five spots" (in middle of box of four Teck holes). In addition, two metallurgical core holes were drilled on the Wilson in 2008 that semi-twined (sic) *Teck RC holes WL-10 and WL-34. To confirm Teck RC drilling on the Wheeler deposit, 11 RC holes were also drilled largely as "five spots". In addition, two metallurgical core holes were drilled on the Wheeler in 2008 that semi-twinned Teck RC holes WR-2 and WR-82.*

Edge drilling on the Wilson patent encountered significant gold which warrants follow-up drilling. Edge drilling on the Wheeler patent did not identify significant gold.

10.5 Drilling by Prior Operators

There is no information regarding the sampling method and approach for the single hole drilled by Quintana in the late 1960's. All information presented for prior operators is in regard to the 188 RC holes and two core holes drilled by Teck Resources. None of the Teck drill hole cuttings, core, assay rejects, assay pulps, and chip trays remain. Much of the following information has been taken from the technical report by Stone (2008).

10.5.1 Core Drilling Sampling Recoveries – Teck Resources

Two core holes, WD-1 and WD-2, were drilled by Teck on the Wheeler deposit. No core drilling was conducted on the Wilson deposit. The top of each hole (15 to 19 ft) was rotary drilled to set casing. All information is taken from detail Teck core logs. Core recoveries are summarized below:

- *WD-1: Vertical, total depth 314 ft. Up-hole core recoveries were poor in gouge zones with typical recoveries of 25 To 65% with the worst interval of 6%. Biotized granodiorite had acceptable recoveries on the order of 95 to 100%. Andesite recoveries ranged from 40 to 66%. Mineralized zones in granodiorite with quartz veinlets and gouge had variable recoveries ranging from 25% to 70% and locally up to 100%. Core recovery became noticeably better (95-100%) below 100 ft in hole depth.*
- *WD-2: Vertical, total depth 300 ft. Overall core recoveries are excellent with most at 100%. Core recovery in the overlying Morgan Ranch Formation (sedimentrary* (sic) *breccia and marl) was 100%. Core recovery in the underlying granodiorite was mostly 100%. Mineralized zones were spotty with recoveries from 80 to 100%.*

10.5.2 Drilling by Lincoln Gold US Corp.

Core drilling for metallurgical samples was conducted in 2008 with two shallow, vertical holes completed on the Wilson deposit and one vertical hole and one angle hole *completed on the Wheeler deposit for a combined total of 799 ft. An additional four core holes were drilled for metallurgical samples in late 2010. Two vertical holes were cored on the Wilson deposit and two vertical holes were cored on the Wheeler deposit for a combined total of 710 ft. Data from these last four core holes remains pending. In 2009-2010, 41 RC holes were drilled on the Wilson deposit and 33 were drilled on the Wheeler deposit for a total of 18,356 ft.*

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

Telesto has reviewed the sections on sampling from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Telesto Qualified Person agrees with the description of the sampling at the Pine Grove Project. Moreover, the Qualified Person reviewed the references listed herein and found Tetra Tech's description to be accurate.

The portions of text below which are in *italics* are from Tetra Tech (2011). Information and analysis by the Telesto Qualified Person is not italicized.

11.1 Drilling by Teck Resources

The following information is summarized from various documents from Teck Resources and written communication with Mr. Phil Jackson, ex-Teck project geologist (June 27, 2008).

11.1.1 Core Drilling Sampling Procedures – Teck Resources

There is no surviving description of the Teck core sampling procedure.

11.1.2 RC Drilling Sampling Procedures – Teck Resources

RC drill holes completed during Teck's exploration programs were sampled over the entire length of most holes at regular intervals of 5 ft. For the vertical drilling at the Wheeler mine, where the mineralization dips at 30 degrees, the samples represent a true length of about 4 ft. The angle holes at the Wheeler intercepted the mineralization at 90 degrees, and these samples represent true widths. Because the mineralization at the Wilson mine is essentially flat, samples there represent true widths.

All of the material returned by the drill from each sample interval was collected in 5-gallon buckets by personnel from the drilling company under the supervision of a Teck geologist. The samples were then divided with a Jones (?) splitter to produce two sample splits, each weighing roughly 11 to 22 pounds for each sample interval. The sample splits were transferred to olefin sample bags and labeled on the outside in permanent marker with the drill hole number and footage. The bagged sample splits were then piled in two separate areas at the drill site.

One set of sample splits (the "assay sample") was transported to Chemex Lab's sample receiving facility in Sparks, Nevada. The assay samples were transported to the lab at the end of each day, or every other day at the most. At the beginning of the program, the assay samples were transported to the lab by Teck personnel directly from the drill site. Later in the drilling program, personnel from the lab picked up the samples each day.

The other pile of samples (the "second splits") remained at the drill site where some samples from each hole were selected for check assaying at various times during the drilling program. At the end of the program the second splits were retrieved from the drill sites and discarded.

A handful of material from each sample interval was collected by the supervising geologist as the sample was collected and split. The material was examined and described on a logging sheet at the drill site. A portion of the material was transferred to a plastic chip-tray and labeled. The chip trays were transported to Teck's office in Reno, Nevada for storage.

11.1.3 Sample Preparation and Analyses – Teck Resources

Teck Resources submitted all of their RC drill cuttings to Chemex Labs, Inc. in Sparks, NV over a period beginning in October 1989 through February 1991. Chemex Labs, now ALS Chemex Labs, is an ISO certified, Quality Management System registered facility and runs a variety of internal certified standards, blanks, and check assays. No aspect of the sample preparation was conducted by an employee, officer, or associate of Teck Resources.

Teck RC drill samples were assayed for gold and copper, with assumed waste rock intervals not assayed in several holes. Initial oven-dried sample weights commonly ranged from 4 to 10+ lbs. Using standard preparation methods, gold was assayed by 1-assay-ton fire assays with A.A. finish. Most gold assay results were reported in ounces gold per ton (oz Au/t). No information is available on the method of copper analyses. Copper assays were reported in ppm copper. Assay rejects and pulps were returned to Teck Resources; none of these materials remain. Copies of all original Certificates of Analysis and drill hole logs are available.

11.1.4 Check Assaying – Teck Resources

Teck Resources conducted a check assay program only on samples from the Wheeler deposit. No samples from the Wilson deposit were involved. Check assaying was accomplished at the Wheeler in four phases:

- *Phase 1 – Check assays on 47 samples from Wheeler underground panel samples and pulps*
- *Phase 2 – Check assays on 24 pulp samples from 14 RC drill holes from initial Wheeler drilling*
- *Phase 3 – Check assays on fine and coarse fractions from second splits from RC drill samples (45 samples)*
- *Phase 4 – Check assays on larger samples, finer crushing, larger pulps, and larger assay charges on 158 samples from 23 holes in the second round of RC drilling on the Wheeler*

The primary laboratory in all check assay phases was Chemex Labs in Sparks, NV. Additional laboratories utilized in the various phases of check assaying were GSI Labs in Sparks, NV and American Assay Labs in Sparks, NV.

11.1.5 Security – Teck Resources

Drill samples were safeguarded on site by Teck personnel until they were transferred to Chemex Labs in Reno, NV. Periodically, a Chemex truck picked up the samples and transported them to the lab. Chemex was responsible for safeguarding the samples under their control. Given the competence of Teck Resources, sample security is presumed to have been excellent.

11.1.6 Quality Control – Teck Resources

Teck Resources did not include certified reference material (blanks and standards) in their sample stream. Teck conducted significant check assaying at the Wheeler deposit but none at the Wilson deposit. Duplicate samples were not included in the check assaying program.

11.1.7 Sample Quality – Teck Resources

The Qualified Person believes that Teck's RC sample quality meets industry standards and has been verified by RC drilling conducted by Lincoln. The Qualified Person also believes that Teck's core sampling meets industry standards for quality. Overall, The Qualified Person believes that the sampling was conducted in a careful and professional manner and that the samples are representative of the mineralized material that was drilled.

11.2 Drilling by Lincoln Gold US Corp.

The following information was provided by Lincoln professional staff.

11.2.1 Core Drilling Sampling Procedures – Lincoln

After each core run, PQ and/or HQ core was carefully removed from the core barrel by the drill crew and put into waxed cardboard core boxes. Core run intervals were clearly marked on wooden dividers within each box. Both the box and lid were clearly marked with the hole number, box number, and core interval. When full, each core box was tied shut with heavy duty string. After each drill shift, the Lincoln project geologist personally transported the core to a locked storage facility in Yerington, NV. At the storage facility, the core was photographed by the geologist and logged. The core was later transported by Lincoln personnel directly to McClelland Laboratories Inc. ("McClelland") in Sparks, NV. At McClelland, a Lincoln geologist selected 40 hand-sized core specimens of various rock units for density measurements. The geologist also determined intervals for assay. The core was crushed by McClelland to an appropriate size from which splits were sent to ALS Chemex in Reno, NV for gold analyses (fire assay with A.A. finish). Subsequent assay data were used to determine mineralized zones which were composited from the core for column leach testing by McClelland. One core hole from the Wilson deposit, hole WL-10A, did not provided (sic) *an adequate volume of mineralization for column leach testing. All other holes provided sufficient material for five 6-inch column leach tests. No intact core survived the metallurgical testing program.*

Note: The five 6-inch column leach tests referred to in the previous paragraph were actually two 8-inch and three 4-inch inside diameter (I.D.) column leach tests.

Two core holes, WR-2A and WR-82A, were drilled on the Wheeler deposit for metallurgical samples. These holes were semi-twins of RC holes WR-2 and WR-82.

- *WR-2A: Vertical, total depth 149 ft. Gold mineralization was present in multiple zones throughout the hole. Nearly the entire hole was in highly broken granodiorite. Overall core recoveries were on 70 to 80% with short internal zones of 90 to 100%.*
- *WR-82A: Angle (-45°), total depth 250 ft. Gold mineralization was present in multiple zones throughout the hole. Core recovery in the overlying Morgan Ranch Formation was good at 90 to 100%. Mineralized zones below the Morgan Ranch were badly broken with core recoveries of 50 to 70% with local intervals of 80 to 100%.*

Two core holes, WL-10A and WR-34A (hole is actually WL-34A, Telesto)*, were drilled on the Wilson deposit for metallurgical samples. These holes were semi-twins of RC holes WL-10 and WL-24* (hole is actually WL-34, Telesto)*.*

- *WR-10A: Vertical, total depth 199 ft. Sparse gold mineralization was encountered. Core recovery up hole in the rhyolite porphyry was 60 to 80%. Core recovery in the underlying granodiorite was good at 90 to 100%.*
- *WR-34A: Vertical, total depth 201 ft. Core recovery in the overlying rhyolite porphyry was about 60%. The rhyolite was highly broken. Similarly, the "pink" feldspar porphyry was highly broken with recoveries on the order of 60%. Overall core recovery in the granodiorite was good at 90 to 100%. Core recovery in the broken mineralized zones ranged from 50 to 60%.*

Data for Wilson 2010 core holes WL-104c and WL-105c and Wheeler 2010 core holes WR-131c and WR-132c remain pending at the time of this report.

11.2.2 RC Drilling Sampling Procedures – Lincoln

All holes were sampled at 5-ft intervals except in cases where there was a change from hammer bit to tricone bit or where mine workings and voids were encountered. Owing to 15 ft of casing in each hole, the first three samples in each hole were collected dry. All sampling below the casing was done "wet" as per Nevada State law. All sampling and drilling were done under the supervision of Lincoln geologists or experienced field technicians trained by Lincoln geologists. A sample log sheet was made for each drill hole that included down-hole sample intervals with sample numbers, the certified standards, blanks and duplicates insertion depths, time of rod changes, depth of hole, presence of voids or recovery problems, and other pertinent information. When each hole was completed, information on the field sheet was entered into an excel worksheet to provide electronic format and backup copy.

Wilson holes WL-63 through WL-96 and Wheeler holes WR-98 through WR-112 were sampled in the following manner. Rock cuttings were discharged from the center return tube into a cyclone and then through a rotary wet splitter where the sample was *separated into waste discharge and assay sample discharge tubes. The volume of material directed to the assay side of the splitter was controlled by "sample dividers" as to not overflow the 5 gallon buckets catching the sample. The remainder of the sample was discharged as waste. A "Y" splitter was used at the sample discharge side of the wet splitter to capture the primary "assay" sample of and a "duplicate" sample. After decanting the water and drying the samples in a lab oven, sample weights were commonly 7 to 12 lbs. The assay sample was always collected from the same side of the "Y" splitter. A sample for geologic logging was always collected from the waste discharge side of the wet splitter. Sample bags were labeled with consecutive numbers down the hole for each sample interval. Within each sample interval a "duplicate" sample was given the same number as the primary assay sample with the addition of the letter "d." Duplicate samples were collected for additional analyses and metallurgical work. Certified standards and blanks were inserted into the sample stream in 50-g plastic sample packets and is further discussed in Section 13. All drill samples were transported by Lincoln staff to the Yerington field office where they were inspected and prepared for transport to ALS Chemex in Reno, NV. All drill samples were kept under lock and key. ALS Chemex made weekly trips for sample pickup.*

Owing to an increasing awareness of a "nugget effect," Lincoln determined that larger RC drill samples would produce more reliable gold assay results. Wilson holes WL-97 through WL-103 and Wheeler holes WR-113 through WR-130 were sampled in the following manner. Rock cuttings were discharged from the center return tube into a cyclone and then through a rotary wet splitter where the sample was separated into waste discharge and assay sample discharge tubes. No "duplicate" sample was collected and the size of the primary assay sample was increased so as to nearly fill a 5-gallon bucket. After decanting the water and drying the samples in a lab oven, sample weights were commonly 15 to 40 lbs.

11.2.3 Sample Preparation and Analyses – Lincoln

All RC drill samples were delivered to ALS Chemex Labs Inc. in Reno, NV. The Nevada laboratory is ISO/IEC 17025:2005 accredited for gold assays and a Quality Management System registered facility and runs a variety of internal certified standards, banks, and check assays. No aspect of sample preparation was conducted by an employee, officer, director, or associate of Lincoln.

Initial dry sample weights were about 7 to 12 lbs. Later in the drill program, Lincoln ceased collecting duplicate samples and the primary sample weights increased to about 15 to 40 lbs. All Lincoln samples were analyzed for gold and copper.

All drill samples were logged into the lab system and inventoried to confirm correctness of the sample transmittal sheet. Samples were then dried under high temperature (code DRY-21) and weighed. After weighing, the samples were fine crushed to 70% <2 mm (code CRU-31) and then split with a Boyd Rotary Splitter (code SPL-22Y). The 1000 g split was then pulverized to 85% <70 µm (code PUL-32).

Gold was analyzed by a 30-gram 1-assay ton fire assay with A.A. finish (code Au-AA23). Samples returning over 10 grams per ton gold (over limit) were re-assayed by fire assay with gravimetric finish (code Au-GRA21). Gold assay results are reported in ounces Au per ton.

Copper was analyzed by inductively coupled plasma with atomic emission spectroscopy ("ICP-AES"). Samples were digested by a four acid "near total" digestion method and analyzed by ICP-AES (code ME-ICP61). Assays over 10,000 ppm Cu (over limit) were re-run with a higher copper assay method (code Cu-OG62). All copper assays are reported in ppm.

11.2.4 Check Assaying – Lincoln

Lincoln ran three check assay programs on samples from the Lincoln's RC drilling.

- *Program 1: Same-lab (ALS Chemex) duplicate pulp assays from 63 drill holes (249 samples).*
- *Program 2: Second-lab (Inspectorate America) assays on new pulps from rejects from 63 drill holes (286 samples).*
- *Program 3: Screen assays (ALS Chemex) on samples from 11 drill holes (28 samples).*

11.2.5 Security – Lincoln

At the end of each drill shift, all samples were removed from the drill site by the project geologist or geotechnician and taken to a secure warehouse and office facility maintained by Lincoln in Yerington, Nevada. At the warehouse, all samples were inventoried and prepared for transport to ALS Chemex in Reno, NV. Upon completion of five to six holes, ALS Chemex picked up the samples and transported them by truck to their lab in Reno. Security of the samples was the responsibility of ALS Chemex once the samples were removed from the Lincoln facility in Yerington. Sample security procedures are very tight at ALS Chemex.

All sample rejects and pulps have been returned to Lincoln and are presently stored in Lincoln's field office-storage facility in Yerington, NV. When no Lincoln personnel are present, the facility gate and building are locked.

11.2.6 Quality Control – Lincoln

Lincoln utilized certified reference material (standards and blanks) and two check assay programs as its primary quality control for the RC drilling at Pine Grove….Duplicate drill samples were also collected.

The results of the analysis of Lincoln's check assay program are found in Section 12 in this report.

Certified reference material was purchased from WCM Minerals of Burnaby, B.C., Canada. This material consisted of granitic rock containing gold and copper values *associated with porphyry copper mineralization and is similar to the granodiorite host rock at Pine Grove which contains both gold and copper. Four certified gold-copper standards were utilized which contained values of 0.008, 0.033, 0.083, and 0.127 oz Au/t (FIGURE 13-2a through 13-2d)* (Figures not included in this report)*. The standards also contained 0.21, 0.31, 0.35, and 1.06% copper (FIGURE 13-3a through 13-3d)* (Figures not included in this report)*. A single blank was utilized with a certified assay of <5ppb gold and 3 ppm copper. The standards and blanks were provided in 50-g plastic packets. The figures show results less than a 3% deviation from the known value in most cases, with a few outliers less than a 5% deviation.*

Standards and blanks were entered into the RC drilling sample stream on roughly 100 ft intervals and/or where deemed appropriate by the geologist or geotechnician. Standards were numbered as part of the normal drill hole sample sequence and identified in a drill hole sample record. Standards represent approximately 5% (1 in 20) of all samples submitted for assay. Blanks represent approximately 2% (1 in 50) of all samples. Duplicate samples were collected in the initial phase of drilling and designated by original sample number followed by a "d."

ALS Chemex also ran sample preparation and analytical quality control for every sample batch. These controls included sieve measurements and the inclusion of blanks, certified standards and analytical duplicates. Crushing (code CRU-QC) and pulverizing (code PUL-QC) tests are routinely run to test preparation. For regular fire assay methods, ALS Chemex runs two standards, 3 duplicates, and one blank for a rack size of 84 samples. For regular ICP-AES assay methods, the lab runs two standards, one duplicate, and one blank for a rack of 40 samples. These data are reported in a QC Certificate of Analysis for each hole drilled by Lincoln at Pine Grove and are all available.

11.2.7 Sample Quality – Lincoln

Lincoln core drilling produced adequate and representative mineralized sample for column leach tests conducted at McClelland Laboratories. The core also verified the geology and mineralization in adjacent RC drill holes. The Qualified Person believes that the quality of Lincoln's RC drill hole samples meets industry standards and is acceptable for confirmation of past Teck RC holes. Rock units and mineralized zones encountered in Lincoln's RC drill holes correlate reasonably well with those identified in past Teck RC drill holes. Overall, The Qualified Person believes that Lincoln sampling was conducted in a careful and professional manner and that the samples are representative of the mineralized material that was drilled.

12.0 DATA VERIFICATION

12.1 Introduction

The Pine Grove database was provided by Lincoln in electronic form that included drill hole collar coordinates, drill hole alignment, down-hole interval, assay, alteration and rock type data. Original assay certificates were provided in the form of paper copies which were then scanned and filed.

The electronic database consists of data from 261 drill holes for a total of 15,472 assay values. The Teck drilling component of the database consists of 166 RC holes and 2 core holes and the Lincoln drilling data consists of 71 RC holes and 8 core holes. Original assay certificates from both company's drill programs have been provided by Lincoln. The majority of assays from the Teck drill program were performed by Chemex Labs Ltd. in Sparks, Nevada, while all primary assays for the Lincoln drill program were performed by ALS Chemex and ALS Minerals in Reno, Nevada.

Data verification has been accomplished by:

1. Review of all assay certificates from commercial analytical laboratories that confirm the presence of gold mineralization and the values in Lincoln's electronic assay database.
2. Comparison of electronic rock coding to original and simplified geologic drill logs.
3. Comparison of check assay data from second independent analytical lab.
4. Statistical evaluation of sample pulp duplicates, drill standards and blanks submitted for analyses by Lincoln.
5. Comparison of core vs. RC holes.
6. Detailed inspection of all cross-sections to compare drill hole collar elevations to recent digital topography.
7. Visual inspection of alteration, rock types, and structure in outcrops at the property.
8. Visual confirmation of drill sample duplicates, standards and sample security measures at the Lincoln warehouse in Yerington, Nevada.
9. Review of all historical documents related to the project area.
10. Review of all geologic, base, soil geochemical, and underground maps.
11. Review of all reports from Tetra Tech, JBR, Kappes, Cassiday & Associates, and McClelland Laboratories.

12.2 Electronic Database Verification

A comprehensive program of data entry and data verification was undertaken by Telesto prior to the building of a resource model. Original assay certificates were compared line by line to the electronic database provided by Lincoln to ensure that the transcription of the data was accurate. Values which disagreed with the certificates were corrected and noted on the assay sheets. The number of incorrect assays was very small, comprising less than 0.1% of the total database. This verification provided a clean database wherein each assay in the database was verified against the original certificate.

After the completion of assay data verification, a program of rock code data verification was conducted by Telesto. Rock codes for all drill hole intervals were provided by Lincoln in electronic form. Logs for all drill holes were provided in the form of paper copies which were then scanned and filed.

During their drilling program, Lincoln geologists gained a good understanding of the rock units at the Pine Grove area. Because of this understanding, a concise and consistent rock unit coding protocol was developed. However, because the Teck drilling program consisted of logs from at least five different geologists, some of the rock units were described in an inconsistent manner. To provide a consistent rock code database, Lincoln geologists reviewed all Teck drill logs and assigned the appropriate rock codes they had developed to each interval.

Telesto compared approximately 20 percent of the Lincoln and Teck drill logs with the electronic rock codes contained in the database. The correlation between the drill logs and the database was very good with only approximately 1 percent of inconsistency. Telesto has determined that the rock codes used in the database have been transcribed accurately and are appropriate for use as a parameter in the resource estimation for Pine Grove.

Contained in the database provided by Lincoln is a set of Teck underground channel sample analyses. No original assay certificate or sample lithologic description for these samples have been provided or reviewed. Because the underground sample analyses cannot be traced back to the original assay source, and the assay data cannot be validated by comparison to nearby drill holes, Telesto has removed the underground sample data from the resource model database.

12.3 Check Assaying – Teck Resources

The following description of the Teck Resources check assay program is extracted from a Lincoln internal report by Phil Jackson (former Teck geologist).

Teck Resources conducted a check assay program only on samples from the Wheeler deposit. No samples from the Wilson deposit were involved. Check assaying was accomplished at the Wheeler in four phases:

Phase 1 – Check assays on 47 samples from Wheeler underground panel samples and pulps.

Note: The underground channel sample analyses have been removed from the Telesto resource model database because of the absence of chain of title.

Phase 2 – Check assays on 24 pulp samples from 14 RC drill holes from initial Wheeler drilling.

Phase 3 – Check assays on fine and coarse fractions from second splits from RC drill samples (45 samples).

Phase 4 – Check assays on larger samples, finer crushing, larger pulps, and larger assay charges on 158 samples from 23 holes in the second round of RC drilling on the Wheeler

The primary laboratory in all check assay phases was Chemex Labs in Sparks, NV. Additional laboratories utilized in the various phases of check assaying were GSI Labs in Sparks, NV and American Assay Labs in Sparks, NV.

*Phase 1 check assays on the Wheeler underground samples produced acceptable results for a deposit containing discrete gold grains with non-uniform distribution. The check assays are fairly consistent across grade. Check assay pulps show an overall correlation coefficient of 0.982 (Chemex vs. GSI) and 0.893 (Chemex vs. American Assay). Check assays in the higher portion of the deposit (6,715 level) showed better reproducibility than did samples from the lower level (6,600 level). A scatter plot is shown on **Figure 12.1**.*

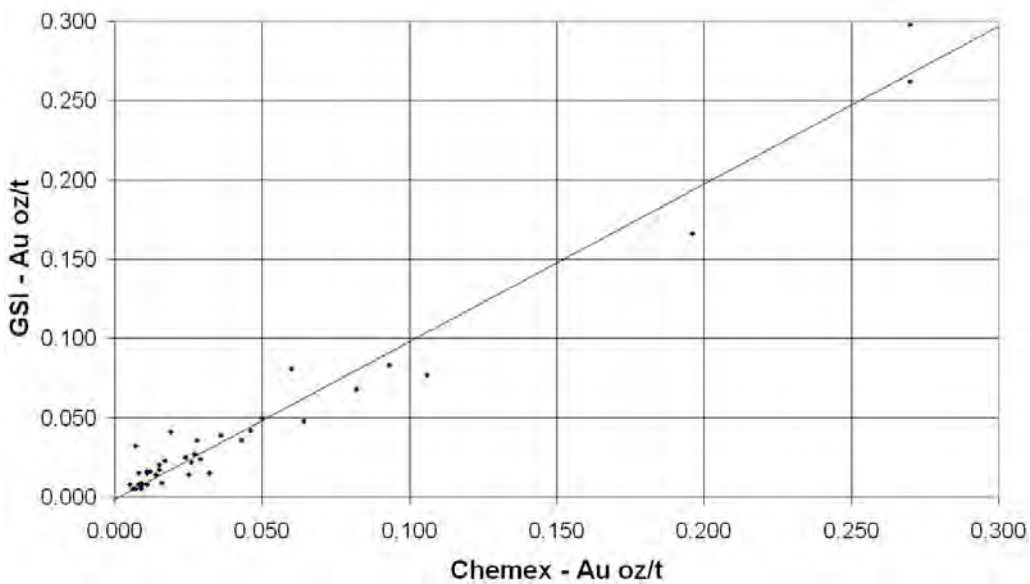


Figure 12.1: UG Pulp Check Assays – Chemex vs. GSI >0.005

*Phase 2 check assays on 24 RC drill hole pulps (Chemex vs. American Assay) show an overall correlation coefficient of 0.907 with oxidized samples showing 0.914 and unoxidized samples showing 0.806. A scatter plot is shown on **Figure 12.2**.*

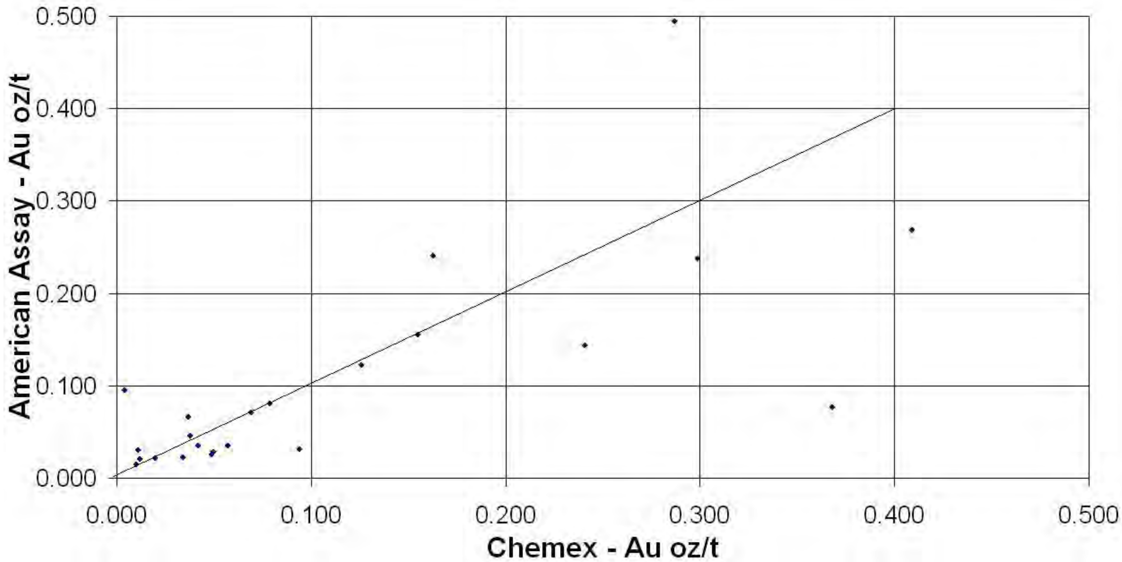


Figure 12.2: Drill Holes WR-1 through WR-14 – First 24 Samples

*Phase 3 check assays showed an overall correlation coefficient of 0.889, oxidized 0.916, and unoxidized 0.708 (Chemex vs. American Assay). Screen analyses revealed that original assays on the >10 mesh fraction correlate better with their check assays (0.951) than did the correlation of original assays on the <10 mesh fraction with their check assays (0.906). It was also determined that larger pulps improved the reproducibility of assays. A precision plot is shown on **Figure 12.3**.*

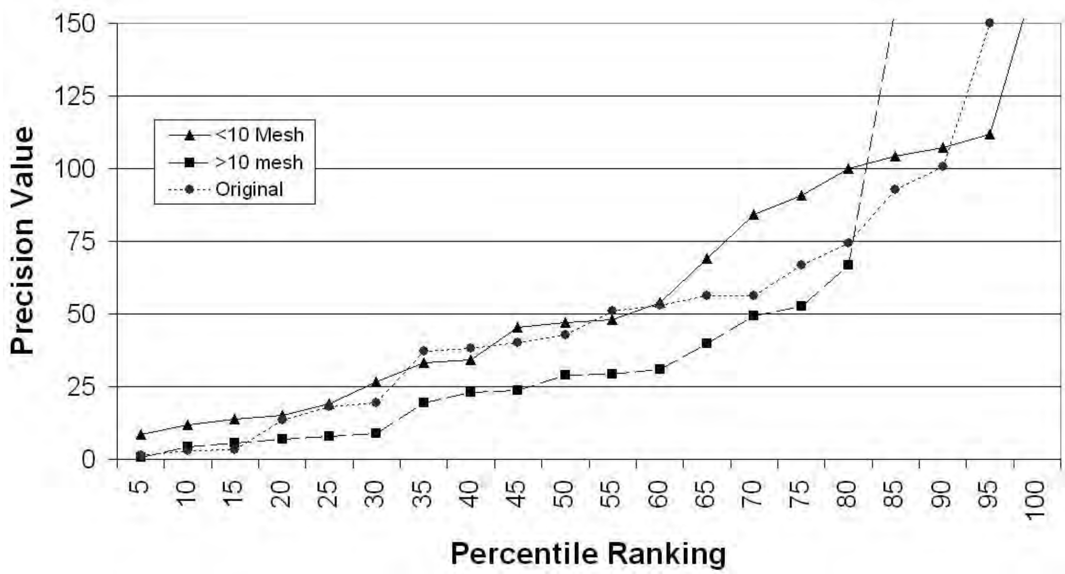


Figure 12.3: Drill Holes WR-1 through WR14 – All Samples <10 Mesh and >10 Mesh Fractions

Phase 4 of check assaying involved samples from 23 subsequent reverse-circulation rotary drill holes at the Wheeler mine (WR-24 through WR-46). In an on-going attempt to further improve *the reproducibility of the drill sample assays, a test was conducted at Chemex Labs using a new preparation and assay procedure that utilized the entire second split for 158 drill samples.*

Each second split was crushed whole to <10 mesh. A large (multi-kg) split was taken that was further crushed to approximately <150 mesh. A 2-kg pulp was prepared from this split, and a very large (5 assay ton) charge was used for the fire assay. The results were compared to the results from the analyses of the assay splits using the protocol discussed in the previous section.

Of the 158 samples, 30 assayed greater than 0.005 oz/t Au. Eleven of the pulps from the assay split were re-run by American Assay Labs, as were 14 of the pulps from the second splits, using one assay ton fire assays.

*The second split ("bulk") assays correlated very well with the assay split ("split") assays (.999). A precision plot for all of the 33 samples shows that at the 50th percentile the samples have check assays are ±28% of the original values, and at the 90th percentile the precision is ±75%, a significant improvement over previous check assaying. A precision plot is shown on **Figure 12.4**.*

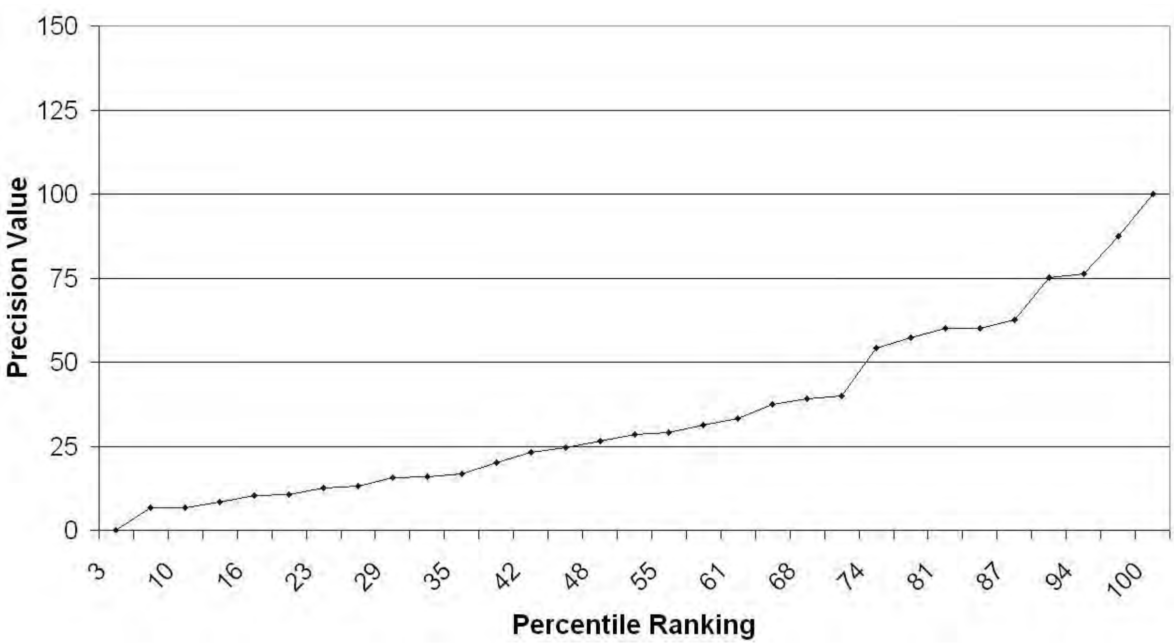


Figure 12.4: Drill Holes WR-24 through WR-46 Splits vs. Bulk >0.005 oz/t Au

*As shown on **Figure 12.5**, the check assays of the bulk pulps showed significantly better correlation, and better precision than did the check assays for the split pulps:*

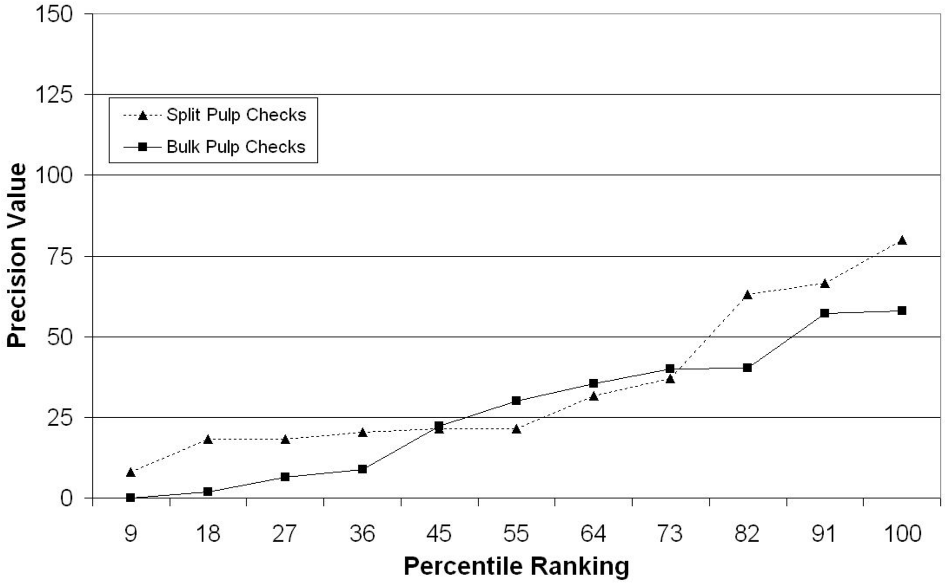


Figure 12.5: Drill Holes WR-24 through WR-46 Splits and Bulk Pulp Checks

Table 12.1: Splits and Bulk Checks Correlation

Samples	Correlation Coefficient	Precision at 50th percentile	Precision at 90th percentile
Split pulp check (n = 11)	0.894	22%	65%
Bulk pulp check (n = 14)	0.961	26%	55%

Note: Table 12.1 is adapted from an unnamed table in Jackson, date unknown

The fact that the bulk pulp checks show that 90% of the check assays are ±55% of the original assay is by far the best reproducibility measure seen for any of the check assay tests. It was deemed that the "bulk" method of sample preparation and analysis gives higher quality results than any of the previous methods.

As a result, the second splits for all reverse-circulation rotary drill samples of oxidized and potentially mineralized material from all holes drilled to date (WR-21 through WR-46) were sent to Chemex Labs and crushed whole to <10 mesh. A large (multi-kg) split was taken that was further crushed to approximately <150 mesh. A 2-kg pulp was prepared from this split, and a very large (5 assay ton) charge was used for the fire assay. These values are used in place of the original assays as the final assays for those samples. The samples for the first 24 holes (WR-1 through WR-24) were not re-run, as the second splits for those samples were consumed by the ±10 mesh re-runs during the third check assay phase.

In addition, a new sampling protocol was instituted for all new reverse-circulation rotary drilling subsequent to this study (from drill hole WR-46 on). An attempt was made to collect all of the sample material that was returned from a five-foot sample interval. The material was collected in covered 5-gallon buckets that were then sent to Chemex Labs for preparation and analysis using the new protocol discussed above.

12.3.1 Statement of Adequacy – Teck Data

The Qualified Person believes that Teck's RC sample quality meets industry standards and has been verified by RC drilling conducted by Lincoln. RC 5-ft sample intervals are appropriate. The Qualified Person also believes that Teck's core sampling meets industry standards for quality. Core sample intervals were determined by rock type and mineralization. Overall, The Qualified Person believes that the sampling was conducted in a careful and professional manner and that the samples are representative of the mineralized material that was drilled.

The Qualified Person concludes that consistent variations seen in the check assay results likely reflect the natural variability of gold in the rocks, rather than problems with the sampling, preparation, or assaying procedures. Considering that the Wheeler mine diamond drill core contains visible gold in places, the check program appears to give acceptable reproducible results, and indicates a satisfactory level of accuracy in the assays.

Although no standards or blanks were utilized in the Teck drilling program, a long and involved series of tests was performed on the drill samples in an attempt to find sample preparation and analytical methods that returned an acceptable correlation between the original values and the check assays, and an acceptable level of reproducibility for the assays. Teck tests resulted in a series of methods that produce acceptable check assay results, and these methods were immediately employed by the on-going drilling program.

Chemex, the analytical lab that performed the assaying for Tech, included blanks, standards and duplicate samples as part of its internal QA/QC protocol.

It is the opinion of the Qualified Person that the Teck drilling assay data was conducted at a high level of accuracy and is appropriate for inclusion in the resource estimation database.

12.4 Check Assaying – Lincoln

Lincoln ran three check assay programs on samples from the Lincoln RC drilling program.

Program 1: Same-lab (ALS Chemex) duplicate pulp assay analyses.

Program 2: Second-lab (Inspectorate America) assays on new pulps from rejects from 63 drill holes (286 samples).

Program 3: Screen assay analyses (ALS Chemex).

12.4.1 Duplicate Assays

Lincoln directed ALS Minerals to conduct assay analyses of duplicate pulp samples from 63 drill holes (249 samples) representative of both the Wheeler and Wilson deposits. Telesto compared the duplicate assays with the original assays to gain an understanding of the reproducibility of assay values. The correlation coefficient (duplicate assay / original assay) for all samples was 0.75 indicating that on average the original assays returned values approximately 25% greater than the duplicate pulps. The presence of a nugget effect at the Pine Grove property may be a factor in the somewhat inconsistent assay reproducibility. A series of scatter charts are shown on **Figures 12.6A** through **12.6D**.

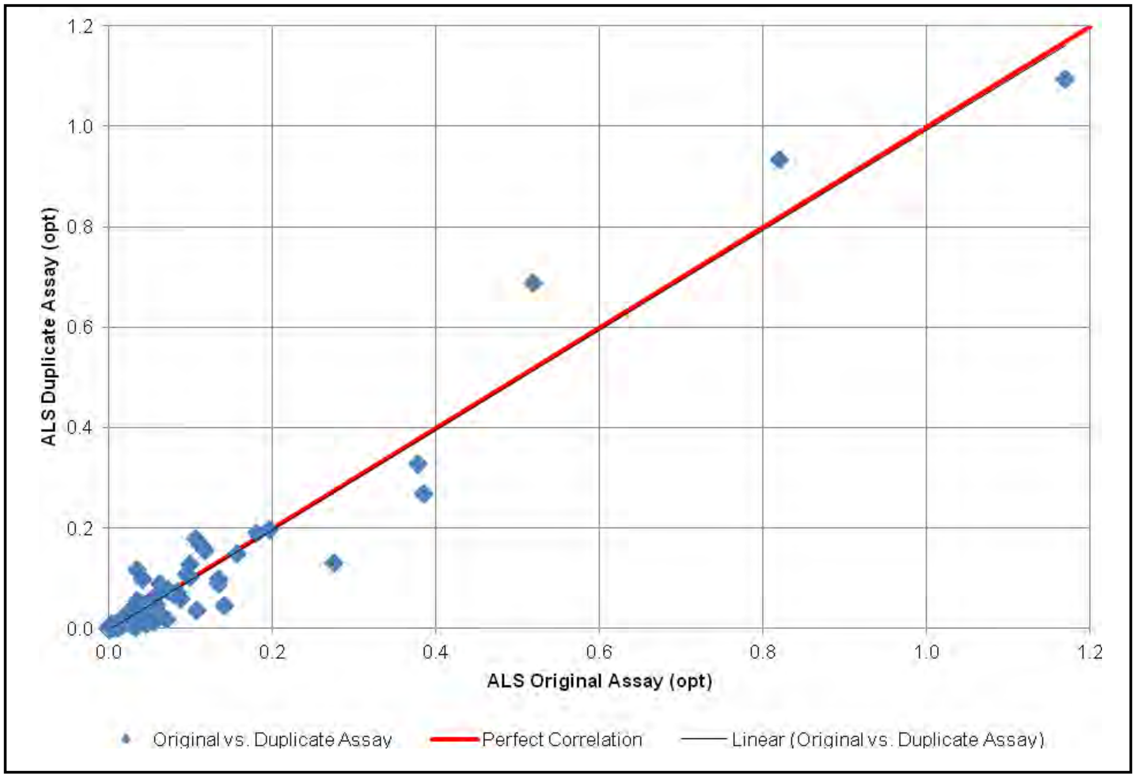


Figure 12.6A: Wilson Deposit Duplicate Assays (All Samples)

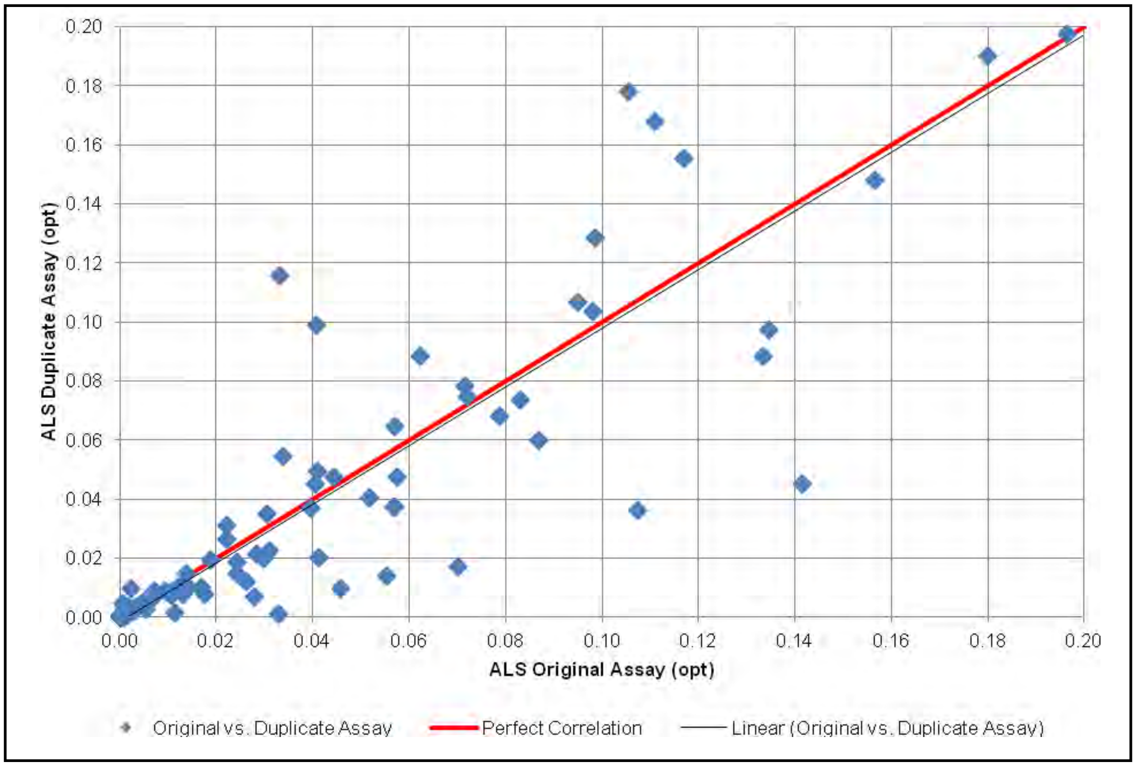


Figure 12.6B: Wilson Deposit Duplicate Assays (Samples < 0.2 opt)

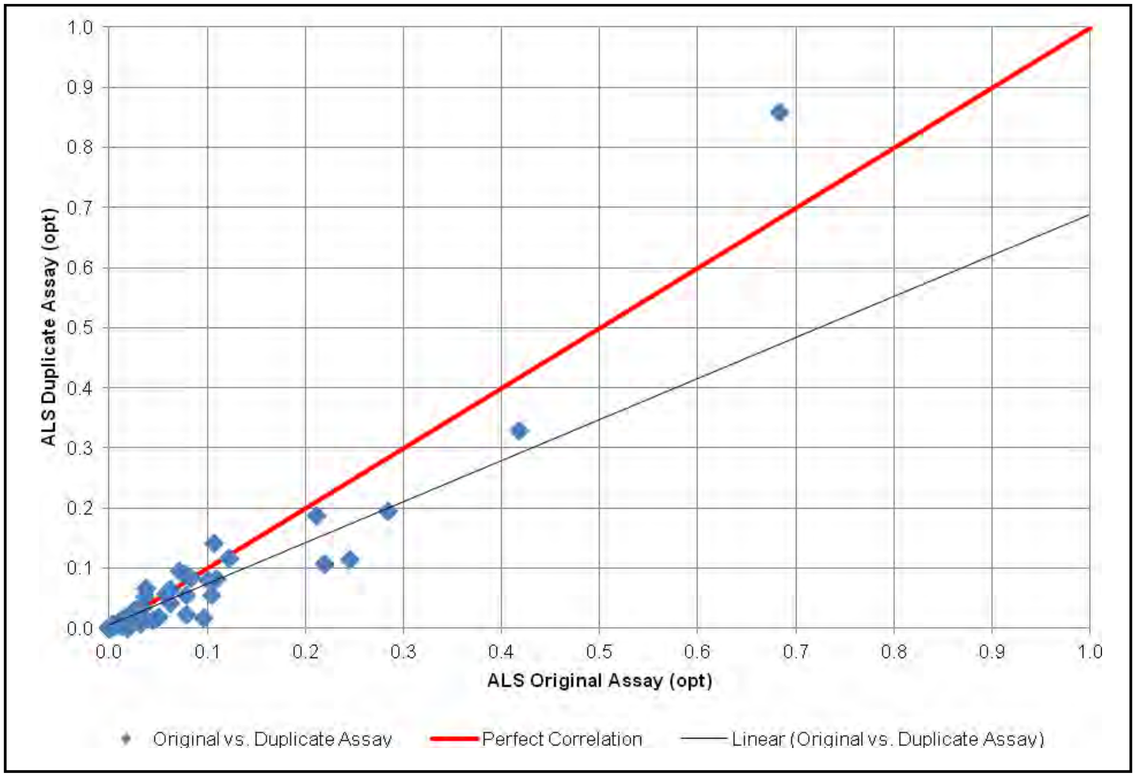


Figure 12.6C: Wheeler Deposit Duplicate Assays (All Samples)

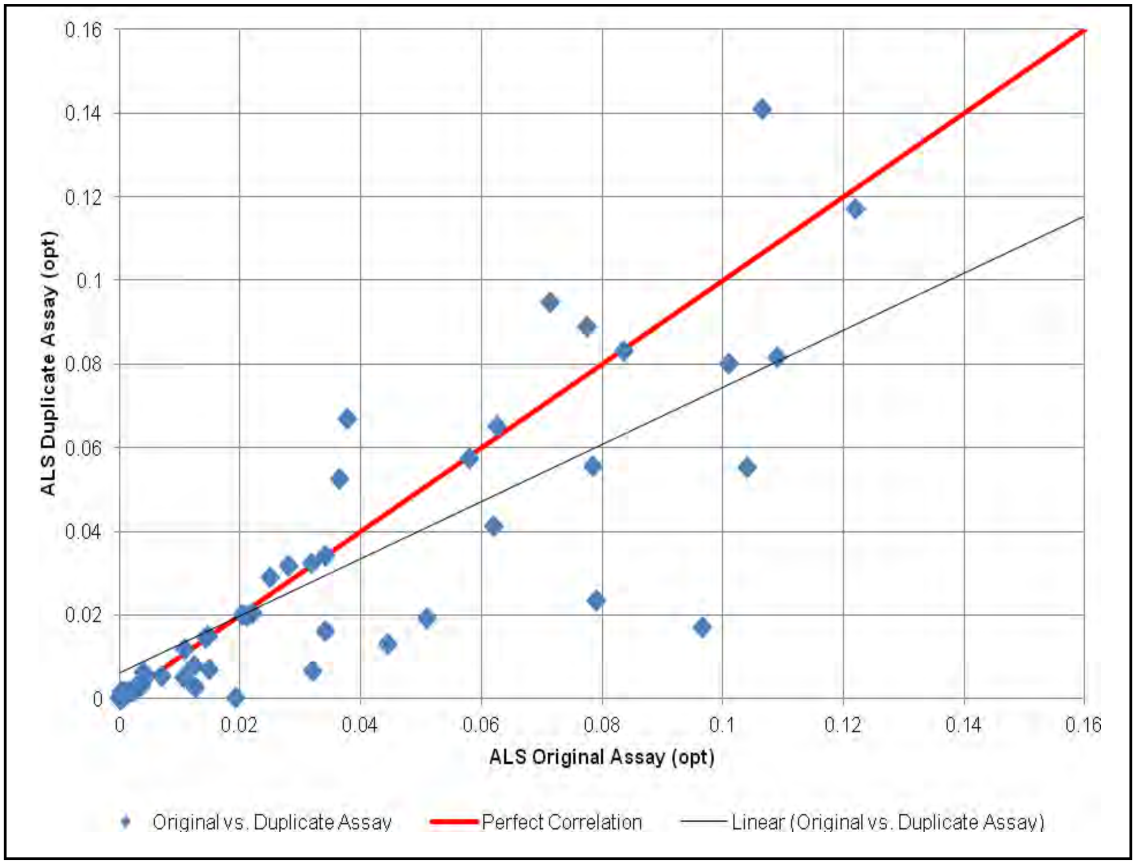


Figure 12.6D: Wilson Deposit Duplicate Assays (Samples < 0.16 opt)

12.4.2 Second Lab Assays

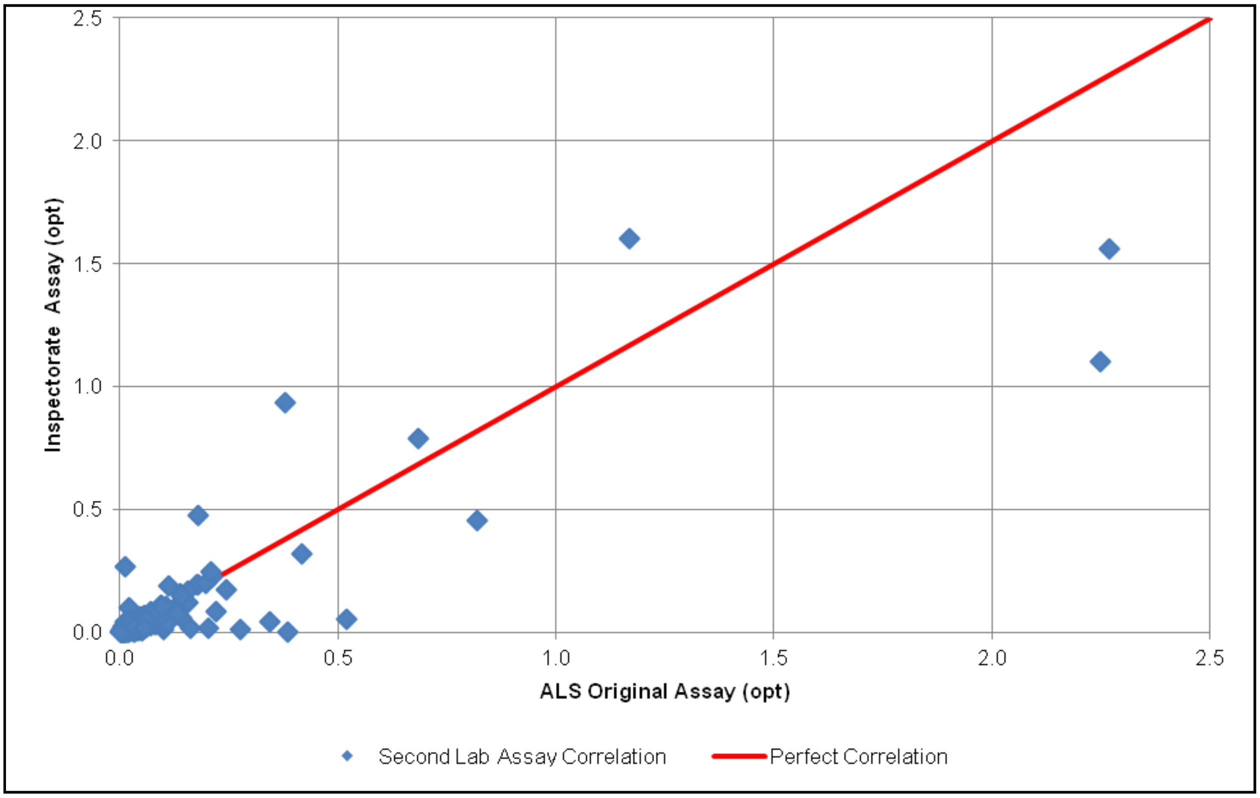


Figure 12.7A: Second Laboratory Assay Comparison

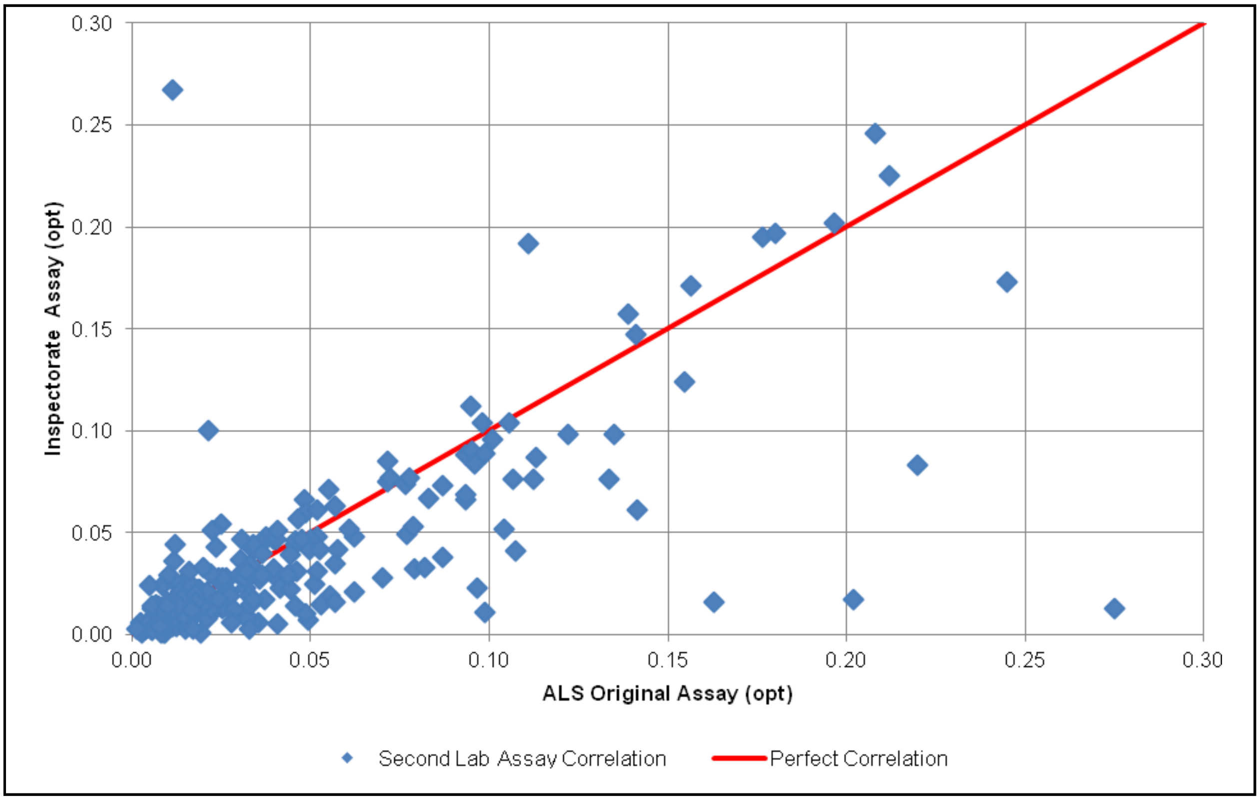


Figure 12.7B: Second Laboratory Assay Comparison (<0.3 opt)

12.4.3 Screen Assays

Lincoln directed ALS Minerals to run screen assay analyses on samples from 11 drill holes (28 samples) with an average grade of 0.21 opt to better understand the relationship of gold and sample particle size. Assay values were reported in Au Total, Au +100 micron (µm) fraction, and Au -100 micron fraction. The analyses indicate that gold at the Pine Grove property tends to occupy the +100 µm fraction and is comparatively less present in the -100 µm fraction. This size fraction comparison indicates that gold at the property tends to display the potential for a nugget effect which could negatively affect the reproducibility of standard bulk assays. **Figures 12.8A** and **12.8B** show the results of the screen assay analyses.

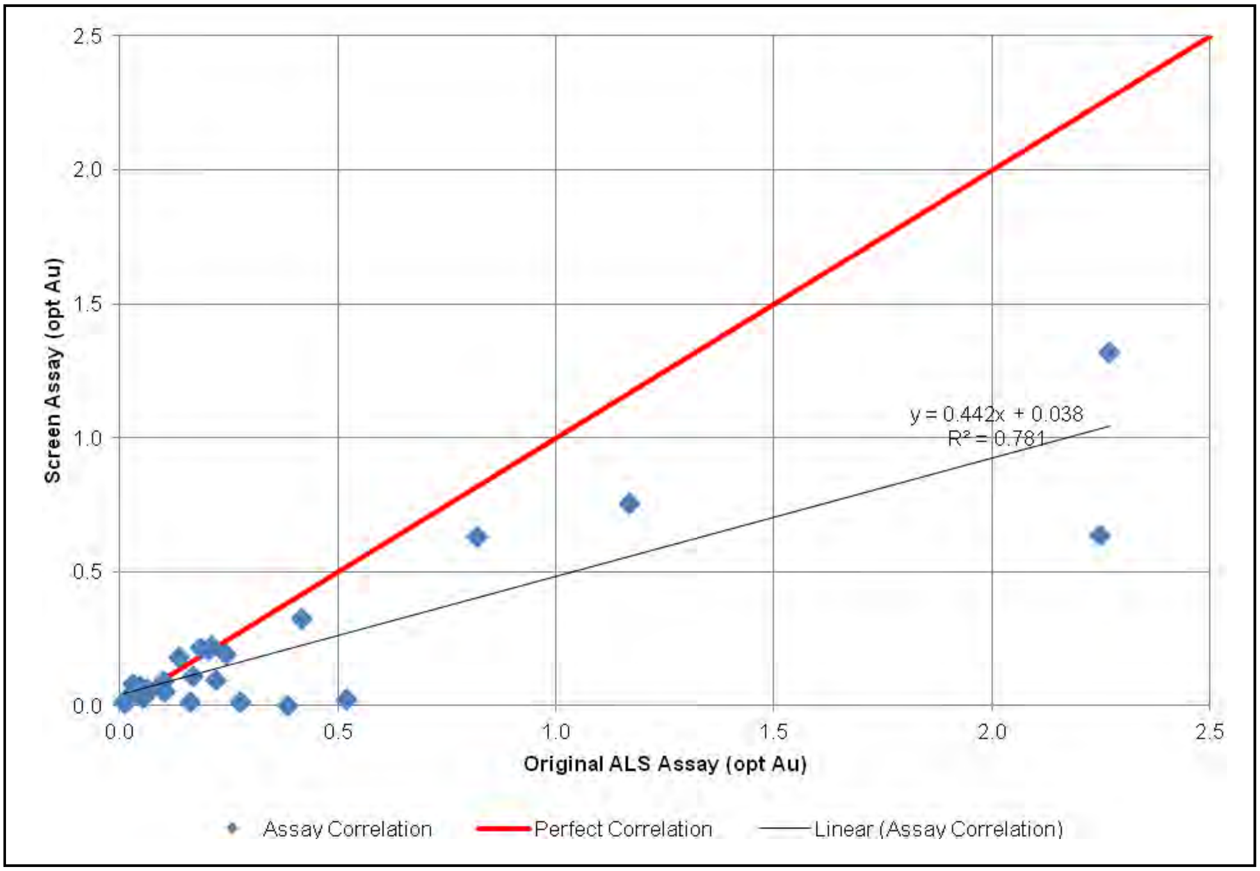


Figure 12.8A: Original Assay/Screen Assay Comparison

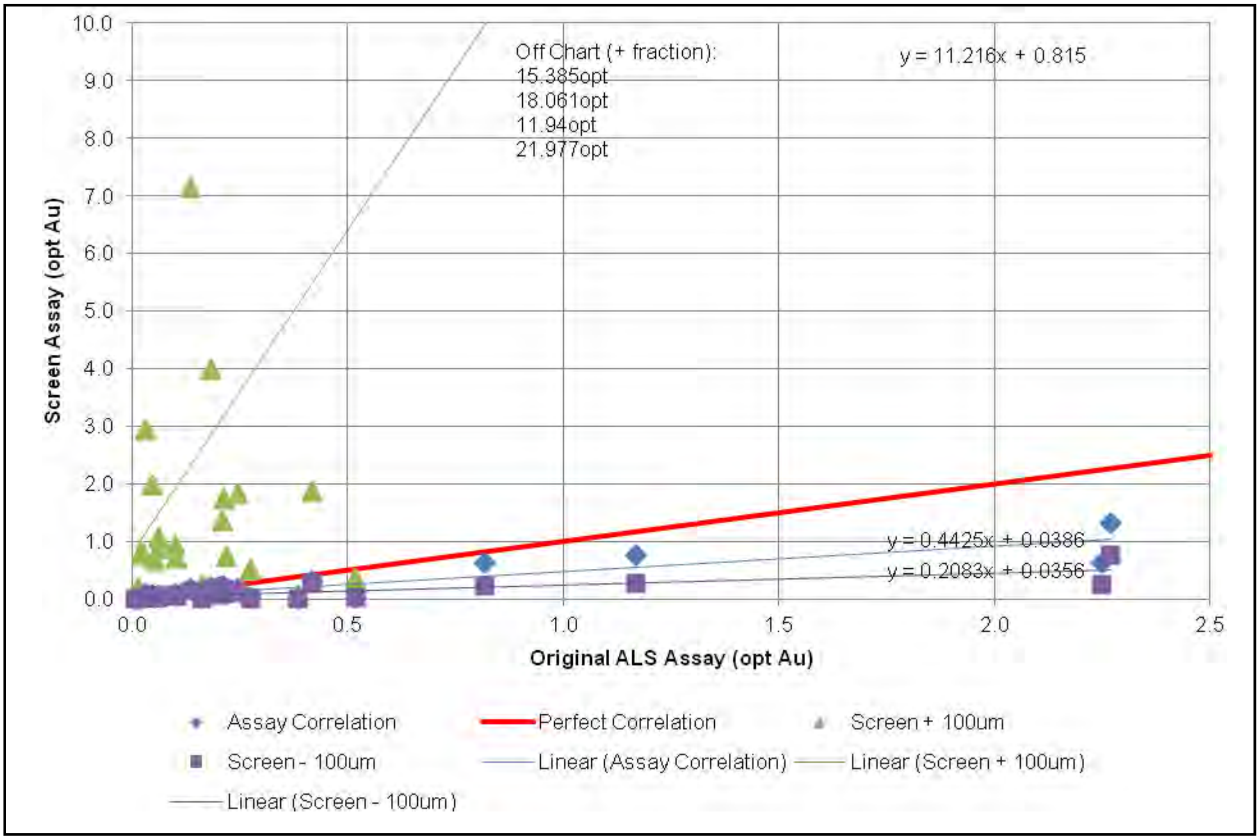


Figure 12.8B: Original Assay/Screen Fraction Assay Comparison

12.4.4 Lincoln Standard and Blank Analyses

Telesto has reviewed the analyses of sample standards and sample blanks that were inserted into the sample stream during the time of drilling. At approximately 100-foot intervals (5%) a standard reference and a blank (as commercially prepared pulps) were inserted into the sample stream. Review of the standard analyses indicates that 95% of all standards are within 3 standard deviations of the certified standard value and 99% are within 5 standard deviations. **Figures 12.9A** through **12.9D** represent scatter charts of Lincoln's standard analyses program. Blank sample analyses indicate an average ALS assay grade of 0.0002 opt Au which is also within industry blank standard tolerances. **Figure 12.10** shows the results of the Lincoln blank assay analyses. The results of the standard and blank analyses indicate that ALS followed a stringent assay analysis protocol during the time of drill sample analyses. It is the Qualified Person's opinion that the standard and blank assay analyses conducted by ALS are reliable and within industry standard tolerances.

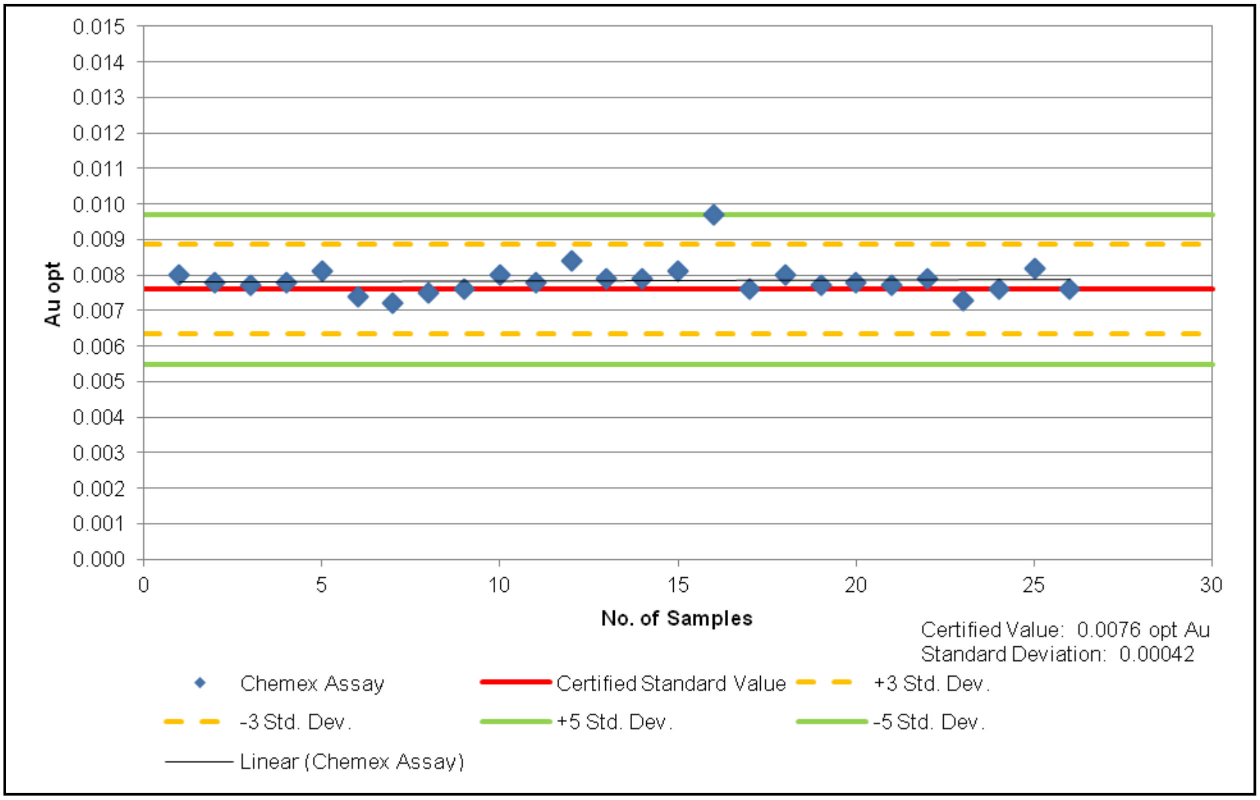


Figure 12.9A: ALS Chemex Standard Analyses (0.008 opt Standard)

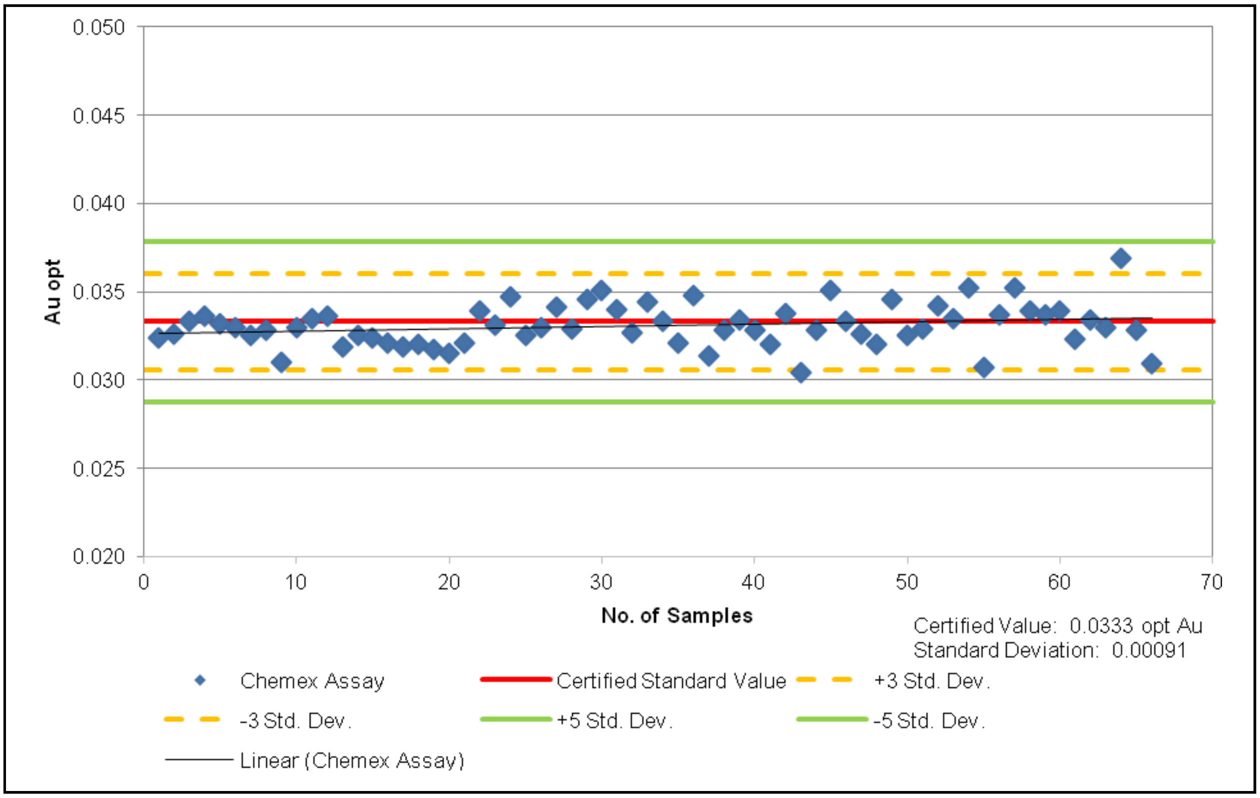


Figure 12.9B: ALS Chemex Standard Analyses (0.033 opt Standard)

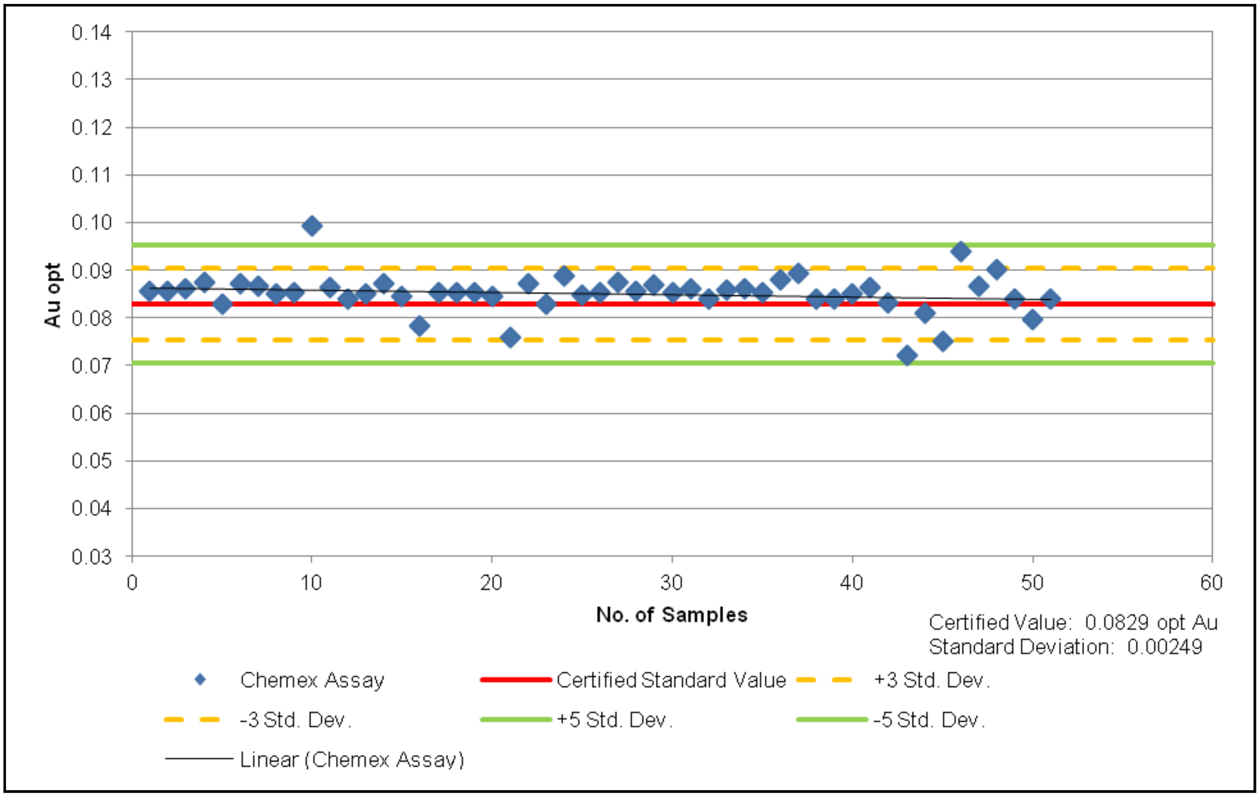


Figure 12.9C: ALS Chemex Standard Analyses (0.083 opt Standard)

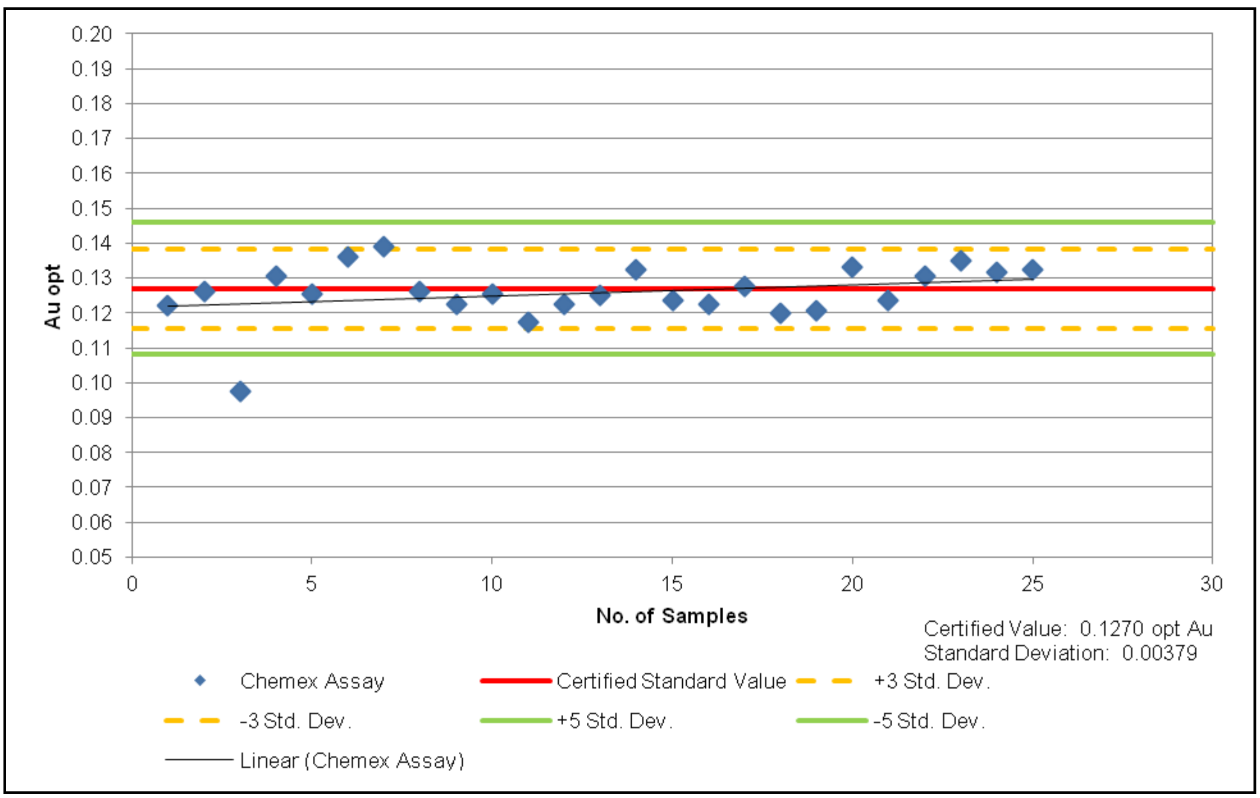


Figure 12.9D: ALS Chemex Standard Analyses (0.127 opt Standard)

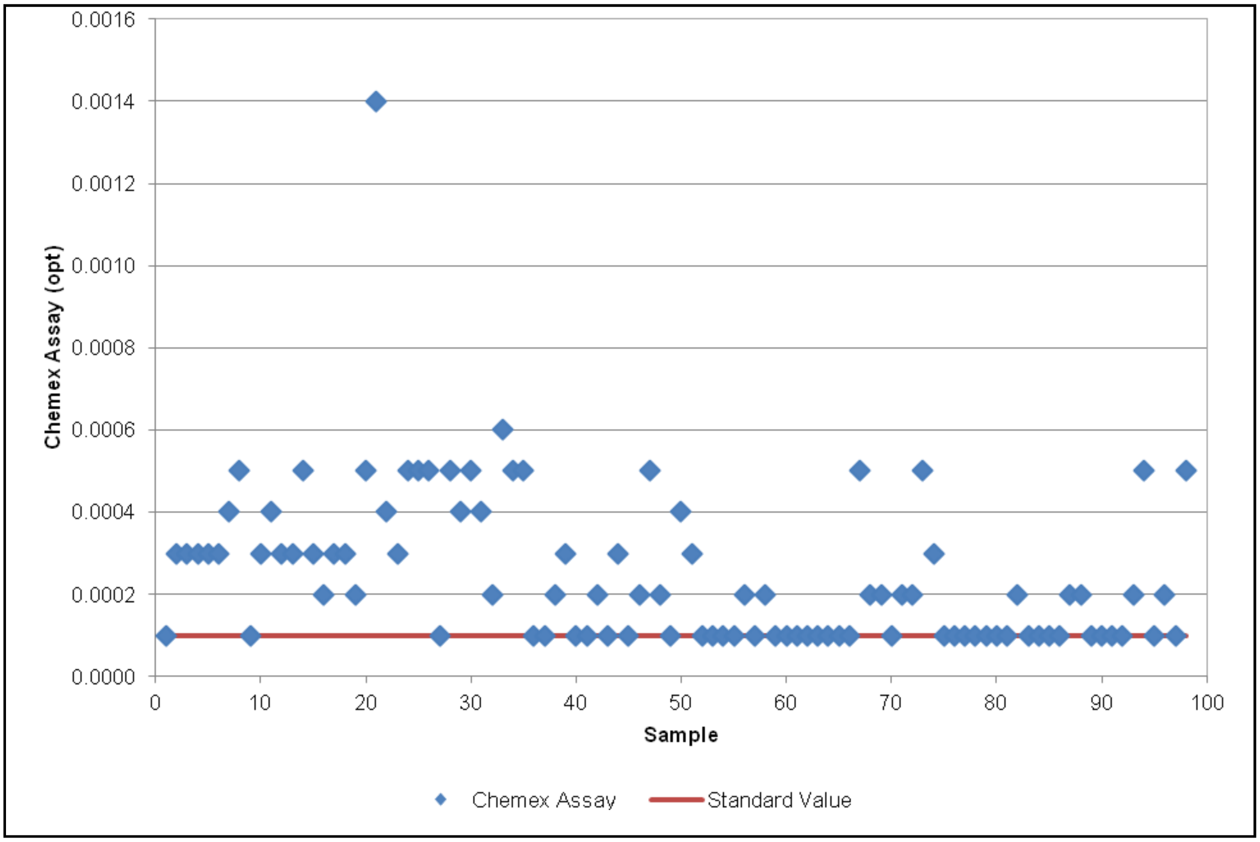


Figure 12.10: ALS Chemex Blank Analyses

12.4.5 Core Drilling vs. RC Drilling Comparison

A comparison of RC and core drilling results was conducted by Telesto to determine the reproducibility of assay values between the two drilling methods.

Two core holes, WR-2A and WR-82A, were drilled on the Wheeler deposit for metallurgical samples. These holes were semi-twins of RC holes WR-2 and WR-82.

- WR-2A: Vertical, total depth 149 ft. Gold mineralization was present in multiple zones throughout the hole. Nearly the entire hole was in highly broken granodiorite. Overall core recoveries were on 70 to 80% with short internal zones of 90 to 100%. The collar of WR-2A is approximately 20 feet from the collar of RC hole WR-2. Down-hole orientations are approximately equal.
- WR-82A: Angle (-45°), total depth 250 ft. Gold mineralization was present in multiple zones throughout the hole. Core recovery in the overlying Morgan Ranch Formation was good at 90 to 100%. Mineralized zones below the Morgan Ranch were badly broken with core recoveries of 50 to 70% with local intervals of 80 to 100%. The collar of WR-82A is approximately 5 feet from the collar of RC hole WR-82. Down-hole orientations are approximately equal.

Two core holes, WL-10A and WR-34A, were drilled on the Wilson deposit for metallurgical samples. These holes were semi-twins of RC holes WL-10 and WL-34.

- WL-10A: Vertical, total depth 199 ft. Sparse gold mineralization was encountered. Core recovery up hole in the rhyolite porphyry was 60 to 80%. Core recovery in the underlying granodiorite was good at 90 to 100%. The collar of WL-10A is approximately 20 feet from the collar of RC hole WL-10. Down-hole orientations are approximately equal.
- WL-34A: Vertical, total depth 201 ft. Core recovery in the overlying rhyolite porphyry was about 60%. The rhyolite was highly broken. Similarly, the "pink" feldspar porphyry was highly broken with recoveries on the order of 60%. Overall core recovery in the granodiorite was good at 90 to 100%. Core recovery in the broken mineralized zones ranged from 50 to 60%. The collar of WL-34A is approximately 33 feet from the collar of RC hole WL-34. Down-hole orientations are approximately equal.

The results of the RC vs. core comparisons are shown of **Figures 12.11A** through **12.11D**.

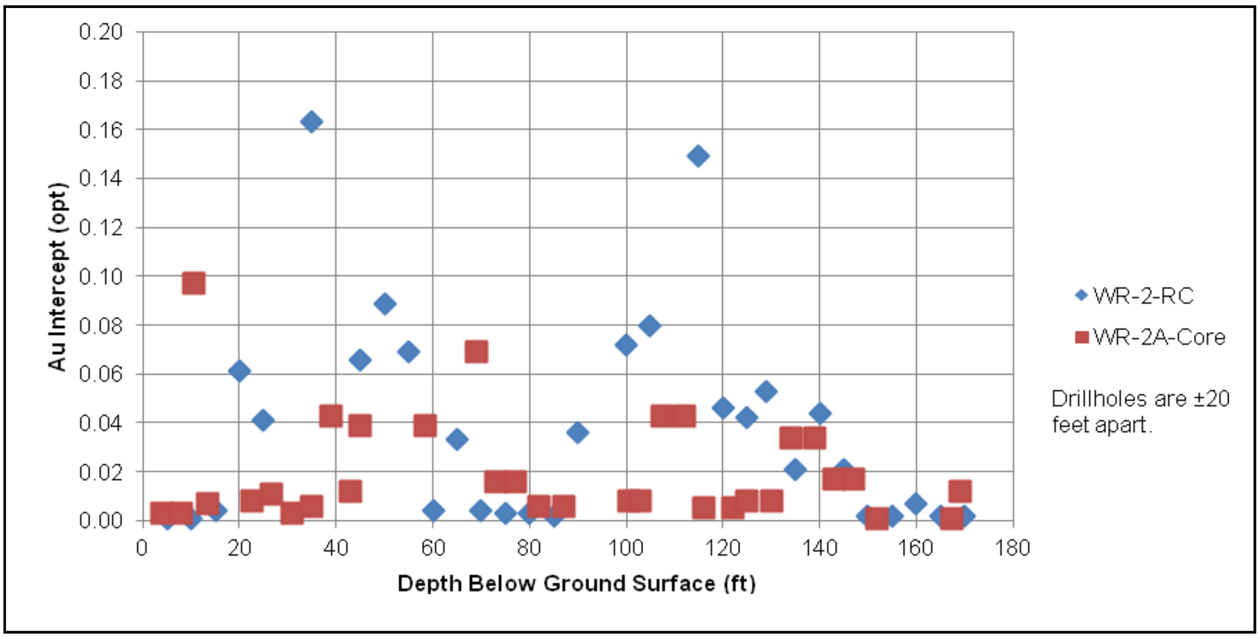


Figure 12.11A: WR-2 (RC) vs. WR-2A (Core) Comparison

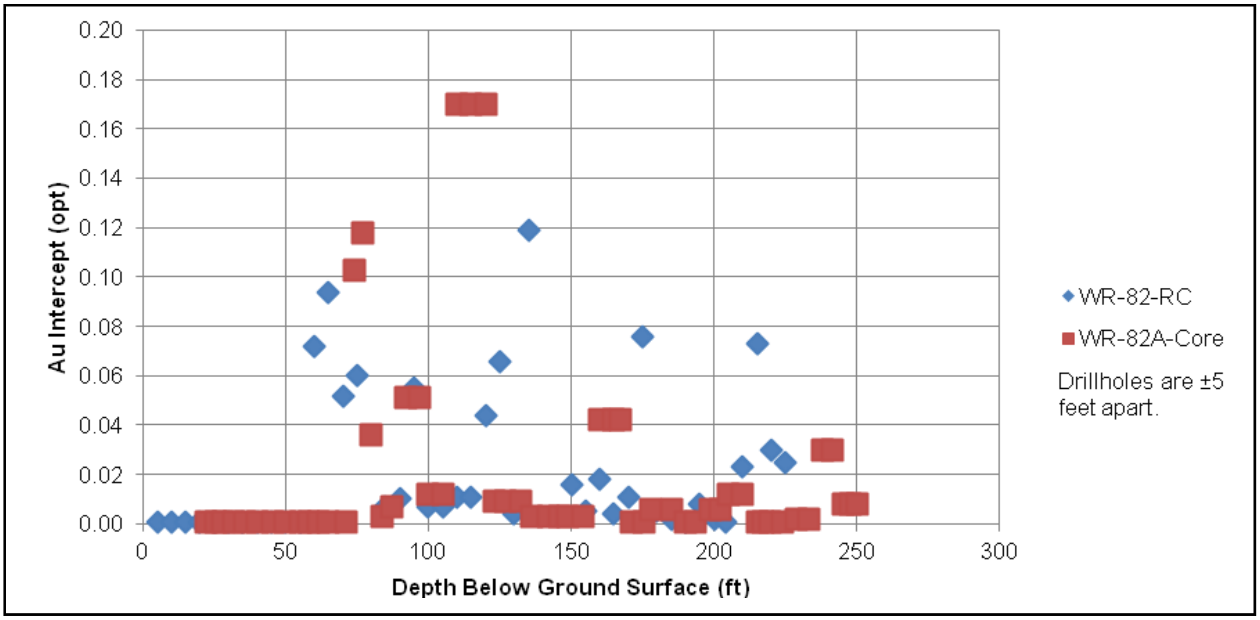


Figure 12.11B: WR-82 (RC) vs. WR-82A (Core) Comparison

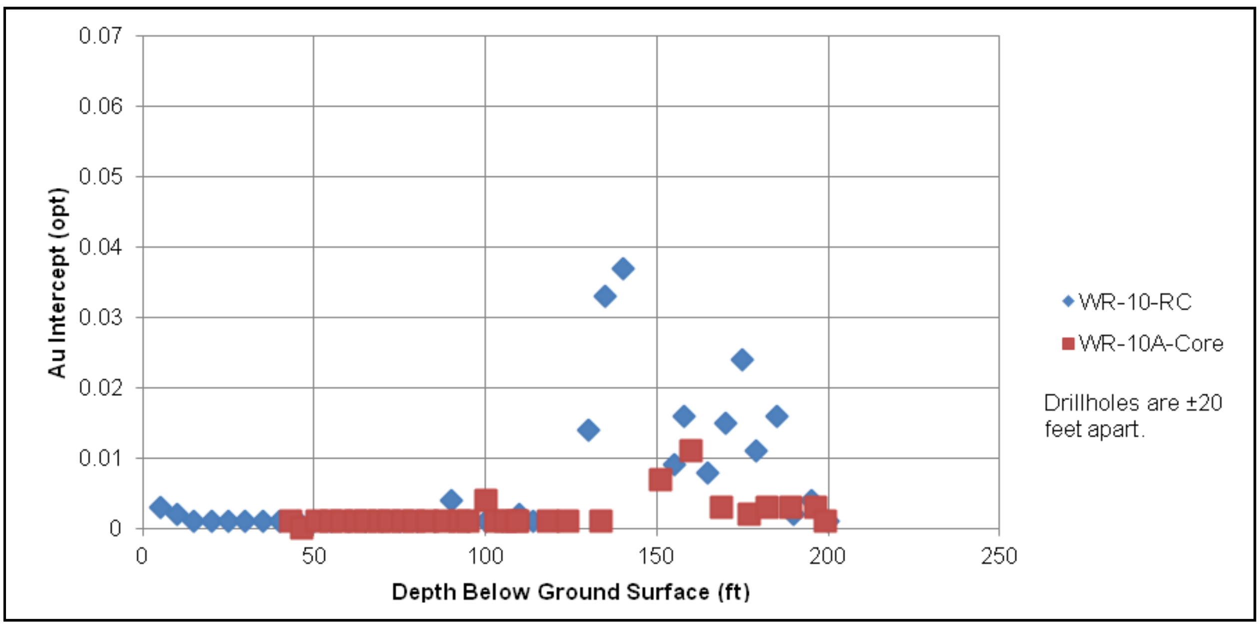


Figure 12.11C: WL-10 (RC) vs. WL-10A (Core) Comparison

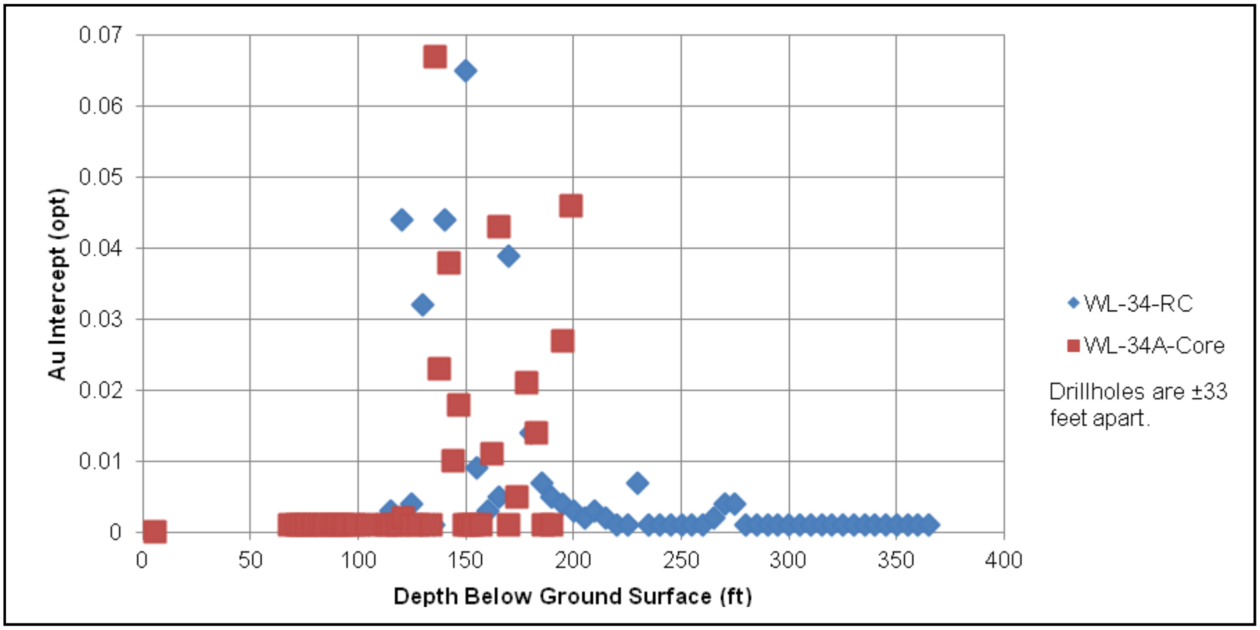


Figure 12.11D: WL-34 (RC) vs. WL-34A (Core) Comparison

The results of the comparisons show that coherent zones of gold mineralization are present in both RC and core hole twins drilled at the Wheeler and Wilson deposits. As is typical of many hydrothermal gold deposits, the results of core vs. RC comparison assays of samples from Pine Grove indicate that the range of precision of the values is wide, but still acceptable. The comparisons suggest that the result reflect the relatively coarse nature of the gold particles and the natural variability of the distribution of the gold particles in the rock, rather than to sampling, preparation, or assaying problems.

12.5 Drill Hole Survey Verification

A detailed line by line comparison of the original drill hole collar location and drill hole orientation (azimuth and dip) with the current database was also conducted by Telesto. All current database drill hole coordinates and orientations have been varified to be accurate. Telesto also conducted a detailed review of drill hole cross-sections to verify the recent digital topography relative to drill hole collar elevations. The results of the review indicate that the drill hole collar locations are in agreement with the digital topographic surface.

No record of any down-hole surveys from any drill program have been identified at the Pine Grove property. The absence of down-hole surveys should not be a significant factor for any of the vertical drill holes as they tend to not deviate greatly. However, the lack of down-hole surveys for the angled holes may slightly limit the confidence level for accuratacy of down-hole assay data locations.

12.6 Field Verification

The Telesto Qualified Person visited the Pine Grove site on June 15, 2011 to gain an understanding of the geologic controls associated with gold mineralization. During the visit, mineralized rock and structural contacts were identified and verified. Mineralized rock consisted of biotized granodiorite with pyrite occurring as fracture coatings and disseminations.

Chalcopyrite was also identified in some road cut exposures. Fault structures consisted of very sharply defined contacts in road cut exposures (**Figure 12.12**). The existence of marked and labeled drill hole collars was also verified by Telesto.

During the field visit in June 2011, the Qualified Person also made a visit to the Lincoln warehouse in Yerington, Nevada. The warehouse was in good condition and fully capable of providing a secure storage facility for drill samples. The existence of drill sample duplicates and drilling standards was also verified.

Figure 12.12: Fault Structure at the Wheeler Deposit

12.7 Statement of Data Adequacy

Telesto has independently checked the data for internal consistency and it is the opinion of the Qualified Person that the data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used in the generation of a resource estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Telesto has reviewed the section on mineral processing and metallurgical testing from the previous NI 43-101 technical report on Pine Grove (Pine Grove Gold Project, Lyon County, Nevada, USA, NI 43-101 Technical Report, dated March 14, 2011) (Tetra Tech, 2011). The Telesto Qualified Person agrees with the description of the mineral processing and metallurgical testing at the Pine Grove Project.

The term "ore" has been used in previous metallurgical investigations and reports that are referenced in this Report section. The term "ore" generally implies that sufficient technical feasibility and economic viability studies have been completed to classify the material as mineral reserve. A Qualified Person has not done sufficient work to classify the mineral resource at Pine Grove as current mineral reserve and the issuer is not treating the mineral resource as mineral reserve. The term "ore" is used to maintain the integrity of the previous metallurgical investigations quoted in this report.

The reader is reminded that the PEA is based on the Project resource model which consists of material in Measured, Indicated and Inferred classifications. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability.

The portions of text below which are in *italics* are from either Tetra Tech (2011) or McPartland (2011), as noted. Information and analysis by the Telesto Qualified Person is not italicized.

13.1 Introduction

Thom Seal, Ph.D., P.E, a Mining, Metallurgy and Mineral Process Engineer of Differential Engineering Inc. was requested by Lincoln Gold US Corp. and Telesto to review the supplied metallurgical portions of reports on the Wheeler and Wilson deposits and determine and recommend gold process options and potential recovery based upon the provided reports. This metallurgical section relies solely on electronic files transferred by e-mail from Telesto of Reno, Nevada. The provided reports that were reviewed originated from other qualified persons or from well-known metallurgical testing laboratories with a strong reputation of providing quality work over the years. The Telesto Qualified Person agrees with the results from the metallurgical testing on samples from the Pine Grove Project. Dr. Seal functioned as a metallurgical manager for Newmont Mining Corp. and has a lengthy history of having McClelland Metallurgical Lab conduct tests on supplied material to design metal extraction processes with accompanied reports similar to the information provided below. Dr. Seal finds the information contained within the reports to be adequate to make recommendations on process and recovery for the Pine Grove Project. In addition, independent drilling and samples for verification of metallurgical testing were not available.

The Pine Grove Gold Project was historically mined for gold and closed in 1915. Gold is found in transitional quartz veins and in thin, crosscutting pyrite-chalcopyrite stockwork veinlets. The property has an extensive underground mining history with significant quantity of material in surface mine dumps. The 1880's mining boom at Pine Grove produced roughly 250,000 ounces in gold. Some 150,000 ounces was produced from the Wilson mine, whereas the remaining 100,000 ounces was produced by the Wheeler mine on the other side of the canyon. During this period some 10,000 feet of underground workings were developed, along with a number of winzes, shafts and adits. (Stone, 2007) (Tetra Tech, 2011)

The primary resource of information on recovery, reagents and crush size used for this report was based upon the recent test work of five column leach tests and 45 bottle roll leach tests conducted by McClelland Laboratories, Inc. in 2010 from metallurgical core drilling conducted and composited by Lincoln Gold US Corp in early 2008. The Wheeler composite cyanide columns tests yielded 74.5% and 87.5% gold recoveries after 141 and 166 days, for the column test P_{80} of $^3/_8$" and 1¼", with high cyanide consumptions of 3.71 and 6.24 lb NaCN per ton mineralized material respectively. The Wilson composite yielded a gold recovery of 62.5% after 164 days of column leaching at a P_{80} of $^3/_8$" with a high cyanide consumption of 5.95 lb NaCN per ton mineralized material respectively. Secondary and tertiary copper minerals in the mineralized material are a concern from that test work which showed that the samples contained copper, which was a cyanide consumer. Additional column leach tests are clearly warranted on representative sample of the deposit. The test work was not directly supervised by Telesto and it is not possible to confirm that the work was done on samples which were representative of the mineral resource.

McClelland Laboratories, Inc. also conducted density and bulk density values on the metallurgical composites from the core drilling in 2008. The Wheeler composite showed an average density of 2.533 g/cm^3 and a bulk density of 158.05 lbs/ft^3 or 12.762 ft^3/ton. The Wilson composite showed an average density of 2.591 g/cm^3 and a bulk density of 161.64 lbs/ft^3 or 12.395 ft^3/ton.

The process flowsheet for this PEA includes a three-stage crushing circuit followed by a heap leach and an activated carbon in a Carbon-in-Column (CIC) system.

13.2 Metallurgical Testing by Prior Operators

Prior to Lincoln Gold US Corp acquiring the Pine Grove Property in 2007 (Stone, 2007), several companies conducted drilling and sampling programs to identify the location, quantity of minable material, grade plus projected metal recovery in the process of developing a open pit mine on the property. The property has an extensive underground mining history with significant quantity of material in surface mine dumps. The 1880's mining boom at Pine Grove produced roughly 250,000 ounces in gold. Grades reportedly averaged 1.40 ounces per ton (opt) at Wilson, and 1.30 opt at Wheeler. During this period some 10,000 feet of underground workings were developed, along with a number of winzes, shafts and adits (Stone, 2007). The following summaries of reports are provided in regards to the extraction of precious metals from these various drilling and sampling campaigns. These reference reports were supplied by Lincoln Gold US Corp for review as electronic files. Additional historical metallurgical reports were referenced, but were not available at this time. In addition, several metallurgical tests were conducted, and reports written on samples that lack description as to date, sample location, type and size of sample, etc.

In September 2009, Mr. J. McPartland, metallurgist for McClelland Laboratories, Inc. of Sparks, Nevada, wrote a memo to provide a summary review of metallurgical testing reports provided to MLI by Lincoln Gold Corp. The following information has been taken primarily from this Metallurgical Review of the Pine Grove Project and Property by McClelland Laboratories' (McPartland, 2009), with information added from the original sited reference reports as available.

J. McPartland of McClelland Labs reviewed five metallurgical reports and one mineralogy report and concluded "heap leaching seems to be the most likely commercial processing option. Overall, the available metallurgical test results indicate significant potential for heap leaching of the Pine Grove ore. Gold recoveries obtained (75% average) by direct agitated cyanidation treatment (bottle roll testing) of drill cuttings samples were typical of many heap leach projects." "The testing reviewed is considered to be very preliminary and limited in scope." "Results from bottle roll tests conducted on drill cuttings samples showed that the samples evaluated were amenable to direct agitated cyanidation treatment at a nominal ¼" feed size. Gold recoveries ranged from 57% to 84% and averaged 75%." And "bottle roll test cyanide consumptions and lime demand tended to be high and variable. These consumptions give cause for concern with respect to heap leaching of the Pine Grove ore." (McPartland, 2009)

"It should be noted that no information concerning the origin of the samples tested was provided. All observations in this memo are based on results from testing of these samples, and rely on the assumption that the samples tested are reasonably representative of the ore to be processed. No representation concerning the suitability of the samples evaluated is intended here." (McPartland, 2009).

In 1987, Mr. W. Cavanaugh of Crown Development and Mining Co, had McClelland Laboratories, Inc. of Sparks NV conduct a cyanide leach via bottle roll test on the Lower Wilson Dump sample of 10 kg at a nominal ½ inch size. No other information on the sample is currently available. The Wilson dump sample LWD-1 shown on **Table 13.1** was amenable to direct cyanidation at a nominal ½" feed size with a gold extraction of 68.1% Cyanide consumption was low; lime requirements were moderate. "The gold extraction rate was fairly rapid, with extraction substantially complete in 24 hours" (Macy, 1987).

Table 13.1: Bottle Roll Tests, McPartland 2009

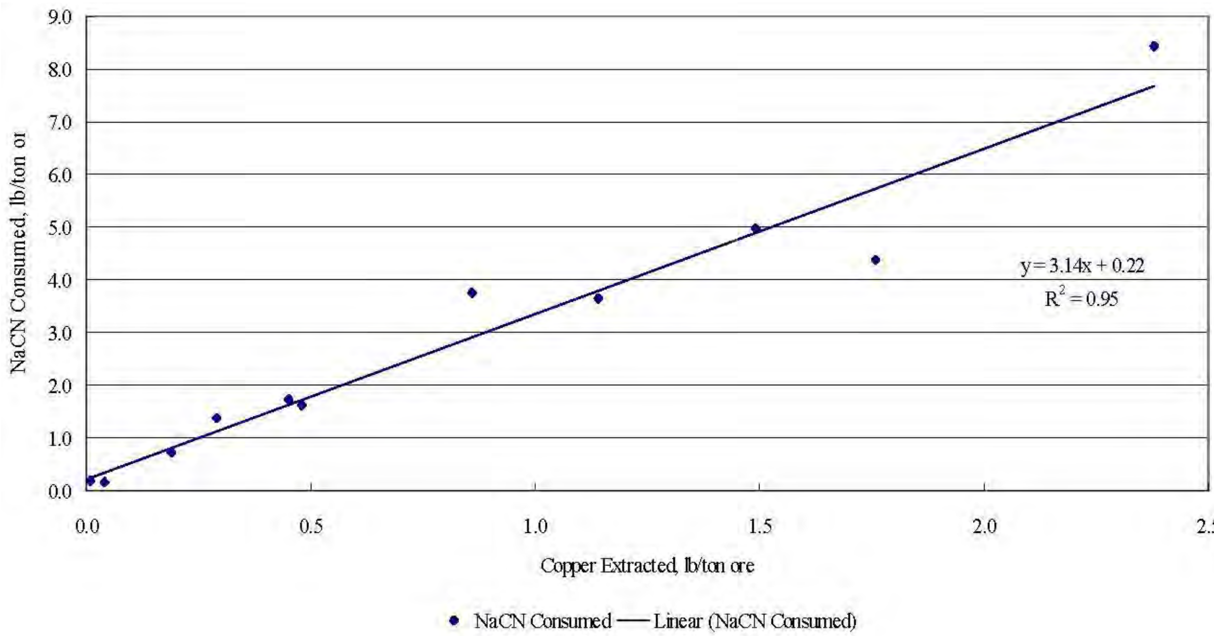
Sample	Mine	Depth, ft	Nominal Crush Size, in	Leach Time, hrs	Au Extraction, %	Calculated Head Grade, oz Au/ton	Cyanide Consumed, lb/ton ore	Lime Addition, lb/ton ore
WR-2	Wheeler	15-20	¼	144	73.7	0.057	0.2	12.4
WR-2	Wheeler	30-35	¼	144	67.1	0.155	0.16	7.5
WR-2	Wheeler	175-180	¼	144	75	0.008	0.73	9.9
WR-2	Wheeler	195-200	¼	144	73.2[a]	0.123	4.37	5.4
WR-13	Wheeler	25-30	¼	144	84.3	0.070	1.61	5.4
WR-13	Wheeler	125-130	¼	144	63.6	0.464	1.74	5.8
WR-13	Wheeler	140-145	¼	144	83.5	0.121	3.75	7.6
WL-51/52	Wilson	85-125, 95-100	¼	144	81.30	0.064	3.65	14.4
WR-13/14	Wheeler	110-125, 10-15	¼	144	82.4	0.017	1.37	19.4
WR-83/85	Wheeler	115-150, 115-120	¼	144	80.9	0.068	4.96	29.8
WR-13	Wilson	100-110	¼	144	57[b]	0.135	8.42	20.8
LWD-1	Wilson		½	96	68.1	0.069	0.9	NA
W-2	Black sand concentrate		-200m	72	99.2	41.92	1.5	NA

a - This sample contained 0.34 ounce silver per ton with a recovery less than 20%.
b - This composite contained 0.26 ounce silver per ton with a recovery of 34.6%.

"A bottle roll test was conducted on one Lower Wilson Dump sample, identified as LWD-l, to determine recovery, recovery rate, and reagent requirements. The sample was amenable to direct cyanidation at a nominal ½" feed size. Gold recovery was 68.1 percent in 96 hours. Gold recovery rate was fairly rapid with the majority of values being extracted in 24 hours. Cyanide consumption (0.9 lbs/ton ore) was low. Lime requirements (7.5 lbs/ton ore) were moderate, and maintaining pH was difficult". The head assay was an average of 0.073 ozAu/ton. "Head assay results, indicate a "spotty" gold occurrence in the ore. "Spottiness" can be caused by contained visible gold or gold enriched in sulfide minerals. The average of the four head assays agreed closely with the calculated head from the bottle roll test." (Macy, 1987).

In addition, a single column leach test was conducted by Western Testing Laboratories of Sparks, Nevada at the request of Crown Development and Mining Co. on a 50 pound Wilson Dump sample (MD-NS-2), screened undersized (-½") material generated at the project site, agglomerated with 10% type II Portland cement (5 lbs), 5 lbs lime and 2.2 cyanide lb/ton, with a gold recovery of 78.0% from a fire assay head sample with 0.058 ozAu/ton and calculated head of 0.0635 ozAu/ton and nil Ag [**Table 13.1** (McPartland, 2009) (Clem, 1983b)]. This metallurgical column test work is the only column test information available on the Wilson property surface dump and is relied upon for metallurgical projections on recovery, reagents and crush size for that resource. It must be noted that it is unknown where and when the sample was collected, or if the sample is representative of the surface dump.

J. McPartland of McClelland Labs states that "cyanide consumptions were strongly correlated to copper extraction." The Telesto Qualified Person agrees with statement on copper extraction and cyanide consumption observed on the testing of the material leached from the Pine Grove Project. (See **Figure 13.1**) "Copper is a known cyanicide." "To whatever degree high cyanide soluble copper ores are present in the Pine Grove Project, high cyanide consumptions during commercial production can be expected." "Optimizing leaching conditions to minimize copper dissolution during cyanidation will probably be important for the project's success." "In the extreme, copper dissolution may also cause sufficient free cyanide depletion to affect gold recovery rate and even ultimate gold recovery." "Milling/cyanidation, milling/gravity concentration and milling/flotation treatment, the most likely alternatives to heap leach processing, were not evaluated in the reports provided." (McPartland, 2009) For future testing, McPartland (2009) recommended and the Telesto Qualified Person agrees that a detailed heap-leaching testing program on the four HQ and PQ samples collected by Lincoln, with waste intervals from the same core to be used for waste-rock characterization testing. We also agree that further recommended preliminary metallurgical testing of samples from known waste dumps and tailings deposits is necessary (McPartland, 2009).



Note: Figure 13.1 is from McPartland (2009). The graph is presented exactly as it appears in McPartland. The Y-axis label should read "NaCN Consumed, lb/ton ore".

Figure 13.1: Cyanide Consumption vs. Copper Dissolution, Bottle Roll Tests, Pine Grove Cuttings Samples, ¼" Feeds

The results from reviewed metallurgical reports consists of 13 bottle roll and a single column cyanidation tests on Pine Grove samples (McPartland, 2009), of which most had been conducted by Teck during their work on the property, and noted that the testing reviewed was considered to be very preliminary and limited in scope. The reports recommended significant future metallurgical testing prior to development of the project. **Table 13.1**, taken from the McPartland (2009) report summarizes the results on these 13 samples. Teck and Atlas Corporation ("Atlas") undertook programs of bottle roll tests on 11 samples of minus ¼-inch rotary drill cuttings (McClelland, 1991; McClelland Laboratories, Inc., 1991). The seven samples labeled WR- on **Table 13.1** appear from hand-written results to have been Teck samples; the four composite samples on **Table 13.1** appear to have been composites of Teck samples that were analyzed at the request of Atlas. The samples were taken from different portions of the deposit, from various depths, and various grades in order to get a representative composite sample. Most of the samples were individual 5-ft assay intervals as indicated in **Table 13.1**, but three were composites from more than one hole (Tetra Tech, 2011).

Composite 1 on **Table 13.1** was from holes WL-51 and WL-52; Composite 2, from WR-13 and WR-14; Composite 3 from WR-83 and WR-85; and Composite 4 was from 100-105 and 105-110 ft from WR-13 (McClelland Laboratories, Inc., 1991). The seven WR- samples all came from the Wheeler mine (McClelland, 1991). Leach times were extended to 144 hours due to the presence of coarse gold, but it was found that the bulk of the gold was in solution within 48 hrs (Tetra Tech, 2011).

In addition to the Teck testing of samples from drill cuttings, additional bottle roll tests were conducted on a waste dump sample from the Wilson mine and a concentrate sample. "A bottle roll test was conducted on a single black sand concentrate sample (W-2) with a grade of 41.92 ozAu/ton and 3.80 ozAg/ton (Macy, 1987) (Clem, 1983a). The concentrate sample W-2 shown on **Table 13.1** was studied at the request of Crown Development and Mining Co. and yielded gold extraction of 99.2% and silver extraction of 99.5% (Clem, 1983a) at a grind of P_{82} of 200 mesh (Macy 1987). Reagent consumption was modest. "No description of the origin of the concentrate sample was provided. Consequently, interpretation of those results is difficult." Normally a "black sand concentrate" results from a gravity concentration unit process operation. A summary of the all of these results is presented in **Table 13.1** (Tetra Tech, 2011).

The recoveries for just Teck's 11 drill cuttings samples range from 57 to 84 percent. Teck concluded there did not appear to be a relationship between recovery and the sample depth. For just the seven Wheeler (WR-) samples, McClelland (1991) noted the following trends:

- Gold and copper recovery was independent of grade.
- Cyanide consumption and consumption rate increased with copper dissolution and copper dissolution rate.
- Lime requirements tended to be high for intervals which consumed small quantities of cyanide.
- Intervals which consumed higher quantities of cyanide required smaller quantities of lime for alkalinity control.
- Based on study of just the Wheeler samples, McClelland (1991) concluded:
    - Wheeler mine cuttings intervals were amenable to agitated cyanidation treatment at the cuttings feed size.
    - Gold recovery rates were generally fairly slow.
    - Copper recovery tended to be low.

- o Cyanide consumptions varied from low to high, and increased with increase in dissolved copper.
- o Lime requirements were moderate to high (Tetra Tech, 2011).

Although a 1990 Teck report (Jackson, 1990) indicated there were plans to take bulk samples from surface exposures of mineralization for column leach tests, Lincoln has reported that this bulk sampling was never done.

13.3 Metallurgical Testing by Lincoln Gold US Corp.

The metallurgical drill campaign conducted in January through February 2008, and the resultant metallurgical testing by McClelland Laboratories, Inc. and reported in 2011 represent the most complete sampling and testing program on the Pine Grove property that can be referenced for this report (McPartland, 2011). The Telesto Qualified Person agrees with the reported results from the metallurgical testing by McClelland Laboratories, Inc. on samples from the Pine Grove Project. Thus this report utilized this information as the basis for reagents, recovery, process flow sheet development, and economic analysis.

> *As of the date of this report, Lincoln Gold US Corp has collected 50 metallurgical samples and completed five column leach tests and 45 bottle-roll leach tests. Additionally, there is a history of both testing and production from the site and certain inferences can be reasonably made with respect to expected process and metallurgical performance. Historical production techniques and prior testing by reputable, independent, third-party laboratories shows the ore to be amenable to cyanide leach technology, and depending on the crush size has shown gold recoveries by straight forward heap leaching in excess of 70 percent. Crushing and grinding to finer sizes increases recovery at an increase in cost. Further study will be required to define the appropriate process to maximize recovery while minimizing costs.* (Tetra Tech, 2011)

The metallurgical test work suggests that gold is recovered from a cyanide leaching process on the deposit's material. The quantity of gold recovery is dependent on both the size distribution or crush size, and the concentration of cyanide, at a normal leach pH, thus smaller particles and higher dosages of cyanide provide the highest gold recovery.

13.3.1 Sample Description and Location

> *Core Drilling – 2008 In January through February 2008, Lincoln drilled four core holes to acquire mineralized material for metallurgical testing. Major Drilling America Inc. ("Major") of Carlin, Nevada, was the drilling contractor, using a truck-mounted LF140 core-rig. Large diameter PQ (85 mm diameter) core and HQ (63.5 mm diameter) core were recovered. Two core holes (WL10A, WL34A) were drilled on the Wilson deposit, and two core holes (WR2A, WR82A) were drilled on the Wheeler deposit for a total of 799 feet. Drilling conditions were extremely difficult due to zones of shattered rock and clays. Mine workings (voids 5 to 7 ft) were encountered in both holes on the Wilson deposit. The core was logged on site, and all core was assayed. Lincoln reports that all of the mineralized core was consumed in five column-leach tests at McClelland Laboratories in Sparks, Nevada.*

After each core run, PQ and/or HQ core was carefully removed from the core barrel by the drill crew and put into waxed cardboard core boxes. Core run intervals were clearly marked on wooden dividers within each box. Both the box and lid were clearly marked with the hole number, box number, and core interval. When full, each core box was tied shut with heavy duty string. After each drill shift, the Lincoln project geologist personally transported the core to a locked storage facility in Yerington, NV. At the storage facility, the core was photographed by the geologist and logged. The core was later transported by Lincoln personnel directly to McClelland Laboratories Inc. ("McClelland") in Sparks, NV. At McClelland, a Lincoln geologist selected 40 hand-sized core specimens of various rock units for density measurements. The geologist also determined intervals for assay. The core was crushed by McClelland to an appropriate size from which splits were sent to ALS Chemex in Reno, NV for gold analyses (fire assay with A.A. finish). Subsequent assay data were used to determine mineralized zones which were composited from the core for column leach testing by McClelland. One core hole from the Wilson deposit, hole WL-10A, did not provided an adequate volume of mineralization for column leach testing. All other holes provided sufficient material for five column leach tests. No intact core survived the metallurgical testing program. Two core holes, WR-2A and WR-82A, were drilled on the Wheeler deposit for metallurgical samples. These holes were semi-twins of RC holes WR-2 and WR-82.

- *WR-2A: Vertical, total depth 149 ft. Gold mineralization was present in multiple zones throughout the hole. Nearly the entire hole was in highly broken granodiorite. Overall core recoveries were on 70 to 80% with short internal zones of 90 to 100%.*
- *WR-82A: Angle (-45°), total depth 250 ft. Gold mineralization was present in multiple zones throughout the hole. Core recovery in the overlying Morgan Ranch Formation was good at 90 to 100%. Mineralized zones below the Morgan Ranch were badly broken with core recoveries of 50 to 70% with local intervals of 80 to 100%.*

Two core holes, WL-10A and WR-34A, were drilled on the Wilson deposit for metallurgical samples. These holes were semi-twins of RC holes WL-10 and WL-24.

1. *WR-10A: Vertical, total depth 199 ft. Sparse gold mineralization was encountered. Core recovery up hole in the rhyolite porphyry was 60 to 80%. Core recovery in the underlying granodiorite was good at 90 to 100%.*
2. *WR-34A: Vertical, total depth 201 ft. Core recovery in the overlying rhyolites porphyry was about 60%. The rhyolite was highly broken. Similarly, the "pink" feldspar porphyry was highly broken with recoveries on the order of 60%. Overall core recovery in the granodiorite was good at 90 to 100%. Core recovery in the broken mineralized zones ranged from 50 to 60%* (Tetra Tech, 2011).

An effort was made to position the core holes in mineralized zones adjacent (±10 ft) to existing RC drillholes completed by Teck. Core hole numbers reflect the adjacent Teck drillhole number with the addition of the letter "A". Core Drilling – 2010 An additional *four, shallow, vertical HQ core holes were completed in December 2010 for metallurgical samples. The drilling contractor was KB Drilling Company, Inc. of Virginia City, NV using a KMB 1.4 Versa Drill mounted on a Hitachi CG70 rubber track chassis and rated at 2,100 ft for PQ core. Two holes were drilled on the Wheeler (WR-131c, WR-132c) and two holes were drilled on the Wilson (WL-104c, WL-105c) for a total footage of 710 ft. Whole diamond drill core (HQ and PQ) from four holes is available for metallurgical testing. Data from these holes were not available at the time of this Technical Report* (Tetra Tech, 2011).

From the core drilling in January through February 2008, “The currently available drill core (4 holes @ - 200' ea.) will be transferred from storage to MLI” (McPartland, 2009). “A total of 140 drill core interval samples were received for initial preparation and interval analysis, and for subsequent preparation of metallurgical composites (3) for heap leach cyanidation testing. Results from interval analyses were reviewed by Lincoln Mining Corp. personnel, and used to generate compositing instructions for metallurgical testing.” (McPartland, 2011) These metallurgical test composites are found in **Tables 13.2** through **13.6**.

From Tetra Tech (2011):

At the end of each drill shift, all samples were removed from the drill site by the project geologist or geotechnician and taken to a secure warehouse and office facility maintained by Lincoln in Yerington, Nevada. At the warehouse, all samples were inventoried and prepared for transport to ALS Chemex in Reno, NV. Upon completion of five to six holes, ALS Chemex picked up the samples and transported them by truck to their lab in Reno. Security of the samples was the responsibility of ALS Chemex once the samples were removed from the Lincoln facility in Yerington. Sample security procedures are very tight at ALS Chemex. All sample rejects and pulps have been returned to Lincoln and are presently stored in Lincoln’s field office-storage facility in Yerington, NV. When no Lincoln personnel are present, the facility gate and building are locked.

Table 13.2: Column Composite Makeup Table, Pine Grove Intervals, Wheeler Deposit 1¼"

	Weight (kg)	Weight %		Assays oz/ton ore	Assays oz/ton ore	Au Distribution		Cu Distribution	
Intervals	To Comp	To Comp	Cumulative	Au	Cu	%	Cum.	%	Cum.
WR-2A 8-10.5	4.00	2.6	2.6	0.0097	0.48	2.6	2.6	1.7	1.7
WR-2A 10.5-13.5	4.78	3.0	5.6	0.007	0.18	0.2	2.8	0.8	2.5
WR-2A 13.5-17.5	4.86	3.1	8.7	0.633	1.05	20.0	22.8	4.5	7.0
WR-2A 17.5-22.5	3.59	2.3	11.0	0.008	0.24	0.2	23.0	0.8	7.8
WR-2A 22.5-26.5	4.29	2.7	13.7	0.011	0.21	0.3	23.3	0.8	8.6
WR-2A 35-39	4.77	3.0	16.7	0.043	1.19	1.3	24.6	4.9	13.5
WR-2A 39-43	5.29	3.4	20.1	0.012	0.83	0.4	25.0	3.9	17.4
WR-2A 43-58.5	5.21	3.3	23.4	0.039	3.41	1.3	26.3	15.6	33.0
WR-2A 58.5-69	2.39	1.5	24.9	0.069	2.47	1.1	27.4	5.1	38.1
WR-2A 69-77	7.24	4.6	29.5	0.016	1.21	0.8	28.2	7.7	45.8
WR-2A 77-87	7.85	5.0	34.5	0.006	0.77	0.3	28.5	5.3	51.1
WR-2A 87-103	5.50	3.5	38.0	0.008	0.87	0.3	28.8	4.2	55.3
WR-2A 103-112	7.09	4.5	42.5	0.378	0.38	17.3	46.1	2.4	57.7
WR-2A 122-130	5.79	3.7	46.2	0.260	0.26	9.8	55.9	1.3	59.0
WR-2A 130-139	7.83	5.0	51.2	0.034	0.43	1.7	57.6	3.0	62.0
WR-2A 139-147	7.38	4.7	55.9	0.408	0.03	19.5	77.1	0.2	62.2
WR-2A 167.5-169	1.27	0.8	56.7	0.144	0.14	1.2	78.3	0.2	62.4
WR-82A 71-74	4.80	3.1	59.8	0.103	0.82	3.2	81.5	3.5	65.9
WR-82A 74-77	3.25	2.1	61.9	0.118	1.26	2.5	84.0	3.7	69.6
WR-82A 77-80	3.92	2.5	64.4	0.036	1.19	0.9	84.9	4.1	73.7
WR-82A 87-97	7.45	4.8	69.2	0.051	0.37	2.5	87.4	2.4	76.1
WR-82A 97-105	7.08	4.5	73.7	0.012	0.31	0.5	87.9	1.9	78.0
WR-82A 105-120	7.12	4.5	78.2	0.170	0.33	7.8	95.7	2.0	80.0
WR-82A 120-132.5	5.81	3.7	81.9	0.009	0.94	0.3	96.0	4.8	84.8
WR-82A 154-167	6.22	4.0	85.9	0.042	0.42	1.7	97.7	2.4	87.2
WR-82A 202-210	6.95	4.4	90.3	0.012	0.57	0.5	98.2	3.5	90.7
WR-82A 233-241.5	7.25	4.6	94.9	0.030	1.09	1.4	99.6	6.9	97.6
WR-82A 241.5-250	8.07	5.1	100.0	0.008	0.34	0.4	100.0	2.4	100.0
	157.04	100.0		0.098	0.72	100.0		100.0	

Note: Table 13.1 is adapted from Table A3-1 in McPartland, 2011.

Of concern on these metallurgical core drill sample collected on January and February 2008 and the respective metallurgical test work concluded in July 2010 is that a significant time had lapsed from the collection of the sample in the winter when precipitation is present and the natural aging or biooxidation of sulfides. The Wilson composite contained 0.49% sulfide sulfur and sulfate sulfur of 0.14% (total sulfur minus sulfide sulfur equals sulfate sulfur) and the Wheeler composite with 0.05% sulfide sulfur and the sulfate sulfur of 0.02%. The Wheeler "mineralized quartz veins contain pyrite with minor chalcopyrite, pyrrhotite, marcasite, and native gold as well as minor rutile and magnetite. Gold occurs as irregular grains from about 0.1 mm to several mm in size. In unoxidized material it is found either in fractures, or on the surface of, pyrite crystals, typically near chalcopyrite grains. It is also found along quartz grain boundaries or as tiny inclusions in pyrite. In oxidized samples, the gold occurs as larger isolated grains in patches of iron oxide. The sulfide content in the veins rarely exceeds 10 percent and is commonly much less." (Tetra Tech, 2011) The Wilson "Gold-bearing veins are similar to veins at the Wheeler" (Tetra Tech, 2011). The analysis of both sulfide and sulfate sulfur indicates the presence of sulfides and the partial oxidization of the composite of approximately 28.6% of the sulfide sulfur in the Wheeler composite and 22.2% of the sulfide sulfur in the Wilson composite. This oxidation of sulfide sulfur does increase the cyanide leaching recovery of precious metals if the precious metals are located in a sulfide matrix. It is unknown if the oxidation of the sulfides in these composites occurred in situ, prior to drilling or occurred due to aging of the sample from the time of drilling and sample collection to the metallurgical test work. If the sample aged and the sulfide sulfur oxidized post drilling and pre-metallurgical testing, all the percent recovery by cyanidation could be biased high and thus inaccurate to represent the properties material in place.

Table 13.3: Column Composite Makeup Table, Pine Grove Intervals, Wheeler Deposit $^3/_8$"

	Weight (kg)	Weight %		Assays		Au Distribution		Cu Distribution	
				oz/ton ore	oz/ton ore				
Intervals	To Comp	To Comp	Cumulative	Au	Cu	%	Cum.	%	Cum.
WR-2A 8-10.5	1.08	2.2	2.2	0.097	0.48	2.0	2.0	1.7	1.7
WR-2A 10.5-13.5	1.45	3.0	5.2	0.007	0.18	0.2	2.2	0.9	2.6
WR-2A 13.5-17.5	1.59	3.3	8.5	0.633	1.05	20.0	22.2	5.5	8.1
WR-2A 17.5-22.5	.99	2.0	10.5	0.008	0.24	0.1	22.3	0.8	8.9
WR-2A 22.5-26.5	1.15	2.4	12.9	0.011	0.21	0.3	22.6	0.8	9.7
WR-2A 35-39	1.56	3.2	16.1	0.043	1.19	1.3	23.9	6.0	15.7
WR-2A 39-43	1.62	3.3	19.4	0.012	0.83	0.4	24.3	4.3	20.0
WR-2A 43-58.5	.24	0.5	19.9	0.039	3.41	0.2	24.5	2.7	22.7
WR-2A 58.5-69	.44	0.9	20.8	0.069	2.47	0.6	25.1	3.5	26.2
WR-2A 69-77	2.28	4.7	25.5	0.016	1.21	0.7	25.8	9.0	35.2
WR-2A 77-87	2.45	5.0	30.5	0.006	0.77	0.3	26.1	6.0	41.2
WR-2A 87-103	1.82	3.7	34.2	0.008	0.87	0.3	26.4	5.0	46.2
WR-2A 103-112	2.30	4.7	38.9	0.378	0.38	17.0	43.4	2.8	49.0
WR-2A 122-130	2.02	4.1	43.0	0.260	0.26	10.2	53.6	1.7	50.7
WR-2A 130-139	2.28	4.7	47.7	0.034	0.43	1.5	55.1	3.2	53.9
WR-2A 139-147	2.80	5.7	53.4	0.408	0.03	22.3	77.4	0.3	54.2
WR-2A 167.5-169	0.31	0.6	54.0	0.144	0.14	0.8	78.2	0.1	54.3
WR-82A 71-74	1.32	2.7	56.7	0.103	0.82	2.7	80.9	3.5	57.8
WR-82A 74-77	.98	2.0	58.7	0.118	1.26	2.3	83.2	4.0	61.8
WR-82A 77-80	1.40	2.9	61.6	0.036	1.19	1.0	84.2	5.4	67.2
WR-82A 87-97	2.71	5.5	67.1	0.051	0.37	2.7	86.9	3.2	70.4
WR-82A 97-105	2.98	6.1	73.2	0.012	0.31	0.7	87.6	3.0	73.4
WR-82A 105-120	2.48	5.1	78.3	0.170	0.33	8.3	95.9	2.6	76.0
WR-82A 120-132.5	1.67	3.4	81.7	0.009	0.94	0.3	96.2	5.1	81.1
WR-82A 154-167	1.52	3.1	84.8	0.042	0.42	1.2	97.4	2.1	83.2
WR-82A 202-210	2.47	5.1	89.9	0.012	0.57	0.6	98.0	4.6	87.8
WR-82A 233-241.5	2.81	5.8	95.7	0.030	1.09	1.7	99.7	9.9	97.7
WR-82A 241.5-250	2.08	4.3	100.0	0.008	0.34	0.3	100.0	2.3	100.0
	48.80	100.0		0.104	0.63	100.0		100.0	

Note: Table 13.2 is adapted from Table A3-2 in McPartland, 2011.

Table 13.4: Column Composite Makeup Table, Pine Grove Intervals, Wilson Deposit 1¼"

				Assays					
	Weight (kg)	Weight %		oz/ton ore	oz/ton ore	Au Distribution		Cu Distribution	
Intervals	To Comp	To Comp	Cumulative	Au	Cu	%	Cum.	%	Cum.
WL-10A 133.5-142	14.04	8.8	8.8	0.142	1.69	21.7	21.7	12.0	12.0
WL-10A 142-151	14.73	9.2	18.0	0.007	0.50	1.1	22.8	3.7	15.7
WL-10A 151-160	14.49	9.1	27.1	0.011	0.42	1.7	24.5	3.1	18.8
WL-34A 128-134	5.48	3.4	30.5	0.067	3.34	4.0	28.5	9.1	27.9
WL-34A 134-135.5	4.71	3.0	33.5	0.023	1.34	1.2	29.7	3.2	31.1
WL-34A 135.5-137.8	5.32	3.3	36.8	0.038	1.06	2.2	31.9	2.8	33.9
WL34A 137.8-140	7.55	4.7	41.5	0.010	1.24	0.8	32.7	4.7	38.6
WL34A 140-144	6.63	4.1	45.6	0.018	0.08	1.3	34.0	0.3	38.9
WL-34A 153-157	12.41	7.8	53.4	0.011	0.30	1.5	35.5	1.9	40.8
WL-34A 157-162	12.56	7.9	61.3	0.043	0.67	5.9	41.4	4.3	45.1
WL34A 170-174	11.85	7.4	68.7	0.021	2.59	2.7	44.1	15.4	60.5
WL34A 174-178	13.49	8.4	77.1	0.014	0.71	2.0	46.1	4.8	65.3
WL34A 186.2-190	13.07	8.2	85.3	0.027	2.19	3.8	49.9	14.5	79.8
WL-34A 190-195	12.67	7.9	93.2	0.046	1.28	6.3	56.2	8.1	87.9
WL34A 195-198	10.90	6.8	100.0	0.371	2.21	43.8	100.0	12.1	100.0
	159.9	100.0		0.058	1.24	100.0		100.0	

Note: Table 13.3 is adapted from Table A3-4 in McPartland, 2011.

Table 13.5: Column Composite Makeup Table, Pine Grove Intervals, Wilson Deposit $^3/_8$"

				Assays					
	Weight (kg)	Weight %		oz/ton ore	oz/ton ore	Au Distribution		Cu Distribution	
Intervals	To Comp	To Comp	Cumulative	Au	Cu	%	Cum.	%	Cum.
WL-10A 133.5-142	4.68	8.5	8.5	0.142	1.69	20.0	20.0	11.5	11.5
WL-10A 142-151	6.16	11.2	19.7	0.007	0.50	1.3	21.3	4.5	16.0
WL-10A 151-160	5.63	10.3	30.0	0.011	.042	1.9	23.2	3.5	19.5
WL-34A 128-134	1.55	2.8	32.8	0.067	3.34	3.1	26.3	7.5	27.0
WL-34A 134-135.5	1.04	1.9	34.7	0.023	1.34	0.7	27.0	2.0	29.0
WL-34A 135.5-137.8	1.24	2.3	37.0	0.038	1.06	1.5	28.5	2.0	31.0
WL34A 137.8-140	1.98	3.6	40.6	0.010	1.24	0.6	29.1	3.6	34.6
WL34A 140-144	1.43	2.6	43.2	0.018	0.08	0.8	29.9	0.2	34.8
WL-34A 153-157	4.15	7.6	50.8	0.011	0.30	1.4	31.3	1.8	36.6
WL-34A 157-162	5.90	10.8	61.6	0.043	0.67	7.7	39.0	5.8	42.4
WL34A 170-174	5.17	9.4	71.0	0.021	2.59	3.3	42.3	19.6	62.0
WL34A 174-178	3.71	6.8	77.8	0.014	0.71	1.6	43.9	3.9	65.9
WL34A 186.2-190	4.13	7.5	85.3	0.027	2.19	3.4	47.3	13.2	79.1
WL-34A 190-195	3.85	7.0	92.3	0.046	1.28	5.3	52.6	7.2	86.3
WL34A 195-198	4.19	7.7	100.0	0.371	2.21	47.4	100.0	13.7	100.0
	54.81	100.0		0.060	1.24	100.0		100.0	

Note: Table 13.4 is adapted from Table A3-5 in McPartland, 2011.

Table 13.6: Column Composite Makeup Table, Pine Grove Intervals, Wheeler Surface Deposit $^3/_8$"

	Weight (kg)	Weight %		Assays		Au Distribution		Cu Distribution	
				oz/ton ore	oz/ton ore				
Intervals	To Comp	To Comp	Cumulative	Au	Cu	%	Cum.	%	Cum.
WR-2A 8-10.5	4.03	7.9	7.9	0.097	0.48	8.4	8.4	3.4	3.4
WR-2A 10.5-13.5	4.97	9.8	17.7	0.007	0.18	0.7	9.1	1.6	5.0
WR-2A 13.5-17.5	5.22	10.3	28.0	0.633	1.05	71.2	80.3	9.7	14.7
WR-2A 17.5-22.5	3.85	7.6	35.6	0.008	0.24	0.7	81.0	1.7	16.4
WR-2A 22.5-26.5	4.49	8.9	44.5	0.011	0.21	1.1	82.1	1.6	18.0
WR-2A 35-39	5.77	11.4	55.9	0.043	1.19	5.3	87.4	12.1	30.1
WR-2A 39-43	6.17	12.2	68.1	0.012	0.83	1.6	89.0	9.0	39.1
WR-2A 43-58.5	5.51	10.9	79.0	0.039	3.41	4.6	93.6	33.1	72.2
WR-2A 58.5-69	2.33	4.6	83.6	0.069	2.47	3.5	97.1	10.1	82.3
WR-2A 69-77	8.33	16.4	100.0	0.016	1.21	2.9	100.0	17.7	100.0
	50.67	100.0		0.092	1.12	100.0		100.0	

Note: Table 13.5 is adapted from Table A3-3 in McPartland, 2011.

13.3.2 Testing Procedures

13.3.2.1 *Sample Preparation*

Core from four metallurgical holes drilled by Lincoln in January and February 2008 was submitted to McClelland Laboratories in Reno, Nevada, for heap-leach cyanidation testing and environmental characterization. “A total of 140 drill core interval samples were received for interval preparation and analysis. Pieces of drill core (40) were selected by Lincoln personnel for bulk density determinations. Those core pieces were removed by hand for testing. After bulk density determinations were completed, the core intervals used were returned to the respective drill core intervals for preparation and assaying procedures.”

Each of the 140 drill core interval samples was stage crushed in entirety to 80% -1½" (100% -2") in size. Crushed intervals were blended by coning and were quartered to obtain approximately $^1/_3$ of each interval for finer crushing. Each $^1/_3$ interval sample was stage crushed to 80% -$^3/_8$" (100% -$^5/_8$") in size. Each $^3/_8$" interval sample was blended and split to obtain approximately 1 kg for analysis.

A 1 kg split from each of the drill core interval samples was submitted to ALS assay laboratory for fire assay to determine total gold content, a four acid digest/A.A. finish assay procedure to determine silver and copper content and a cyanide shake procedure to determine cyanide soluble gold and copper content.

Results from interval analyses were reviewed by Lincoln Mining Corp. personnel, and used to generate compositing instructions for metallurgical testing. Select Wheeler interval samples at the 1¼" feed size were each blended by coning and were quartered to obtain an approximately one-half split for preparation of a 1¼" feed size column test composite. The Wheeler 1¼" column test composite was thoroughly blended by repeated coning and was quartered to obtain 125 kg for a column leach test, 20 kg for a head screen analysis and approximately 10 kg for finer crushing. The 10 kg split was stage crushed to 80% -10M in size and was blended and split to obtain 1 kg for a bottle roll test and triplicate samples for head assay.

Sample saved at the $^3/_8$" feed size from the same Wheeler interval samples were each blended and split to obtain an approximately one-half split for preparation of a $^3/_8$" feed size column test composite. The $^3/_8$" feed size column test composite was thoroughly blended and split to obtain approximately 35 kg for a column test and 15 kg for a head screen analysis.

Remaining interval samples at the 1¼" feed size (from the Wheeler drill hole WR-2A, 8-77') were combined with remaining $^3/_8$" feed size material (from the same intervals) to produce the Wheeler Surface composite. The Wheeler Surface composite was stage crushed to 80% -$^3/_8$" in size, and was blended and split to obtain approximately 35 kg for a column test and 15 kg for a head screen analysis.

Based on interval analysis results, composite make-up instructions were provided for a Wilson Deposit composite. All available drill core interval samples at the 1¼" feed size from the select Wilson intervals were combined to produce the Wilson 1¼" column test composite. The 1¼" column test composite was thoroughly blended by repeated coning and was quartered to obtain 125 kg for a column leach test, 20 kg for a head screen analysis and approximately 10 kg for finer crushing. The 10 kg split was stage crushed to 80% -10M in size and was blended and split to obtain 1 kg for a bottle roll test and triplicate samples for head assay.

Samples saved at the $^3/_8$" feed size from the same Wilson interval samples were combined in entirety to produce the $^3/_8$" feed size Wilson column test composite. The $^3/_8$" feed size column test composite was thoroughly blended and split to obtain approximately 35 kg for a column test and 15 kg for a head screen analysis.

Composite head samples from the Wilson and Wheeler composites were submitted for triplicate fire assay to determine gold content and triplicate four acid digest/A.A. analyses to determine silver and copper content. A single head sample from each of those two composites was also submitted for a multi-element ICP scan, a "classical whole rock" analysis and sulfur speciation analyses. Direct head assays were not performed on the Wheeler Surface composite because of sample availability limitations. Head assay results and head grade comparisons are presented in the McClelland Lab report (McPartland, 2011).

13.3.2.2 *Bottle Roll Test Procedures and Results*

Direct agitated cyanidation (bottle roll) tests were conducted on the Wheeler Deposit and Wilson Deposit composites at an 80% -10M feed size to determine gold, silver and copper recovery, recovery rates and reagent requirements.

Ore charges (~1 kg ea.) were mixed with water to achieve 40% weight percent solids. Natural pulp pHs were measured. Lime was added to adjust the pH of the pulps to between 10.5 and 11.0 before adding the cyanide. Sodium cyanide, equivalent to 2 lbs NaCN per ton of solution, was added to the alkaline pulps.

Leaching was conducted by rolling the pulps in bottles on laboratory rolls for 96 hours. Rolling was suspended briefly after 2, 6, 24, 48, 72 and 96 hours to allow the pulps to settle so samples of pregnant solution could be taken for gold, silver and copper analysis by A.A. methods.

Pregnant solution volumes were measured and sampled. Make-up water, equivalent to that withdrawn, was added to the pulps. Cyanide concentrations were restored to initial levels. Lime was added when necessary, to maintain the leaching pH between 10.5 and 11.0. Rolling was then resumed.

After 96 hours, the pulps were filtered to separate liquids and solids. Final pregnant solution volumes were measured and sampled for gold, silver and copper analysis. Final pH and cyanide concentrations were determined. Leached residues were filtered, dried and assayed in triplicate to determine residual precious metal content (McPartland, 2011).

13.3.2.3 *Column Percolation Leach Test Procedures and Results*

Column percolation leach tests were conducted on the Wheeler and Wilson composites at 80% -1¼" and $^3/_8$" feed sizes to determine gold, silver and copper recovery, recovery rate, reagent requirements and sensitivity to feed size under simulated heap leaching conditions. A column percolation leach test was also conducted on the Wheeler Surface composite at an 80% -$^3/_8$" feed size.

Lime was mixed with the dry ore charges before column loading procedures. Lime additions were based on bottle roll test lime requirements. Charges were placed into the 8" I.D. x 10' (for the 1¼" feeds) and 4" I.D. x 10' (for the $^3/_8$" feeds) PVC leaching columns before applying leach solution. Charges were placed into the columns in a manner to minimize particle segregation and compaction.

Leaching was conducted by applying cyanide solution (2.0 lb NaCN/ton solution) over the charges at a rate of 0.005 gallons per minute (gpm)/ft^2 of column cross-sectional area. Pregnant effluent solutions were collected each 24 hour period. Pregnant and barren solution volumes were measured by weighing, and samples were taken for gold, silver and copper assay using conventional A.A. methods. Cyanide concentration and pH were determined for each pregnant solution. Pregnant solutions were pumped through a three stage carbon absorption circuit for recovery of dissolved precious metal values. Barren solution, with appropriate make-up reagents, was applied to the ore charges daily. After leaching, a one liter sample of the final pregnant solution was taken from select column tests and submitted to Western Environmental Testing (WET) Laboratory for a Profile II (water quality panel) analysis.

After leaching, fresh water rinsing was conducted to remove residual cyanide (County requirement) and to recover dissolved precious metal values. Moisture required to saturate the ore charges (in process solution inventory) and retained moistures were determined. Apparent bulk densities were measured before and after leaching.

For select column tests, the rinse effluent was analyzed for WAD cyanide to assist in determining the rinse down characteristics of the leached residues. Fresh water was applied, at the same rate used for leaching, until the effluent WAD cyanide concentration of 0.20 mg/L was achieved. A one liter sample of the final rinse effluent from these columns was taken and submitted to WET Lab for a Profile II (water quality panel) analysis.

Drain down tests were conducted after rinsing was complete. Tests were conducted by terminating solution application, and at that time, measuring drain down volume. Drain volumes were collected and measured periodically by weighing until drain down was complete.

After leaching, rinsing and draining, the residues were removed from the columns and moisture samples taken immediately. The remaining leached residues were air dried and split to obtain a sample for tail screen analysis, and in the case of the 9.5mm residues for a load/permeability test.

The three 9.5mm residue splits taken for load/permeability testing were composited, and sent to AMEC for testing. Tail screens were conducted using the same procedure and size fractions as for the head screens to determine residual precious metal content and to obtain recovery by size fraction data. For select columns, a split of the column leached residue was taken for Mod acid-base accounting (ABA) and Meteoric Water Mobility Procedure (MWMP) (McPartland, 2011).

13.3.3 Density Determinations

Core from four metallurgical holes drilled by Lincoln in February 2008 was submitted to McClelland Laboratories in Reno, Nevada, for heap-leach cyanidation testing and environmental characterization (Tetra Tech, 2011). A total of 37 density measurements have been made on different rock units (Tetra Tech, 2011).

The Pine Grove drill composites have density, saturated % moisture and drain down moisture as shown in **Tables 13.7** and **13.8**.

Table 13.7: Bulk Density Test Results, Wilson and Wheeler Core Samples

Sample ID	Depth (feet)	Specific Gravity	Bulk Density (lb/ft^3)	ft^3/ton	Rock Type	Rock Code
WR-2A-Box #17	103.8-104.5	2.58	161.0	12.4	Granodiorite	14
WR-2A-Box #20	137.5-138.0	2.64	164.7	12.1	Granodiorite with sulfides	143
WR-2A-Box #22	150.0-150.5	2.67	166.6	12.0	Granodiorite with sulfides	143
WR-2A-Box #23	168.3-168.5	3.09	192.8	10.4	Granodiorite with sulfides	143
WL-10A-Box #1	41.5-42.0	2.62	163.5	12.2	Colluvium	1
WL-10A-Box #4	53.5-54.0	2.58	161.0	12.4	Rhyolite porphyry dike	11
WL-10A-Box #8	68.7-69.0	2.55	159.1	12.6	Rhyolite porphyry dike	11
WL-10A-Box #12	82.0-82.5	2.58	161.0	12.4	Rhyolite porphyry dike	11
WL-10A-Box #17	100.5-101.0	2.58	161.0	12.4	Rhyolite porphyry dike	11
WL-10A-Box #22	135.5-136.0	2.56	159.7	12.5	Granodiorite with sulfides	143
WL-10A-Box #24	157.6-157.9	2.76	172.2	11.6	Granodiorite with sulfides	143
WL-10A-Box #27	187.0-187.3	2.68	167.2	12.0	Granodiorite with sulfides, faulted	1438
WR-2A-Box #5	17.0-17.5	2.64	164.7	12.1	Granodiorite	14
WR-2A-Box #9	34.0-34.5	2.42	151.0	13.2	Granodiorite	14
WR-2A-Box #12	57.5-58.0	2.46	153.5	13.0	Granodiorite with quartz veins	16
WR-2A-Box #15	84.0-84.5	2.57	160.4	12.5	Granodiorite	14
WR-82A-Box #2	23.0-23.5	2.17	135.4	14.8	Morgan Ranch Fm., sandstone	6

Table 13.7: Bulk Density Test Results, Wilson and Wheeler Core Samples

Sample ID	Depth (feet)	Specific Gravity	Bulk Density (lb/ft^3)	ft^3/ton	Rock Type	Rock Code
WR-82A-Box #5	35.0-35.5	2.13	132.9	15.0	Morgan Ranch Fm., sandstone	6
WR-82A-Box #8	45.5-46.3	2.21	137.9	14.5	Morgan Ranch Fm., sandstone	6
WR-82A-Box #12	62.5-63.3	2.41	150.4	13.3	Granodiorite	14
WR-82A-Box #18	84.0-84.8	2.45	152.9	13.1	Granodiorite	14
WR-82A-Box #21	106.0-106.3	2.63	164.1	12.2	Granodiorite	14
WR-82A-Box #25	165.7-166.0	2.19	136.7	14.6	Granodiorite	14
WR-82A-Box #27	176.8-177.3	2.40	149.8	13.4	Granodiorite	14
WL-10A-Box #29	197.5-198.0	2.74	171.0	11.7	Granodiorite with sulfides	143
WL-34A-Box #15	68.8-69.0	2.12	132.3	15.1	Granodiorite	14
WL-34A-Box #17	74.0-74.5	2.59	161.6	12.4	Rhyolite porphyry dike	11
WL-34A-Box #21	85.0-85.5	2.56	159.7	12.5	Rhyolite porphyry dike	11
WL-34A-Box #25	94.0-94.5	2.54	158.5	12.6	Rhyolite porphyry dike	11
WL-34A-Box #28	114.5-114.8	2.62	163.5	12.2	Rhyolite porphyry dike	11
WL-34A-Box #32	126.7-127.0	2.65	165.4	12.1	Rhyolite porphyry dike	11
WL-34A-Box #39	148.0-148.5	2.60	162.2	12.3	Granodiorite with sulfides	143
WL-34A-Box #41	156.0-156.4	2.56	159.7	12.5	Granodiorite with sulfides	143
WL-34A-Box #46	175.7-176.0	2.67	166.6	12.0	Granodiorite with sulfides	143
WL-34A-Box #51	197.5-197.8	2.66	166.0	12.0	Granodiorite with sulfides	143
WR-82A-Box #28	189.5-190.0	2.72	169.7	11.8	Andesite dike	9
WR-82A-Box #29	200.0-200.5	2.70	168.5	11.9	Andesite dike	9
WR-82A-Box #33	232.6-233.0	2.57	160.4	12.5	Granodiorite with sulfides	143
WR-82A-Box #35	244.5-245.0	2.72	169.7	11.8	Granodiorite with sulfides	143
WR-82A-Box #35	246.8-247.1	2.82	176.0	11.4	Basalt dikes or flows with sulfides	53
Average		**S.G.**	**lb/ft^3**	**ft^3/ton**		
		2.56	**159.8**	**12.6**		
McClelland Laboratories, Inc. MLI Job No. 3376 – May 10, 2011 Note: Table 13.6 is adapted from Table A2-1 in McPartland, 2011						

Table 13.8: Physical Ore Characteristic Data, Column Leach Tests, Wilson and Wheeler Composites

Sample Designation	Feed Size	Test No.	Ore Charge (lbs)	Moisture, wt. % As Rec'd	Moisture, wt. % To Saturate*	Moisture, wt. % Retained	Apparent Bulk Density, lbs ore/ft^3 Before	Apparent Bulk Density, lbs ore/ft^3 After
Wilson Deposit Comp	80% -1¼"	P-1	266.64	0.1	9.0	3.8	106.08	106.08
Wilson Deposit Comp	80% -$^3/_8$"	P-3	75.86	0.2	23.9	5.8	99.12	98.90
Wheeler Deposit Comp	80% -1¼"	P-2	273.92	0.1	10.2	7.5	112.39	111.75
Wheeler Deposit Comp	80% -$^3/_8$"	P-4	74.25	0.1	15.8	9.0	100.49	99.89
Wheeler Surface Comp	80% -$^3/_8$"	P-5	73.15	0.0	14.9	7.8	100.13	100.13

* Calculated on a dry ore. Includes initial moisture.
Note: Table 13.7 is adapted from Table 26 in McPartland, 2011

13.3.4 ICP and Whole Rock Analyses

A total of 140 drill core interval samples were received for initial preparation and interval analysis, and for subsequent preparation of metallurgical composites (McPartland, 2011), and the ICP analysis of the composite is presented in **Table 13.9**. The whole rock analysis is presented in **Table 13.10** (McPartland, 2011).

Table 13.9: ICP Metals Analysis Results

Element		Composite	
Analysis	Unit	Wilson Deposit	Wheeler Deposit
Ag	ppm	0.84	0.74
Al	%	7.55	7.45
As	ppm	4.1	4.0
Ba	ppm	930	910
Be	ppm	1.26	1.23
Bi	ppm	1.16	1.14
Ca	%	2.62	2.61
Cd	ppm	0.03	0.03
Ce	ppm	38.0	38.1
Co	ppm	15.3	14.8
Cr	ppm	18	17
Cs	ppm	3.46	3.37
Cu	ppm	582	547
Fe	%	4.16	4.09
Ga	ppm	17.45	17.20
Ge	ppm	0.21	0.22
Hf	ppm	0.8	0.8
Hg	ppm	0.5	<0.1
In	ppm	0.105	0.101
K	%	3.23	3.13
La	ppm	18.1	18.2
Li	ppm	12.5	12.4
Mg	%	1.26	1.25
Mn	ppm	1,370	1,340
Mo	ppm	1.37	1.47
Na	%	1.48	1.47
Nb	ppm	5.6	5.5
Ni	ppm	7.7	7.4
P	ppm	540	530
Pb	ppm	9.7	10.0
Rb	ppm	115.5	118.0
Re	ppm	<0.002	<0.002
S	%	0.73	0.71
Sb	ppm	1.19	1.27
Sc	ppm	19.1	19.2
Se	ppm	2	2
Sn	ppm	4.1	4.3
Sr	ppm	125	126
Ta	ppm	0.43	0.43
Te	ppm	0.18	0.20
Th	ppm	10.4	10.4
Ti	%	0.317	0.309
Tl	ppm	0.54	0.54

Table 13.9: ICP Metals Analysis Results

Element		Composite	
Analysis	Unit	Wilson Deposit	Wheeler Deposit
U	ppm	5.6	5.6
V	ppm	118	114
W	ppm	3.1	3
Y	ppm	20.3	20.8
Zn	ppm	34	33
Zr	ppm	17.4	17.2

McClelland Laboratories, Inc.
MLI Job No. 3376 – May 10, 2011
Note: Table 13.6 is adapted from Table 4 in McPartland (2011).

Table 13.10: Classical Whole Rock and Sulfur Speciation Analysis Results

Analyte	Unit	Composite	
		Wilson Deposit	Wheeler Deposit
Al_2O_3	%	15.7	15.85
BaO	%	0.11	0.12
CaO	%	3.89	3.93
CrO	%	0.01	<0.01
Fe_2O_3	%	6.34	6.64
K_2O	%	4.07	4.21
MgO	%	2.21	2.28
MnO	%	0.18	0.19
Na_2O	%	2.01	2.12
P_2O_5	%	0.16	0.1
SiO_2	%	61.3	61.4
SrO	%	0.01	0.02
TiO_2	%	0.55	0.55
LOI*	%	3.15	2.62
Total	%	99.7	100
$S_{(total)}$	%	0.63	0.07
$S_{(Sulfide)}$	%	0.49	0.05

* Loss on Ignition
McClelland Laboratories, Inc.
MLI Job No. 3376 – May 10, 2011
Note: Table 13.6 is adapted from Table 5 in McPartland (2011).

13.3.5 Fire Assay and Cyanide Soluble Test Results

Nomenclature and Symbols: The grade of oz Au/ton is equal to troy ounces gold per ton (2,000 pounds). The ratio is defined as the quantity of soluble gold in troy oz/ton (2,000 pounds) determined by a cyanide shake test on a ground sample or a cyanide bottle roll test divided by the total gold in troy oz/ton determined by standard fire assay. Thus the ratio is a decimal or percentage of the total gold potential recoverable by direct cyanide leaching.

"Head grade comparisons showed that gold head grade agreement was lower than normally expected, and that gold occurrence was somewhat "spotty". Gold head grade standard deviation ranged from 0.010 to 0.020 ozAu/ton ore. Head grades for comparative column tests

(1¼" vs. $^3/_8$") agreed closely, and in general calculated head grades from the head screen analyses also agreed closely (McPartland, 2011).

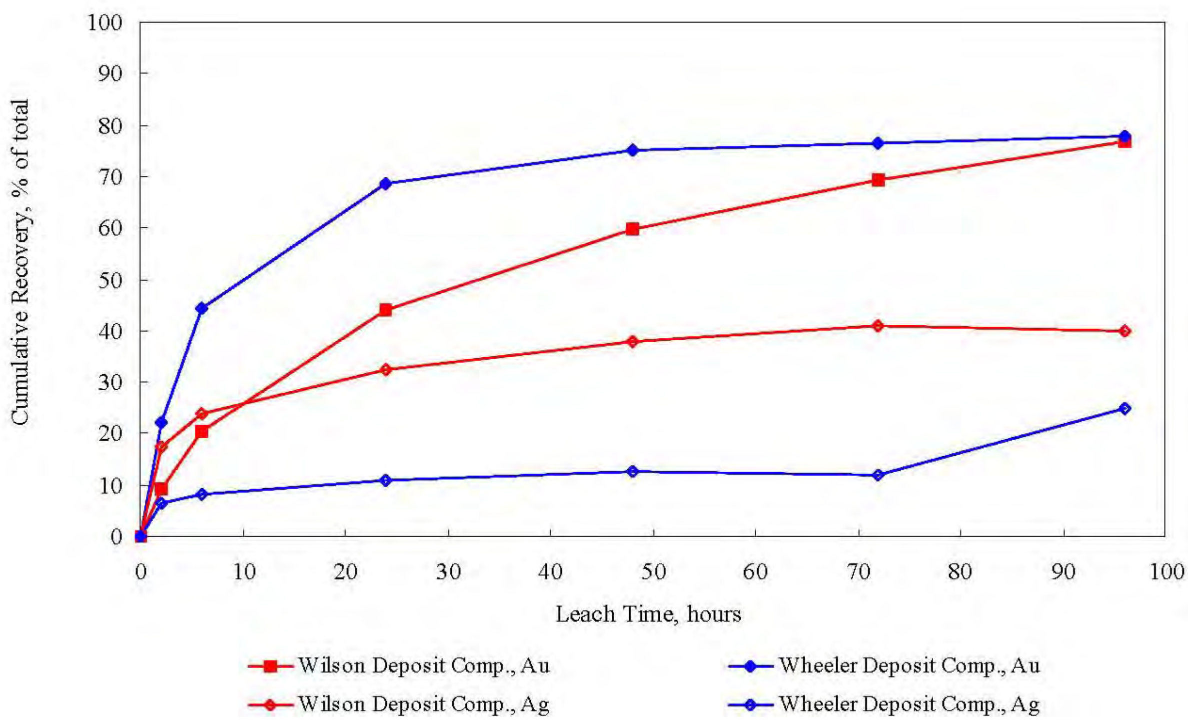
Head grade comparisons showed that none of the composites contained greater than 0.05 ozAg/ton ore (average). Consequently, silver recovery data are not discussed in detail in this report (McPartland, 2011).

13.3.5.1 *Wheeler Deposit – Drill Core*

Preliminary bottle roll test results showed that the Wheeler Deposit composite was readily amenable to direct agitated cyanidation treatment, at an 80% -10M feed size. Respective gold recovery was 78.0%, in 96 hours of leaching. Respective silver recovery was 25.0%. Respective copper extraction was 12.5%. The gold recovery rate for the Wheeler composite was moderate. Silver recovery rates were fairly rapid. Cyanide consumption was low for the Wheeler composite at 0.43 lb NaCN/ton ore. Lime requirements for the Wheeler were 3.7 lbs lime per ton ore (McPartland, 2011).

13.3.5.2 *Wilson Deposit – Drill Core*

Preliminary bottle roll test results showed that the Wilson Deposit composite was readily amenable to direct agitated cyanidation treatment, at an 80% -10M feed size (**Figure 13.2**). Respective gold recovery was 76.9%, in 96 hours of leaching. The calculated head analysis from the bottle roll test was 0.104 ozAu/ton which was significantly higher than the three direct assays of 0.062, 0.064 and 0.084 ozAu/ton and the $^3/_8$" calculated column head of 0.064 and the 1.25" calculated column head of 0.064 ozAu/ton. Respective silver recovery was 40.0%. Respective copper extraction was 25.5%. The gold recovery rate for the Wilson composite was significantly slower than the Wheeler composite. Thus a longer leach cycle would improve gold recoveries in the Wilson composite. Silver recovery rates were fairly rapid. Cyanide consumption was significantly higher for the Wilson composite at 1.02 lb NaCN/ton ore. The lime requirement was low. Lime requirements for the Wilson composite were 4.5 lbs lime per ton ore (McPartland, 2011).



Note: Figure 13.2 is from Figure 1, McPartland, 2011

Figure 13.2: Gold and Silver Leach Rate Profiles, Bottle Roll Tests, 10M Feed Size

13.3.5.3 *Wheeler and Wilson Surface Dumps*

"Lincoln files indicate metallurgical studies of the dumps and tailings were undertaken in 2009. Scoping bottle-roll tests on two dump samples from Wheeler and four dump samples from Wilson yielded gold extractions ranging from 58.8% to 87.0% (**Table 13.11**). Silver extractions from the same samples were 25% and 40%." (Tetra Tech, 2011)

It is assumed that the material sampled from the finger or surface dumps and the finger dumps themselves have been exposed to nature for a sufficient long time to be considered partially oxidized material, or aged, having been subject to natural biooxidation of sulfidic material, if present, and not representative of potential material to be freshly mined from a mine. Thus metallurgical recoveries, reagent consumption and leaching kinetics determined on such material cannot be inferred to other material to be freshly mine. Recent collection of a representative surface sample with agglomeration and column testing information is not available at this time.

Table 13.11: Summary Metallurgical Results, Bottle Roll Tests, 10M Feed Size

MLI Tests Number	Composite	Au Recovery (%)	Ag Recovery (%)	Reagent Requirements lbs/ton ore		Cu Recovery (%)
				NaCN con	Lime Added	
CY-1	Wheeler Finger Dump #1	70.4	40.0	1.21	2.6	15.0
CY-2	Wheeler Finger Dump #2	87.0	25.0	0.60	3.2	14.3
CY-3	Wilson Mine Dump #3	82.4	25.0	0.30	3.0	0.0
CY-4	Wilson Mine Dump #4	58.8	40.0	0.90	2.4	12.5
CY-5	Wilson Mine Main Dump #5	81.8	25.0	0.65	3.0	10.0
CY-6	Wilson Mine Dump #6	81.4	25.0	0.90	2.7	22.2

13.3.6 Screen Fire – Free Gold

"Considering that the Wheeler deposit mine core holes contain visible gold, the check assay program appears to give acceptable reproducible results, and indicates a satisfactory level of accuracy in the assays." (Tetra Tech, 2011) Results from Lincoln bottle roll leach tests conducted during 2010 by McClelland Laboratories in Sparks, NV, showed that, "Gold extraction rates were slow, likely due to the effect of coarse gold. Significant additional gold could likely be extracted if the tests were extended beyond 96 hours." (Tetra Tech, 2011)

As noted in the previous metallurgical reports, visible, free gold can be found on the Pine Grove Property. A review of the available metallurgical reports did not provide a quantitative value or the size of the observed visible free gold. For optimal recovery of this "visible gold" in any mineral or leaching process, the size and quantity of this free gold should be determined from a representative sample. This may be determined from screen fire or pulp and metallic fire assays, a gravity and/or flotation lab or pilot plant testing on a representative sample. Twenty eight samples (5 ft intervals) of the Wheeler and Wilson drill hole program were analyzed using the screen fire assay technique by ALS Minerals of Sparks, Nevada in 2010. The average grade of the samples was 0.21 opt gold and 0.117 opt gold in the minus 100 µm fraction. Thus, an average of 33.6% of the gold values were plus 100 µm for these samples, with nine samples of this set of twenty eight below a grade of 0.04 opt Au. Of the nine samples below 0.04 opt gold, an average of 29.5 percent of the gold was larger than 100 µm. Of importance in such testwork is conducting a metallurgical balance on the testwork. This issue is discussed further in the recoverability section.

13.3.7 Bottle Roll vs. Column Recovery

Core from four metallurgical holes drilled by Lincoln in February 2008 was submitted to McClelland Laboratories in Reno, Nevada, for heap-leach cyanidation testing and environmental characterization. on two holes were from the Wilson deposit and two from Wheeler. Two 8-inch columns (for -1¼" crush) and three 4-inch columns (for -$^3/_8$" crush) were completed in July 2010 (Tetra Tech, 2011).

A total of 140 drill core interval samples were received for initial preparation and interval analysis, and for subsequent preparation of metallurgical composites (3) for heap leach cyanidation testing. Assays to determine gold, silver and copper content, as well as *cyanide shake analyses to determine cyanide soluble gold and copper content, were performed in each drill core interval sample. Results from those analyses were used to construct three metallurgical composites, designated the Wilson Deposit composite, the Wheeler Deposit composite and the Wheeler Surface composite. Column leach tests were conducted on all three composites, at an 80% -$^3/_8$" feed size, to determine heap leach amenability. Comparative column leach tests were conducted on the Wilson Deposit and Wheeler Deposit composites, at an 80% 1¼" feed size, to determine feed size sensitivity of the material from the two deposits. Preliminary direct agitated cyanidation tests were conducted on the same two composites, at an 80% -10M feed size, to obtain preliminary information concerning heap leach amenability." (McPartland, 2011) "Average head grades for the Wilson Deposit, Wheeler Deposit and Wheeler Surface composites were 0.072, 0.056 and 0.094 oz Au/ton ore, respectively. None of the composites contained greater than 0.05 oz Ag/ton ore. Copper content for the three samples ranged from 450 to 650 ppm* (McPartland, 2011).

As shown in **Table 13.12**, the Wheeler deposit achieves good extraction near 75% at a -1¼ inch crush size which improves to over 87% at a -$^3/_8$" crush size. The Wilson deposit exhibits poor recovery at a coarse crush and achieves only greater than 62% extraction when crushed to -$^3/_8$".

Table 13.12: Column Leach Testing

Sample ID	Feed Size, P_{80}	Leach/Rinse Time, days	Extracted, opt	Head Screen Assay, opt	Extracted, %	NaCN Consumed, lb/ton ore	Lime Addition, lb/ton ore
Wilson Comp	-1¼"	141	0.024	0.069	37.5	4.40	4.6
Wilson Comp	-$^3/_8$"	164	0.040	0.062	62.5	5.95	4.6
Wheeler Comp	-1¼"	141	0.035	0.054	74.5	3.71	3.6
Wheeler Comp	-$^3/_8$"	166	0.042	0.043	87.5	6.24	3.6
Surface Comp	-$^3/_8$"	146	0.068	0.109	85.0	6.60	3.6

Tables 13.13 and **13.14** are the results from Lincoln bottle roll leach tests conducted during 2010 by McClelland Laboratories in Sparks, NV. Gold extraction rates were slow, likely due to the effect of coarse gold. Significant additional gold could likely be extracted if the tests were extended beyond 96 hours (Tetra Tech, 2011).

Table 13.13: Bottle Roll Tests, Wheeler Deposit, 2010

Sample No.	Hole	Depth, ft	Nominal Crush Size, in	Leach Time, hrs	Au Extraction, %	Calculated Head Grade, oz Au/ton	Cyanide Consumed, lb/ton ore	Lime Addition, lb/ton ore
CY-1	WR-105	55-65	-10 M	96	93.6	0.047	0.25	3.7
CY-2	WR-106	30-40	-10 M	96	78.0	0.159	0.26	3.3
CY-3	WR-106	80-90	-10 M	96	84.6	0.013	0.49	3.1
CY-4	WR-106	140-150	-10 M	96	57.6	0.059	1.33	2.2
CY-5	WR-108	115-125	-10 M	96	61.7	0.047	1.04	1.7
CY-6	WR-110	45-55	-10 M	96	59.6	0.047	0.73	3.7
CY-7	WR-110	55-65	-10 M	96	65.5	0.055	0.17	4.7
CY-8	WR-111	175-185	-10 M	96	60.9	0.046	0.45	2.6
CY-9	WR-113	115-125	-10 M	96	56.1	0.139	0.60	3.0
CY-10	WR-116	15-25	-10 M	96	86.2	0.029	<0.14	6.4
CY-11	WR-116	50-60	-10 M	96	85.0	0.020	0.46	4.4
CY-12	WR-118	25-35	-10 M	96	55.0	0.020	0.15	3.2
CY-13	WR-118	115-125	-10 M	96	74.4	0.043	0.88	3.4
CY-14	WR-118	150-160	-10 M	96	58.8	0.017	0.63	3.4
CY-15	WR-118	210-220	-10 M	96	79.1	0.043	1.19	4.4
CY-16	WR-118	220-230	-10 M	96	78.6	0.014	0.59	2.7
CY-17	WR-119	30-40	-10 M	96	78.9	0.057	0.19	5.3
CY-18	WR-119	55-65	-10 M	96	79.3	0.029	1.78	4.3

Table 13.14: Bottle Roll Tests, Wilson Deposit, 2010

Sample No.	Hole	Depth, ft	Nominal Crush Size, in	Leach Time, hrs	Au Extraction, %	Calculated Head Grade, oz Au/ton	Cyanide Consumed, lb/ton ore	Lime Addition, lb/ton ore
CY-19	WL-63	180-190	-10 M	96	77.3	0.088	1.48	2.2
CY-20	WL-66	125-135	-10 M	96	59.1	0.022	0.22	2.5
CY-21	WL-66	145-155	-10 M	96	44.6	0.139	2.30	1.9
CY-22	WL-85	155-165	-10 M	96	60.2	0.103	2.08	1.8
CY-23	WL-87	85-95	-10 M	96	55.6	0.018	0.60	3.6
CY-24	WL-90	90-100	-10 M	96	60.0	0.020	0.32	2.5
CY-25	WL-91	140-150	-10 M	96	76.9	0.039	0.14	2.7
CY-26	WL-92	90-100	-10 M	96	79.6	0.049	0.45	3.8
CY-27	WL-92	155-165	-10 M	96	57.5	0.040	0.14	2.2
CY-28	WL-93	75-85	-10 M	96	72.2	0.018	0.15	3.3
CY-29	WL-93	90-100	-10 M	96	63.6	0.033	0.45	2.9
CY-30	WL-94	140-150	-10 M	96	66.7	0.006	0.18	1.8
CY-31	WL-98	135-145	-10 M	96	78.4	0.037	0.18	3.7
CY-32	WL-99	95-105	-10 M	96	78.0	0.050	<0.14	7.9
CY-33	WL-100	85-95	-10 M	96	66.7	0.015	<0.14	2.5
CY-34	WL-100	285-295	-10 M	96	50.0	0.022	0.30	3.1
CY-35	WL-101	120-130	-10 M	96	67.9	0.159	3.70	4.6
CY-36	WL-101	160-170	-10 M	96	71.6	0.081	0.15	2.7
CY-37	WL-102	55-65	-10 M	96	72.1	0.043	<0.14	4.1
CY-38	WL-103	140-150	-10 M	96	64.7	0.017	0.14	2.2
CY-39	WL-103	270-280	-10 M	96	64.8	0.054	0.14	2.2

Head screen analysis results showed that the Wilson composite 1¼" and $^{3}/_{8}$" feeds contained 0.069 and 0.062 ozAu/ton ore, respectively, and that the Wheeler 1¼" and $^{3}/_{8}$" feeds contained 0.054 and 0.043 ozAu/ton ore. The Wheeler Surface composite ($^{3}/_{8}$" feed) contained 0.109 ozAu/ton ore. Contained gold values were not evenly distributed throughout the various size fractions. Size fraction assays tended to be "spotty", possibly indicating the presence of free-milling, particulate gold values. Further testing would be required to confirm this observation (McPartland, 2011).

Tail screen analysis results show that the Wilson composite 1¼" and $^{3}/_{8}$" column leached residues contained 0.040 and 0.024 ozAu/ton ore, respectively. Tail screen results and recovery by size fraction data from the 1¼" feed size test indicate that crushing finer than ½" in size would substantially improve gold recovery by cyanidation. The actual gold recovery obtained at the $^{3}/_{8}$" feed size (62.5%) confirmed that observation. Tail screen results and recovery by size fraction data from both feed sizes indicate that fine grinding (-100M) would be required to maximize gold recovery by cyanidation (McPartland, 2011).

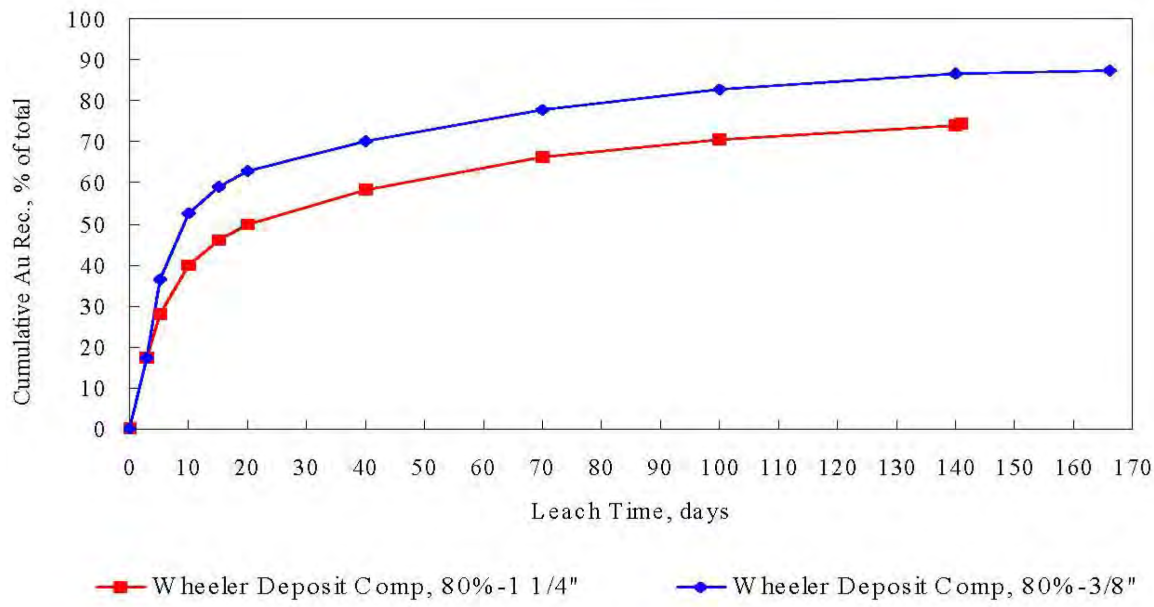
Tail screen results show that the Wheeler 1¼" and $^{3}/_{8}$" column leached residues contained 0.012 and 0.006 ozAu/ton ore, respectively. Tail screen results and recovery by size fraction data from the 1¼" feed size test indicate that crushing finer than 1" in size would significantly improve gold recovery by cyanidation. The actual gold recovery obtained at the $^{3}/_{8}$" feed size (87.5%) supports that observation. Tail screen results and recovery by size fraction data from both feed sizes indicate that grinding (-65M) would be required to maximize gold recovery by cyanidation (McPartland, 2011).

The Wheeler Surface composite ($^{3}/_{8}$" feed) column leached residue contained 0.012 ozAu/ton ore. Residual gold values were fairly evenly distributed throughout the various size fractions, with a minor enrichment of values noted in the intermediate (-$^{3}/_{8}$" +10M) size fractions. Tail screen results and recovery by size fraction data indicate that crushing finer than ½" in size would improve gold recovery slightly, and that grinding (-65M) would be required to maximize gold recovery by cyanidation (McPartland, 2011).

Solution versus tail and loaded carbon versus tail metallurgical balances generally agreed very closely (<0.001 ozAu/ton ore deviation). The loaded carbon versus tail balance for the Wilson composite at the 1¼" feed size did not agree as closely (0.006 ozAu/ton ore deviation), but did agree within normally excepted precision limits (>90%). Head versus tail metallurgical balances generally did not agree as closely, because of the head grade variability described earlier in this report. Solution versus tail balance is considered the most reliable because of the number of check analyses performed on column test pregnant solutions. That balance was used for all percent recovery calculations, except as otherwise noted (McPartland, 2011).

Results from load/permeability testing conducted on a composite of the three 9.5mm feed size column leached residues (Ref. Sect. 6, App.) showed that the material displayed adequate permeability characteristics under simulated heap stack heights of as high as 100m. The measured hydraulic conductivity at a simulated 100m stack height was 5.33 x 10^{-2} cm/sec (McPartland, 2011).

13.3.7.1 *Wheeler Drill Composite Cyanide Column Leach Testing*



Note: Figure 13.3 is from Figure 3, McPartland, 2011

Figure 13.3: Gold Leach Rate Profile, Column Leach Tests for Wheeler Composites

From McPartland (2011):

Overall metallurgical results show that the Wheeler Deposit composite was readily amenable to simulated heap leaching treatment at the 80% -1¼" and $^3/_8$" feed sizes (**Figure 13.3**). *Gold recoveries obtained from the Wheeler Deposit composite at the 1¼" and $^3/_8$" feed sizes were 74.5% and 87.5%, respectively, in 141 to 166 days of leaching and rinsing. Respective screen recoveries were <50.0% and 50.0%.*

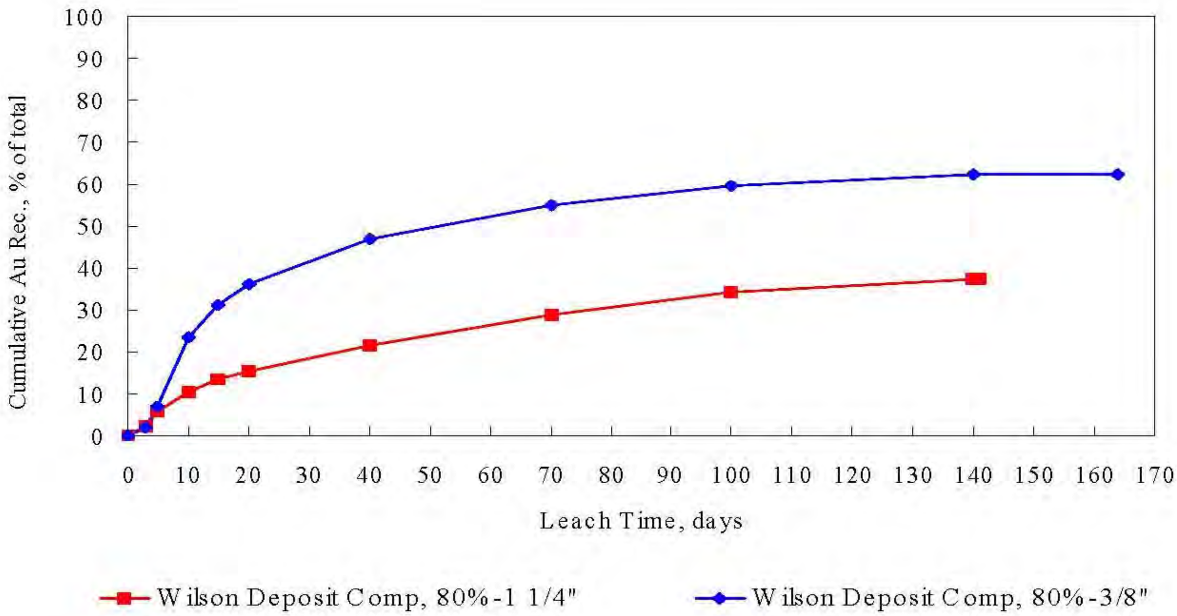
Gold recovery rates were moderate and increased with decreasing feed size. A longer leaching cycle would not significantly improve precious metal recoveries.

Cyanide consumptions were high for both feed sizes. Cyanide consumptions for the 1¼" and $^3/_8$" feeds were 3.71 and 6.24 lb NaCN/ton ore respectively.

The lime added to the ore charges before leaching, 3.6 lb lime per ton ore, was sufficient for maintaining protective alkalinity throughout the leaching cycle.

Copper extractions for both tests were 0.1 lb/ton Cu ore (50 ppm), which was equivalent to 11%-13% of the total contained copper. Copper concentrations in the column test pregnant solutions increased to as much as approximately 150 ppm (mg/L) during leaching, which was substantially lower than observed with the Wilson Deposit composite tests.

13.3.7.2 *Wilson Drill Composite Cyanide Column Leach Testing*



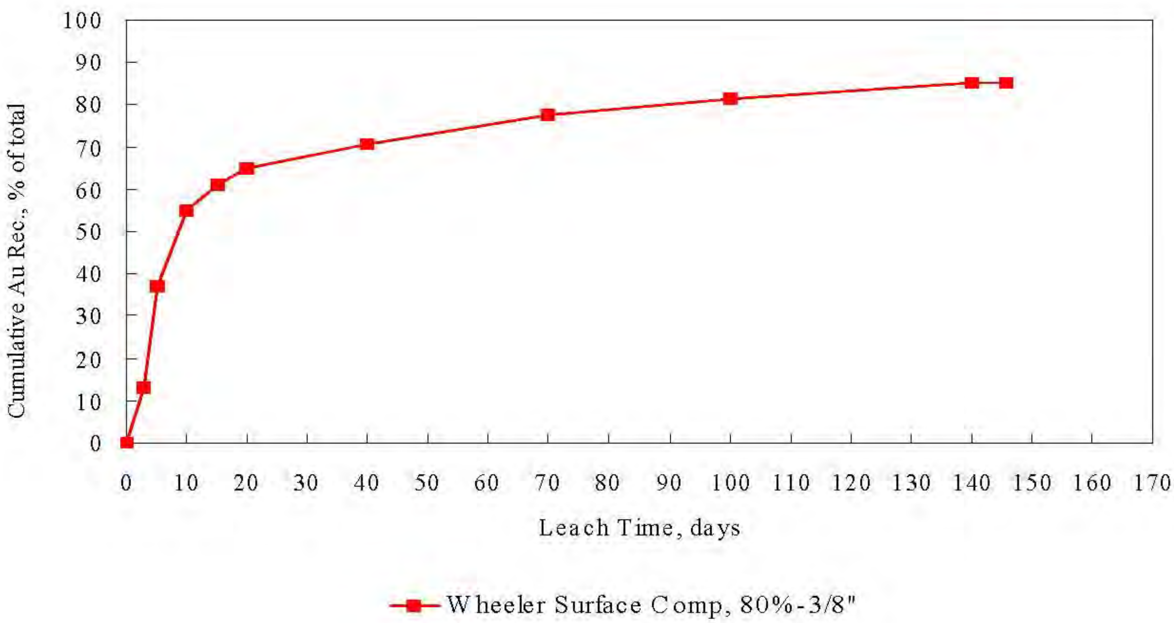
Note: Figure 13.4 is from Figure 2, McPartland, 2011

Figure 13.4: Gold Leach Rate Profile, Column Leach Tests for Wilson Composites

From McPartland (2011):

> *Overall metallurgical results show that the Wilson Deposit composite was amenable to simulated heap leaching treatment at the 80% -$^{3}/_{8}$" feed size (***Figure 13.4***). The Wilson Deposit composite was not amenable to simulated heap leaching treatment at the 80% -1¼" feed size. Gold recoveries obtained from the Wheeler Deposit composite at the 1¼" and $^{3}/_{8}$" feed sizes were 37.5% and 62.5%, respectively, in 141 to 164 days of leaching and rinsing. Respective silver recoveries were 20.0% and 25.0%.*
>
> *Gold recovery rates were moderate and increased with decreasing feed size. A longer leaching cycle would not significantly improve precious metal recoveries.*
>
> *Cyanide consumptions were high for both feed sizes. Cyanide consumptions for the 1¼" and $^{3}/_{8}$" feeds were 4.40 and 5.95 lb NaCN/ton ore respectively.*
>
> *The lime added to the ore charges before leaching, 4.6 lb lime per ton ore, was sufficient for maintaining protective alkalinity throughout the leaching cycle. Copper extractions obtained at both feed sizes were 0.2 lbCu/ton ore (100 ppm), which was equivalent to 15% of the total contained copper. Copper concentrations in the column test pregnant solutions increased to as much as approximately 400 ppm (mg/L) during leaching. These concentrations are considered to be high enough to complicate down-stream solution recovery and refining processes solution recovery in a commercial heap leach circuit.*

13.3.7.3 *Wheeler Surface Composite*



Note: Figure 13.5 is from Figure 4, McPartland, 2011

Figure 13.5: Gold Leach Rate Profile, Column Leach Tests for Wheeler Surface Composites

From McPartland (2011):

Overall metallurgical results show that the Wheeler Deposit Surface composite was amenable to simulated heap leaching treatment at the 80% -$^{3}/_{8}$" feed size (**Figure 13.5**)*. Gold and silver recoveries obtained from the Wheeler Deposit Surface composite at the $^{3}/_{8}$" feed size were 85.0%, and 50%, respectively, in 146 days of leaching and rinsing.*

Gold recovery rate was moderate. A longer leaching cycle would not significantly improve precious metal recoveries.

Cyanide consumption was high. Cyanide consumption for the $^{3}/_{8}$" feed was 6.60 lb NaCN/ton ore.

The lime added to the ore charge before leaching, 3.6 lb lime per ton ore, was sufficient for maintaining protective alkalinity throughout the leaching cycle.

The copper extracted during leaching was 0.2 lbCu/ton ore, which was equivalent to 17% of the total contained copper. Column test pregnant solution copper concentrations increased to as high as approximately 300 ppm (mg/L) during leaching.

13.3.8 Bottle Roll vs. Column Recovery

Table 13.15 displays the primary leach reagents utilized in the cyanide column tests on the Pine Grove Property found in McPartland, 2011 and Clem, 1983.

Table 13.15: Cyanide Leach Utilization Data (McPartland, 2011)

				Wheeler Composite		Wheeler Surface		Wilson Composite		Wilson Surface	
P_{80} Size:				lbs/ton	lbs/ton	lbs/ton	lbs/ton	lbs/ton	lbs/ton	lbs/ton	lbs/ton
Test	Reported	mm	inch	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO
BRT	10	2	0.079	0.43	3.70	0.91	2.90	1.02	4.50	0.69	2.78
Column	$^{3}/_{8}$"	9.5	0.375	6.24	3.60	6.60	3.60	5.95	4.60		
Column	½"*	12.5	0.500							2.20	10.00
Column	1¼"	31.5	1.25	3.71	3.60			4.40	4.60		

* - This sample is a nominal ½ inch not a P_{80} of ½" and reported in Clem, 1983.

Table 13.15 shows the difference in the cyanide reagent consumption found in the bottle roll tests in comparison with the cyanide utilized in the column tests. Thus it is important to use reagent utilizations from bottle roll tests and not rely on reagent consumptions from cyanide column tests. But, "cyanide consumption in a heap leach is generally around 25-30% of cyanide consumption in a column leach test" (KCA, 2011). Due to the large concentration of dissolved copper in the pregnant solution from the column leach tests and respective high cyanide utilization, a value of 41.5% of the column test cyanide utilization was adopted for this report.

13.3.9 Gold Recovery vs. Size Distribution

Dr. Thom Seal, P.E. has been working with heap leach ores for over 15 years, was a metallurgical manager for Newmont Mining Corp in the metallurgical laboratory, milling and heap leach operations, teaches undergraduate, graduate engineering and design courses at University of Nevada-Reno on Heap Leaching and is a qualified person. Dr. Seal has developed a heap leach model that uses cyanide soluble values as a function of size distribution of the sample to predict heap leach gold recoveries. Size distribution tables came from a mineral process textbook (Kelly, 1981).

The model in **Figure 13.6** (McPartland, 2001 and Clem, 1983) shows a good correlation for the Wilson drill composite with an r^2 of 0.98, but poor correlation of the Wheeler composite with an r^2 of 0.27. This effect could be due to the gold nugget effect, sample preparation, lab test errors and aging of various samples, etc. From this cyanide leach bottle roll and column testwork it is recommended to crush the Wheeler material to a maximum size with a P_{80} of 1.25" to achieve a recovery of 74.5% of the gold and the Wilson material to a maximum size of a P_{80} of 0.375" to achieve a recovery of 62.5%. The Wilson composite's gold extraction is definitely limiting with respect to crush size and cyanide concentration. This small heap leach crush size of a P_{80} of $^{3}/_{8}$" requires high capital and operational costs and demands adequate agglomeration for acceptable heap permeability and percolation. "The Wilson Deposit composite was not amenable to simulated heap leaching treatment at the 80% -1¼" feed size" (McPartland, 2011) or larger crushed size. It is recommended to conveyor stack the agglomerated, fine, crushed mineralized material onto the heap and not truck dump the material to reduce compaction, heap blinding and thus enhance percolation, leaching and recovery.

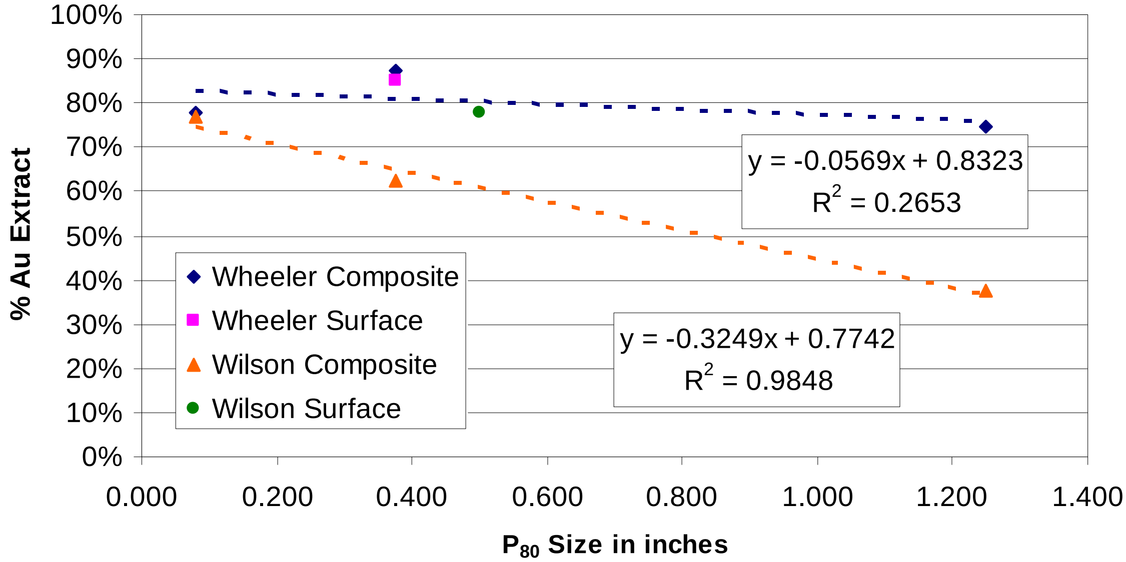


Figure 13.6: Cyanide Leach Model, Pine Grove Composite Samples

13.4 Gold Recovery vs. [Cu]

The presence of cyanide soluble copper in an ore has a detrimental effect on the leachability of precious metals in that ore. From the Wilson Column Composite “Copper extractions obtained at both feed sizes were 0.2 lb Cu/ton ore (100 ppm), which was equivalent to 15% of the total contained copper. From the Wheeler Composite “Copper extractions for both tests were 0.1 lbCu/ton ore (50 ppm), which was equivalent to 11%-13% of the total contained copper.” (McPartland, 2011)

Cyanide soluble copper will consume the available free CN^- in the solution that wets the ore, thus limiting the available CN^- to leach gold. Copper forms several soluble species with cyanide, $Cu(CN)_x$ in which x can be 2, 3, or 4 (Marsden, 1993). In addition, the $Cu(CN)_x$ competed with gold loading on activated carbon in a CIC system. Copper bottle roll extractions on the Wilson and Wheeler surface dump samples show 0 to 22.2% (Tetra Tech, 2011). Thus the reagent and operating costs for this ore will be high compared to a normal heap leach operation.

13.5 Testwork Recommendations

Extensive recommendations are offered in Section 26.

13.6 Mineral Processing

Lincoln has not provided any metallurgical testwork reports on traceable representative samples of the Pine Grove property which involve grinding and concentration of the precious metal via mineral processing. While several of the fire assay gold results on Pine Grove property show sufficient grade (>0.05 Auoz/ton) for economic mineral processing and precious metal recovery via grinding and concentration (gravity and/or flotation), the lack of metallurgical test work on the higher grade material precludes any metallurgical recovery inferences, evaluations and further flow sheet speculation.

13.7 Mineralized Material and ROM System

The recovery of precious metals from this Pine Grove material is greatly dependent on the size distribution of the leached material, crushed size, with higher recovery associated with smaller particles with more surface area and greater microporosity. The column test data on drill composites show a required crushed P_{80} size of 1¼ inches to achieve a gold extraction of 74.5% for the Wheeler drill composite and a P_{80} of $^{3}/_{8}$ inch for the gold extraction of 62.5% for the Wilson drill composite. Thus intensive crushing and agglomeration will be required to achieve these gold extractions and generally precludes the opportunity to mine and stack run-of-mine (ROM) on the heap. There may be an opportunity to stack very low grade ROM Wheeler material (<0.01 ozAu/ton) at these high gold prices (>$1,500), but this must be thoroughly examined with a pilot heap or large column testing and evaluated prior to any economic evaluation.

13.8 Leaching and Recovery Systems

The Wheeler and Wilson material crushed and agglomerated show good percolation and permeability for heap leaching. “Results from load/permeability testing conducted on a composite of the three 9.5mm feed size column leached residues (Ref. Sect. 6, App.) showed that the material displayed adequate permeability characteristics under simulated heap stack heights of as high as 100m. The measured hydraulic conductivity at a simulated 100m stack height was 5.33 x 10^{-2} cm/sec.” (McPartland, 2011)

Generally precious metal recovery systems that recover metals from pregnant solutions use the Merrill-Crowe zinc precipitation process or CIC process. The Merrill-Crowe zinc precipitation is primarily used for pregnant solution with silver concentrations greater than 10-20:1 silver:gold ratio (Marsden, 1993). CIC systems are more efficient for low grade gold solutions, but require an additional stripping, electrowinning and regeneration process that require carbon shipment offsite or onsite permitting and treatment. The property does contain 0.5 ppm Hg determined by ICP on the Wilson composite, so a quantity of Hg could be leached and reported to the recovery system, which must be mitigated in the overall design of the stripping and regeneration system, or addressed in carbon shipment procedures. In addition, the Wheeler and Wilson deposit composites tested and reported in McPartland (2011) showed cobalt of 15.3 and 14.8 ppm respectively. Cobalt does leach and form cyanide complexes that could impact the downstream collection and refining of the precious metals, and should be tracked in future metallurgical testing. The Qualified Person responsible for preparation of this section of the report have reviewed the information and conclusions provided and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this report. No independent metallurgical or mineral processing testing was performed by Telesto.

14.0 MINERAL RESOURCE ESTIMATES

Modeling and estimation of gold resources demonstrate that there are measured, indicated and inferred resources at the Pine Grove Project. All modeling of the project area was performed using MicroMODEL mining software. The resource estimated from the modeling is reported in Imperial units (short tons and Troy ounces per ton, opt) and metric units (tonnes and grams per ton, g/t) of gold, as noted.

14.1 Sources of Information

The raw data for the review was provided by Lincoln. This data consisted of RC and core drilling data which was in a digital database. This data was supplied by Lincoln but had been put into a digital database by Tetra Tech. Data was checked by Telesto personnel. See Section 12 for a detailed description of the data verification efforts performed by Telesto.

Lincoln also provided the topography data. The topography data originated from Dudley Thomas Mapping of Surrey, BC. Larry Grube, R.L.S., of Summit Engineering of Reno, Nevada took the Teck local grid, surveyed the existing holes and converted the locations to Nevada State Plane West Zone NAD83 feet. As a check on the conversion to state plane coordinates, Dudley Thomas Mapping performed a statistical analysis on the coordinates and found them to be statistically acceptable. Telesto did not verify the coordinate locations of any of the drill holes in the field. However, a review of the drill hole locations and collar elevations relative to mapped topography was performed by Telesto and the drill hole locations appear to be correctly assigned in the drill hole database.

14.2 Deposit Geology Pertinent to Resource Modeling

Telesto noted seventeen (17) fundamental rock types which have been logged at Pine Grove (codes 7, 8 and 13 were not used). An additional code (number 18) in the database denotes voids which were encountered when the drill passed through historic workings. **Table 14.1** shows the rock types and their associated numeric codes.

Table 14.1: Rock Types and Codes

Rock Code	Rock Type
1	Colluvium and slope cover, talus, overburden, etc.
2	Mine dump material
3	Alluvium, gravel, sand and silt stream deposits
4	Rhyolite dikes and flows
5	Basalt dikes or flows
6	Morgan Ranch Fm; conglomerate, sandstone, clay, limestone
7	Not used
8	Not used
9	Andesite dikes; may also be fine-grained phase of granodiorite
10	Dacite dikes; may also be fine-grained phase of granodiorite
11	Rhyolite porphyry dikes; pinkish gray groundmass w/ feldspar phenocrysts; sometimes called felsite
12	Aplite dikes or zones; also sometimes called felsite
13	Not used
14	Granodiorite; fine, medium, & coarse grained; often biotized
15	Metavolcanics
16	Granodiorite with quartz veins
17	Rhyolite porphyry with quartz veins
18	Void; mine workings, stopes, back-filled workings, zones of wood debris
19	Backfill or caved rock material in mine workings and stopes
20	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar
21	Feldspar porphyry with quartz veins

State of oxidation was noted for most intervals in the drillhole logs so the rock type field contains codes for oxidation state. The rock type code contains either two, three or four digits, depending on whether the rock type code is 1 through 9 or 10 through 21. For three digit rock type codes, the second digit or third digit represents the oxidation state as defined in **Table 14.2**. In four digit rock type codes, the third digit represents the alteration. If the oxidation state code is absent, the interval is oxidized.

Table 14.2: Pine Grove Oxidation State Codes

Code	Explanation	Oxidation State
x2	One digit rock type, no structure code	Mixed oxide and sulfide, generally oxidized on fractures and open spaces with sulfides in groundmass
x2x	One digit rock type, structure code is the last digit	
xx2	Two digit rock type, no structure code	
xx2x	Two digit rock type, structure code is the last digit	
x3	One digit rock type, no structure code	Pyrite, lesser Cu sulfides as fracture coatings, small disseminated blebs or crystals, no significant oxides
x3x	One digit rock type, structure code is the last digit	
xx3	Two digit rock type, no structure code	
xx3x	Two digit rock type, structure code is the last digit	

Another code in the rock type code records structural information. In all cases when present, the structure code is the last digit of the rock code. See **Table 14.3** for an explanation of structure codes. If a code for structure (7, 8 or 9) is absent, the interval has no significant structure noted.

Table 14.3: Pine Grove Structure Codes

Code	Explanation	Structure
x7	One digit rock type, no oxidation state code, plus structure code	Broken and fractured rock, rock chips recovered in RC cuttings and large, angular fragments
xx7	One digit rock type, oxidation state code plus structure code, or two digit rock type, no oxidation state code, plus structure code	
xxx7	Two digit rock type, oxidation state code and structure code	
x8	One digit rock type, no oxidation state code, plus structure code	Faults, RC cuttings are large, angular and contain some clay as fault gouge
xx8	One digit rock type, oxidation state code plus structure code, or two digit rock type, no oxidation state code, plus structure code	
xxx8	Two digit rock type, oxidation state code and structure code	
x9	One digit rock type, no oxidation state code, plus structure code	Fault gouge, few rock cuttings, mainly all clay gouge
xx9	One digit rock type, oxidation state code plus structure code, or two digit rock type, no oxidation state code, plus structure code	
xxx9	Two digit rock type, oxidation state code and structure code	

Assays were recorded in the drill logs as gold (opt), copper (ppm) and copper (%). Therefore, gold was estimated in terms of opt in the resource estimate. An estimate for copper (%) was created in the resource model using the nearest neighbor methodology.

14.3 Geostatistics

14.3.1 Rock Type Statistics Based on Drill Hole Information

Statistics for all rock types are shown in **Table 14.4**. Related codes are grouped based in the 17 rock types as listed in **Table 14.1**.

Table 14.4: Gold Statistics for All Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)	Variance	Std. Dev.
1	Colluvium and slope cover, talus, overburden, etc.	390	0.0000	0.2840	0.033	0.005	0.0213
2	Mine dump material	40	0.0000	0.0990	0.0176	0.005	0.0225
3	Alluvium, gravel, sand and silt stream deposits	6	0.0010	0.0030	0.0013	0.0000	0.0008
4	Rhyolite dikes and flows	35	0.0010	0.0010	0.0010	0.0000	0.0000
43	Rhyolite dikes and flows with pyrite	33	0.0010	0.0020	0.0016	0.0000	0.0005
439	Rhyolite dikes and flows with pyrite; mainly clay fault gouge	21	0.0010	0.0020	0.0013	0.0000	0.0005
49	Rhyolite dikes and flows; mainly clay fault gouge	1	0.0010	0.0010	0.0010	0.0000	0.0000
5	Basalt dikes or flows	3	0.0010	0.0010	0.0010	0.0000	0.0000
53	Basalt dikes or flows with pyrite	3	0.0010	0.0080	0.0033	0.0000	0.0040

Table 14.4: Gold Statistics for All Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)	Variance	Std. Dev.
6	Morgan Ranch Fm; conglomerate, sandstone, clay, limestone	787	0.0001	0.0170	0.0010	0.0000	0.0010
62	Morgan Ranch Fm with mixed oxide and sulfide	5	0.0010	0.0010	0.0010	0.0000	0.0000
628	Morgan Ranch Fm with mixed oxide and sulfide; fault with some clay in gouge	1	0.0020	0.0020	0.0020	0.0000	0.0000
63	Morgan Ranch Fm with pyrite	13	0.0010	0.0010	0.0010	0.0000	0.0000
638	Morgan Ranch Fm with pyrite; fault with some clay in gouge	3	0.0010	0.0030	0.0017	0.0000	0.0012
68	Morgan Ranch Fm; fault with some clay in gouge	37	0.0010	0.0620	0.0037	0.0001	0.0116
69	Morgan Ranch Fm; mainly clay fault gouge	4	0.0010	0.0160	0.0070	0.0000	0.0065
9	Andesite dikes; may also be fine-grained phase of granodiorite	16	0.0010	0.0060	0.0023	0.0000	0.0021
92	Andesite dikes with mixed oxide and sulfide	3	0.0010	0.0010	0.0010	0.0000	0.0000
929	Andesite dikes with mixed oxide and sulfide; mainly clay fault gouge	3	0.0010	0.0090	0.0050	0.0000	0.0040
93	Andesite dikes with pyrite	32	0.0001	0.0070	0.0016	0.0000	0.0016
937	Andesite dikes with pyrite; broken and fractured	1	0.0003	0.0003	0.0003	0.0000	0.0000
938	Andesite dikes with pyrite; fault with some clay in gouge	1	0.0002	0.0002	0.0002	0.0000	0.0000
939	Andesite dikes with pyrite; mainly clay fault gouge	2	0.0003	0.0010	0.0007	0.0000	0.0005
99	Undefined	219	0.0000	0.2700	0.0203	0.0015	0.0389
10	Dacite dikes; may also be fine-grained phase of granodiorite	169	0.0000	0.0940	0.0016	0.0001	0.0073
102	Dacite dikes with mixed oxide and sulfide	5	0.0010	0.0020	0.0012	0.0000	0.0004
103	Dacite dikes with pyrite	74	0.0000	0.0750	0.0041	0.0002	0.0126
1038	Dacite dikes with pyrite; fault with some clay in gouge	2	0.0020	0.0070	0.0045	0.0000	0.0035
1039	Dacite dikes with pyrite; mainly clay fault gouge	4	0.0010	0.0020	0.0013	0.0000	0.0005
107	Dacite dikes; broken and fractured	6	0.0000	0.0010	0.0003	0.0000	0.0005
108	Dacite dikes; fault with some clay in gouge	10	0.0000	0.0010	0.0003	0.0000	0.0005
109	Dacite dikes; mainly clay fault gouge	6	0.0000	0.0120	0.0028	0.0000	0.0049
11	Rhyolite porphyry dikes; pinkish gray groundmass w/ feldspar phenocrysts; sometimes called felsite	797	0.0000	0.2400	0.0015	0.0001	0.0096

Table 14.4: Gold Statistics for All Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)	Variance	Std. Dev.
112	Rhyolite porphyry dikes with mixed oxide and sulfide	12	0.0010	0.0070	0.0017	0.0000	0.0018
113	Rhyolite porphyry dikes with pyrite	159	0.0000	0.3850	0.0086	0.0014	0.0376
1138	Rhyolite porphyry dikes with pyrite; fault with some clay in gouge	5	0.0000	0.0000	0.0000	0.0000	0.0000
1139	Rhyolite porphyry dikes with pyrite; mainly clay fault gouge	3	0.0010	0.0010	0.0010	0.0000	0.0000
117	Rhyolite porphyry dikes; broken and fractured	15	0.0000	0.0440	0.0040	0.0001	0.0113
118	Rhyolite porphyry dikes; fault with some clay in gouge	8	0.0000	0.0070	0.0016	0.0000	0.0026
119	Rhyolite porphyry dikes; mainly clay fault gouge	27	0.0000	0.3780	0.0214	0.0054	0.0734
12	Aplite dikes or zones; also sometimes called felsite	11	0.0010	0.0480	0.0163	0.0003	0.0172
122	Aplite dikes or zones with mixed oxide and sulfide	1	0.0430	0.0430	0.0430	0.0000	0.0000
123	Aplite dikes or zones with pyrite	56	0.0010	0.1630	0.0126	0.0006	0.0254
1239	Aplite dikes or zones with pyrite; mainly clay fault gouge	1	0.0010	0.0010	0.0010	0.0000	0.0000
129	Aplite dikes or zones; mainly clay fault gouge	1	0.0340	0.0340	0.0340	0.0000	0.0000
14	Granodiorite; fine, medium, & coarse grained; often biotized	2,916	0.0000	**1.9060**	0.0113	0.0034	0.0585
142	Granodiorite with mixed oxide and sulfide	444	0.0010	**1.0930**	0.0155	0.0043	0.0654
1427	Granodiorite with mixed oxide and sulfide; broken and fractured	1	0.0225	0.0225	0.0225	0.0000	0.0000
1428	Granodiorite with mixed oxide and sulfide; fault with some clay in gouge	2	0.0378	0.0482	0.0430	0.0001	0.0074
1429	Granodiorite with mixed oxide and sulfide; mainly clay fault gouge	25	0.0010	0.7820	0.0386	0.0242	0.1554
143	Granodiorite with pyrite	5,770	0.0000	0.7720	0.0051	0.0006	0.0247
1437	Granodiorite with pyrite; broken and fractured	115	0.0000	**2.2500**	0.0251	0.0441	0.2100
1438	Granodiorite with pyrite; fault with some clay in gouge	138	0.0000	0.2200	0.0051	0.0004	0.0206
1439	Granodiorite with pyrite; mainly clay fault gouge	568	0.0000	0.7200	0.0062	0.0015	0.0382
147	Granodiorite; broken and fractured	72	0.0000	0.2450	0.0084	0.0009	0.0304
148	Granodiorite; fault with some clay in gouge	189	0.0000	**1.1700**	0.0199	0.0084	0.0917
149	Granodiorite; mainly clay fault gouge	241	0.0000	0.7150	0.0133	0.0029	0.0538
153	Metavolcanics with pyrite	1	0.0010	0.0010	0.0010	0.0000	0.0000
16	Granodiorite with quartz veins	178	0.0000	0.3200	0.0283	0.0026	0.0509

Table 14.4: Gold Statistics for All Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)	Variance	Std. Dev.
162	Granodiorite with quartz veins with mixed oxide and sulfide	37	0.0010	0.2280	0.0207	0.0016	0.0401
1629	Granodiorite with quartz veins with mixed oxide and sulfide; mainly clay fault gouge	2	0.0070	0.0342	0.0206	0.0004	0.0192
163	Granodiorite with quartz veins with pyrite	259	0.0000	**2.2700**	0.0370	0.0391	0.1978
1637	Granodiorite with quartz veins with pyrite; broken and fractured	11	0.0001	0.0498	0.0098	0.0002	0.0145
1638	Granodiorite with quartz veins with pyrite; fault with some clay in gouge	18	0.0000	0.5190	0.0333	0.0149	0.1220
1639	Granodiorite with quartz veins with pyrite; mainly clay fault gouge	27	0.0000	0.5490	0.0257	0.0110	0.1050
167	Granodiorite with quartz veins; broken and fractured	13	0.0000	0.1390	0.0347	0.0025	0.0502
168	Granodiorite with quartz veins; fault with some clay in gouge	16	0.0000	0.0147	0.0027	0.0000	0.0037
169	Granodiorite with quartz veins; mainly clay fault gouge	27	0.0000	0.0400	0.0068	0.0001	0.0097
173	Rhyolite porphyry with quartz veins with pyrite	7	0.0010	0.2750	0.0627	0.0111	0.1054
18	Void; mine workings, stopes, back-filled workings, zones of wood debris	0	–	–	–	–	–
19	Backfill or caved rock material in mine workings and stopes	35	0.0010	0.6830	0.0542	0.0149	0.1222
192	Backfill or caved rock material in mine workings and stopes with mixed oxide and sulfide	11	0.0010	0.0934	0.0355	0.0007	0.0272
193	Backfill or caved rock material in mine workings and stopes with pyrite	11	0.0007	0.0034	0.0018	0.0000	0.0010
20	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar	323	0.0000	0.3100	0.0024	0.0003	0.0177
202	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with mixed oxide and sulfide	25	0.0010	0.0030	0.0013	0.0000	0.0005
2027	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with mixed oxide and sulfide; broken and fractured	1	0.0020	0.0020	0.0020	0.0000	0.0000
203	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with pyrite	320	0.0001	0.0220	0.0016	0.0000	0.0015

Table 14.4: Gold Statistics for All Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)	Variance	Std. Dev.
2037	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with pyrite; broken and fractured	8	0.0020	0.0020	0.0020	0.0000	0.0000
2038	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with pyrite; fault with some clay in gouge	2	0.0001	0.0007	0.0004	0.0000	0.0004
2039	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with pyrite; mainly clay fault gouge	10	0.0010	0.0020	0.0011	0.0000	0.0003
207	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; broken and fractured	13	0.0000	0.0180	0.0028	0.0000	0.0060
208	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; fault with some clay in gouge	10	0.0000	0.0020	0.0004	0.0000	0.0007
209	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; mainly clay fault gouge	13	0.0000	0.0040	0.0009	0.0000	0.0010
21	Feldspar porphyry with quartz veins	4	0.0000	0.1970	0.0493	0.0097	0.0985

Statistics were calculated for the all of the rock codes which include the oxidation state and structure codes. Refer to **Table 14.4** for an explanation of all codes. **Figures 14.1A** and **14.1B** show bar graphs of the mean gold values for each rock type in the database.

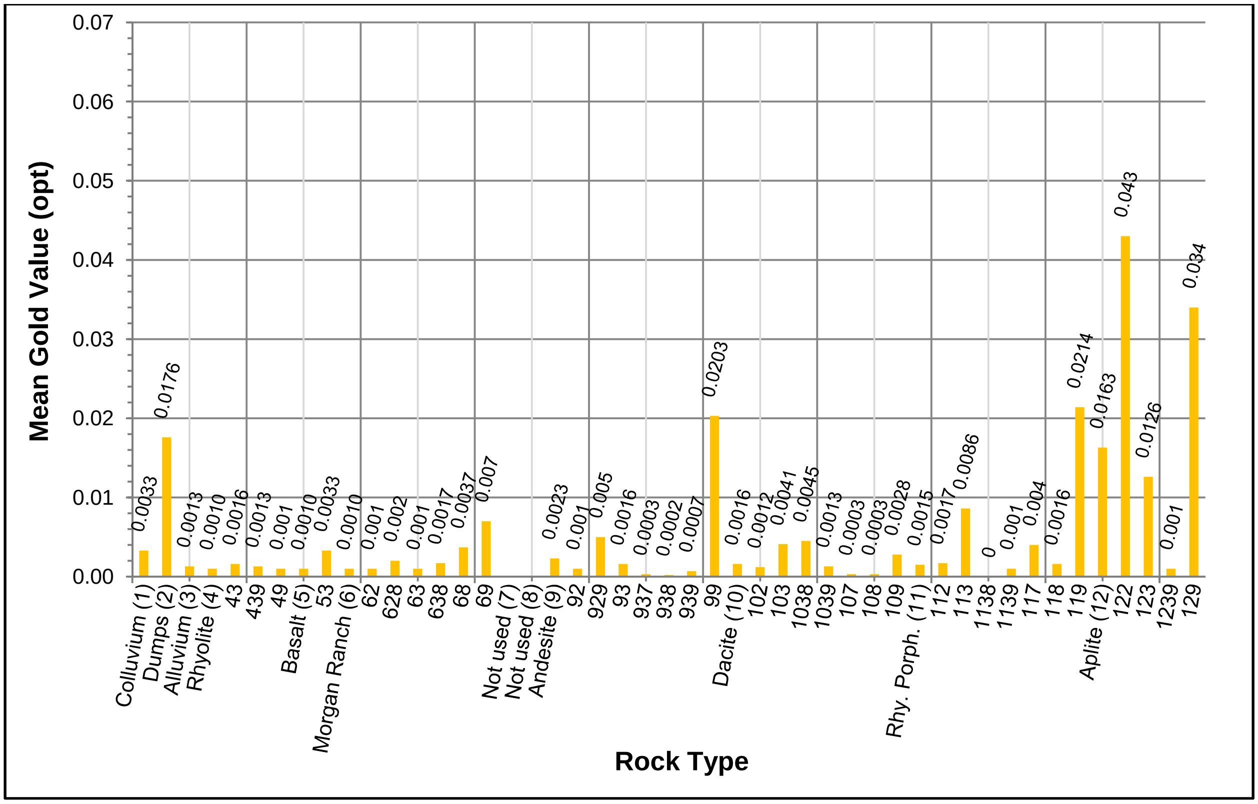


Figure 14.1A: Mean Gold Value by Rock Code in Drillhole Assays

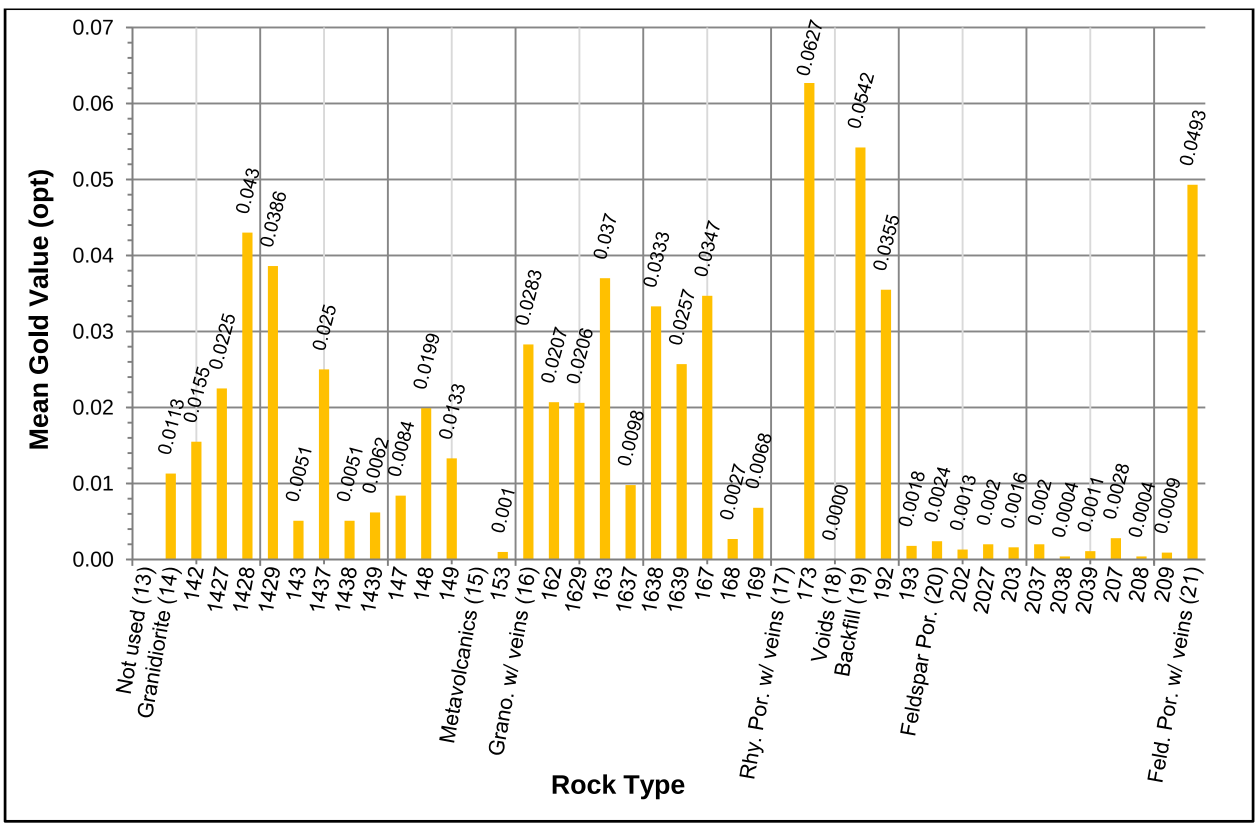


Figure 14.1B: Mean Gold Value by Rock Code in Drillhole Assays

14.3.2 Oxidation State Statistics

14.3.2.1 *Sulfides*

Rock codes which contain a "3" contain pyrite and other sulfides. **Table 14.5** shows selected statistics for rocks which contain sulfides. Refer to **Table 14.4** for variance and standard deviation for all rock types. **Figure 14.2** shows the mean gold values for all assay intervals in the database which are sulfidic.

Table 14.5: Gold Statistics for All Sulfide Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)*
43	Rhyolite dikes and flows	33	0.0010	0.0020	0.0016
439	Rhyolite dikes and flows with pyrite; mainly clay fault gouge	21	0.0010	0.0020	0.0013
53	Basalt dikes or flows	3	0.0010	0.0080	0.0033
63	Morgan Ranch Fm	13	0.0010	0.0010	0.0010
638	Morgan Ranch Fm; fault with some clay in gouge	3	0.0010	0.0030	0.0017
93	Andesite dikes with pyrite	32	0.0001	0.0070	0.0016
937	Andesite dikes; broken and fractured	1	0.0003	0.0003	0.0003
938	Andesite dikes; fault with some clay in gouge	1	0.0002	0.0002	0.0002
939	Andesite dikes; mainly clay fault gouge	2	0.0003	0.0010	0.0007
103	Dacite dikes	74	0.0000	0.0750	0.0041
107	Dacite dikes; broken and fractured	6	0.0000	0.0010	0.0003
108	Dacite dikes; fault with some clay in gouge	10	0.0000	0.0010	0.0003
109	Dacite dikes; mainly clay fault gouge	6	0.0000	0.0120	0.0028
1038	Dacite dikes; fault with some clay in gouge	2	0.0020	0.0070	0.0045
1039	Dacite dikes; mainly clay fault gouge	4	0.0010	0.0020	0.0013
113	Rhyolite porphyry dikes	159	0.0000	0.3850	0.0086
1138	Rhyolite porphyry dikes with pyrite; fault with some clay in gouge	5	0.0000	0.0000	0.0000
1139	Rhyolite porphyry dikes; mainly clay fault gouge	3	0.0010	0.0010	0.0010
123	Aplite dikes or zones	56	0.0010	0.1630	0.0126
1239	Aplite dikes or zones; mainly clay fault gouge	1	0.0010	0.0010	0.0010
143	Granodiorite	5,770	0.0000	0.7720	0.0051
1438	Granodiorite; fault with some clay in gouge	138	0.0000	0.2200	0.0051
1439	Granodiorite; mainly clay fault gouge	568	0.0000	0.7200	0.0062
153	Metavolcanics	1	0.0010	0.0010	0.0010
163	Granodiorite with quartz veins	259	0.0000	**2.2700**	0.0370
1637	Granodiorite with quartz veins; broken and fractured	11	0.0001	0.0498	0.0098
1638	Granodiorite with quartz veins; fault with some clay in gouge	18	0.0000	0.5190	0.0333
1639	Granodiorite with quartz veins; mainly clay fault gouge	27	0.0000	0.5490	0.0257
173	Rhyolite porphyry with quartz veins	7	0.0010	0.2750	0.0627
193	Backfill or caved rock material in mine workings and stopes	11	0.0007	0.0034	0.0018

Table 14.5: Gold Statistics for All Sulfide Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)*
203	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar	320	0.0001	0.0220	0.0016
2037	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; broken and fractured	8	0.0020	0.0020	0.0020
2038	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; fault with some clay in gouge	2	0.0001	0.0007	0.0004
2039	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; mainly clay fault gouge	10	0.0010	0.0020	0.0011
* See Table 14.4 for variance and standard deviation		7,555			

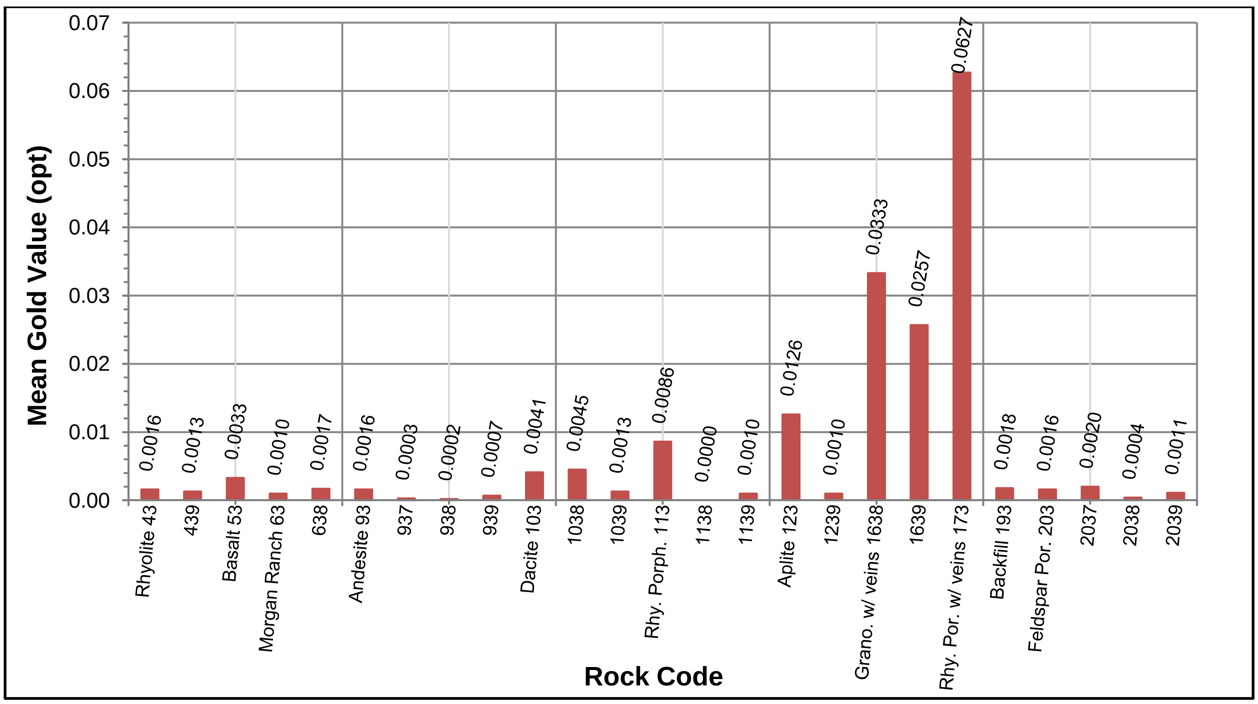


Figure 14.2: Mean Gold Value by Rock Code in Sulfide Rock Types

14.3.2.2 *Mixed Oxides and Sulfides*

As described previously, rock codes which contain a “2” have an oxidation state which is mixed oxide and sulfide. **Table 14.6** shows selected statistics for rocks which contain mixed oxide and sulfide rock types. Refer to **Table 14.4** for variance and standard deviation for all rock types. **Figure 14.3** shows the mean gold values for all assay intervals in the database which are mixed oxide and sulfide.

Table 14.6: Gold Statistics for All Mixed Oxide and Sulfide Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)*
62	Morgan Ranch Fm	5	0.0010	0.0010	0.0010
628	Morgan Ranch Fm; fault with some clay in gouge	1	0.0020	0.0020	0.0020
92	Andesite dikes	3	0.0010	0.0010	0.0010
929	Andesite dikes; mainly clay fault gouge	3	0.0010	0.0090	0.0050
102	Dacite dikes	5	0.0010	0.0020	0.0012
112	Rhyolite porphyry dikes	12	0.0010	0.0070	0.0017
122	Aplite dikes or zones	1	0.0430	0.0430	0.0430
142	Granodiorite	444	0.0010	**1.0930**	0.0155
1427	Granodiorite; broken and fractured	1	0.0225	0.0225	0.0225
1428	Granodiorite; fault with some clay in gouge	2	0.0378	0.0482	0.0430
1429	Granodiorite; mainly clay fault gouge	25	0.0010	0.7820	0.0386
162	Granodiorite with quartz veins	37	0.0010	0.2280	0.0207
1629	Granodiorite with quartz veins; mainly clay fault gouge	2	0.0070	0.0342	0.0206
192	Backfill or caved rock material in mine workings and stopes	11	0.0010	0.0934	0.0355
202	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar	25	0.0010	0.0030	0.0013
2027	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; broken and fractured	1	0.0020	0.0020	0.0020
* See Table 14.4 for variance and standard deviation		578			

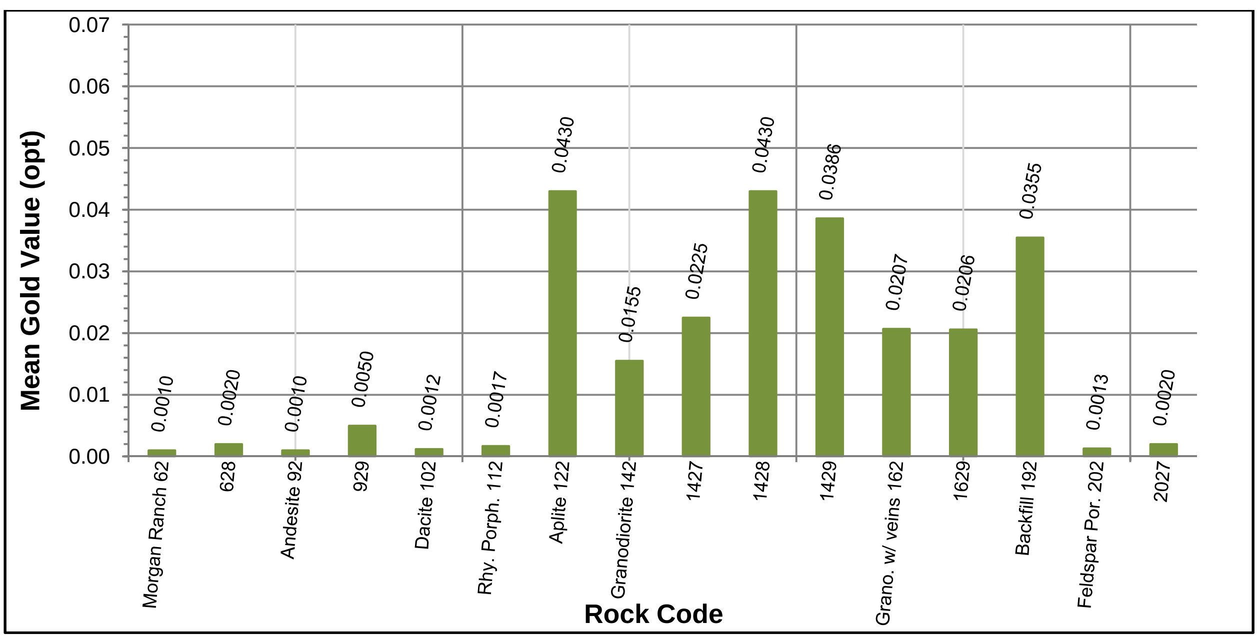


Figure 14.3: Mean Gold Value by Rock Code in Mixed Oxide and Sulfide Rock Types

14.3.2.3 *Oxides*

Rock codes which do not contain either a “2” or a “3” are oxidized. **Table 14.7** shows selected statistics for rocks which contain oxides. Refer to **Table 14.4** for variance and standard deviation for all rock types. **Figure 14.4** shows the mean gold values for all assay intervals in the database which are oxidized.

Table 14.7: Gold Statistics for All Oxidized Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)*
1	Colluvium and slope cover, talus, overburden, etc.	390	0.0000	0.2840	0.0033
2	Mine dump material	40	0.0000	0.0990	0.0176
3	Alluvium, gravel, sand and silt stream deposits	6	0.0010	0.0030	0.0013
4	Rhyolite dikes and flows	35	0.0010	0.0010	0.0010
49	Rhyolite dikes and flows; mainly clay fault gouge	1	0.0010	0.0010	0.0010
5	Basalt dikes or flows	3	0.0010	0.0010	0.0010
6	Morgan Ranch Fm; conglomerate, sandstone, clay, limestone	787	0.0001	0.0170	0.0010
68	Morgan Ranch Fm; fault with some clay in gouge	37	0.0010	0.0620	0.0037
69	Morgan Ranch Fm; mainly clay fault gouge	4	0.0010	0.0160	0.0070
9	Andesite dikes; may also be fine-grained phase of granodiorite	16	0.0010	0.0060	0.0023
99	Undefined	219	0.0000	0.2700	0.0203
10	Dacite dikes; may also be fine-grained phase of granodiorite	169	0.0000	0.0940	0.0016
107	Dacite dikes; broken and fractured	6	0.0000	0.0120	0.0028
11	Rhyolite porphyry dikes; pinkish gray groundmass w/ feldspar phenocrysts; sometimes called felsite	797	0.0000	0.2400	0.0015
117	Rhyolite porphyry dikes; broken and fractured	15	0.0000	0.0440	0.0040
118	Rhyolite porphyry dikes; fault with some clay in gouge	8	0.0000	0.0070	0.0016
119	Rhyolite porphyry dikes; mainly clay fault gouge	27	0.0000	0.3780	0.0214
12	Aplite dikes or zones; also sometimes called felsite	11	0.0010	0.0480	0.0163
129	Aplite dikes or zones; mainly clay fault gouge	1	0.0340	0.0340	0.0340
14	Granodiorite; fine, medium, & coarse grained; often biotized	2,916	0.0000	**1.9060**	0.0113
147	Granodiorite; broken and fractured	72	0.0000	0.2450	0.0084
148	Granodiorite; fault with some clay in gouge	189	0.0000	**1.1700**	0.0199
149	Granodiorite; mainly clay fault gouge	241	0.0000	0.7150	0.0133
15	Metavolcanics				
16	Granodiorite with quartz veins	178	0.0000	0.3200	0.0283
167	Granodiorite with quartz veins; broken and fractured	13	0.0000	0.1390	0.0347
168	Granodiorite with quartz veins; fault with some clay in gouge	16	0.0000	0.0147	0.0027
169	Granodiorite with quartz veins; mainly clay fault gouge	27	0.0000	0.0400	0.0068
17	Rhyolite porphyry with quartz veins				

Table 14.7: Gold Statistics for All Oxidized Rock Types

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)*
18	Void; mine workings, stopes, back-filled workings, zones of wood debris	0	–	–	–
19	Backfill or caved rock material in mine workings and stopes	35	0.0010	0.6830	0.0542
20	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar	323	0.0000	0.3100	0.0024
207	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; broken and fractured	13	0.0000	0.0180	0.0028
208	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; fault with some clay in gouge	10	0.0000	0.0020	0.0004
209	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar; mainly clay fault gouge	13	0.0000	0.0040	0.0009
21	Feldspar porphyry with quartz veins	4	0.0000	0.1970	0.0493
* See Table 14.4 for variance and standard deviation		6,634			

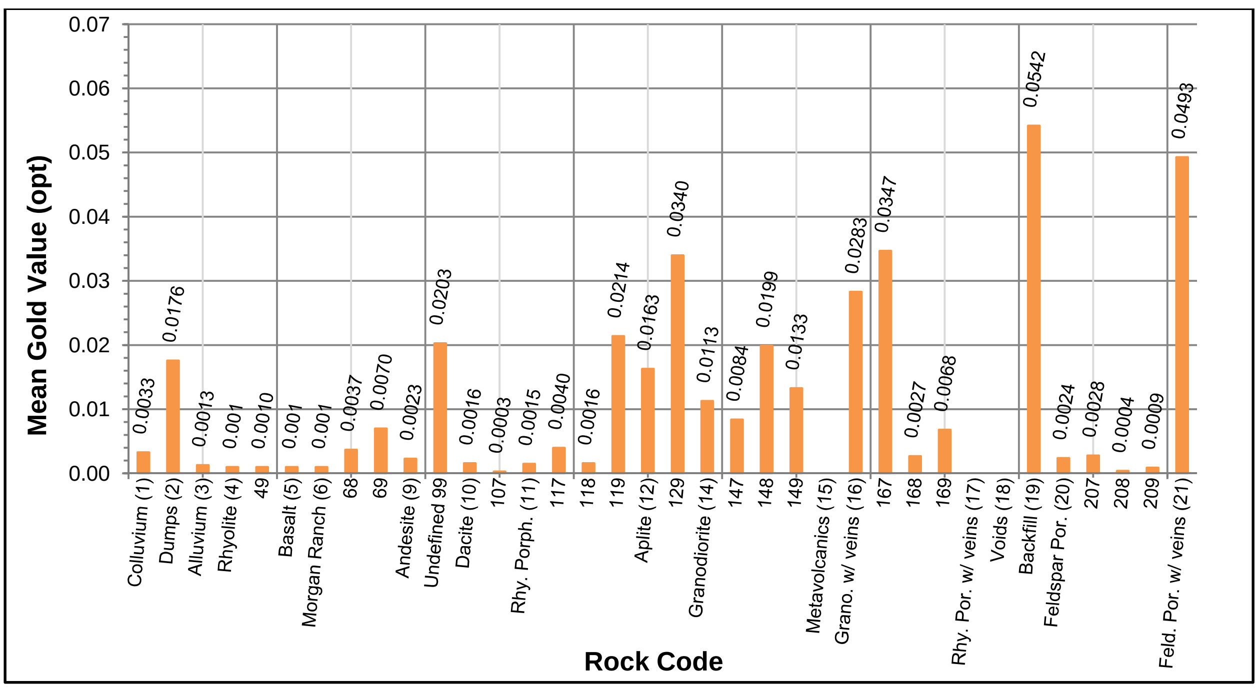


Figure 14.4: Mean Gold Value by Rock Code in Oxide Rock Types

14.3.3 Structure Statistics

14.3.3.1 *Broken and Fractured Rock*

Rock codes which contain a “7” are broken or fractured. **Table 14.8** shows selected statistics for rocks which are broken or fractured. Refer to **Table 14.4** for variance and standard deviation for all rock types. **Figure 14.5** shows the mean gold values for all assay intervals in the database which are broken or fractured.

Table 14.8: Gold Statistics for Broken and Fractured Rocks

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)*
937	Andesite dikes with pyrite	1	0.0003	0.0003	0.0003
107	Dacite dikes	6	0.0000	0.0120	0.0028
117	Rhyolite porphyry dikes	15	0.0000	0.0440	0.0040
147	Granodiorite	72	0.0000	0.2450	0.0084
1427	Granodiorite with mixed oxide and sulfide	1	0.0225	0.0225	0.0225
1437					
1637	Granodiorite with quartz veins with pyrite	11	0.0001	0.0498	0.0098
167	Granodiorite with quartz veins	13	0.0000	0.1390	0.0347
2027	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with mixed oxide and sulfide	1	0.0020	0.0020	0.0020
2037	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with pyrite	8	0.0020	0.0020	0.0020
207	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar	13	0.0000	0.0180	0.0028
* See Table 14.4 for variance and standard deviation		141			

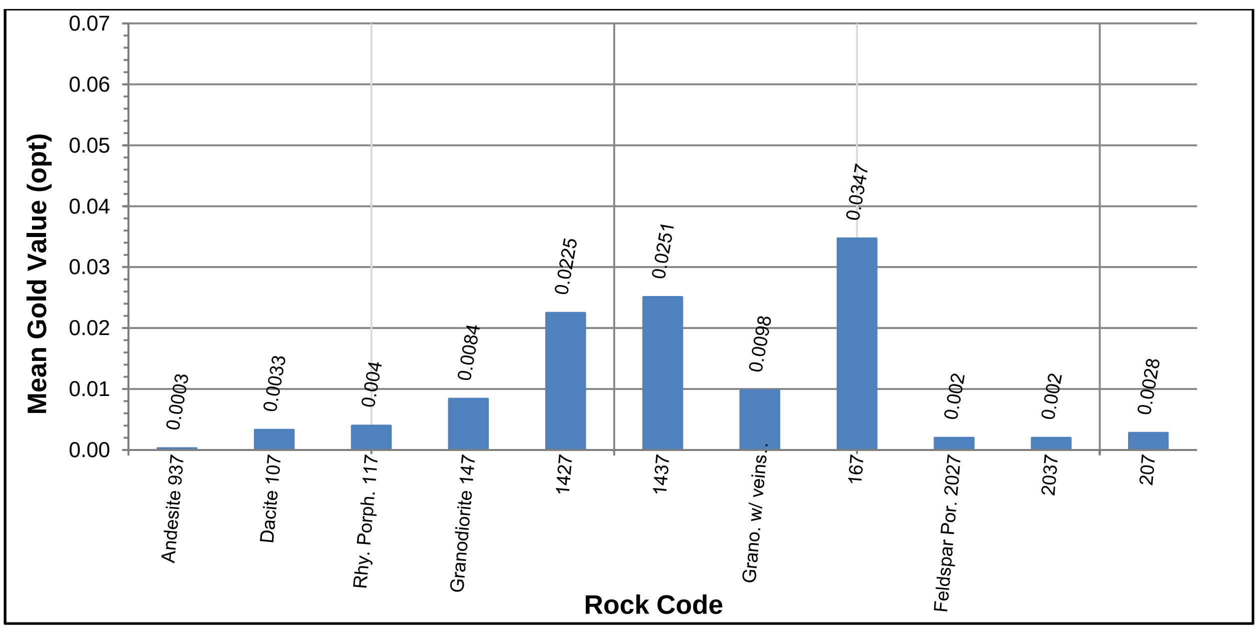


Figure 14.5: Mean Gold Value by Rock Code in Broken and Fractured Rock Types

14.3.3.2 *Faulted Rocks with some Clay as Fault Gouge*

Rock codes which contain an "8" have faults with some clay as fault gouge. **Table 14.9** shows selected statistics for rocks which have faults with some clay as fault gouge. Refer to **Table 14.4** for variance and standard deviation for all rock types. **Figure 14.6** shows the mean gold values for all assay intervals in the database which have faults with some clay as fault gouge.

Table 14.9: Gold Statistics for All Rock Types with some Clay as Fault Gouge

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)
68	Morgan Ranch Fm	37	0.0010	0.0620	0.0037
628	Morgan Ranch Fm with mixed oxide and sulfide	1	0.0020	0.0020	0.0020
638	Morgan Ranch Fm with pyrite	3	0.0010	0.0030	0.0017
938	Andesite dikes with pyrite	1	0.0002	0.0002	0.0002
1038	Dacite dikes with pyrite	2	0.0020	0.0070	0.0045
118	Rhyolite porphyry dikes	8	0.0000	0.0070	0.0016
148	Granodiorite	189	0.0000	**1.1700**	0.0199
1428	Granodiorite with mixed oxide and sulfide	2	0.0378	0.0482	0.0430
1437	Granodiorite with pyrite; broken and fractured	115	0.0000	**2.2500**	0.0251
1438	Granodiorite with pyrite	138	0.0000	0.2200	0.0051
168	Granodiorite with quartz veins	16	0.0000	0.0147	0.0027
1638	Granodiorite with quartz veins with pyrite	18	0.0000	0.5190	0.0333
208	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar	10	0.0000	0.0020	0.0004
2038	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with pyrite	2	0.0001	0.0007	0.0004
* See Table 14.4 for variance and standard deviation		427			

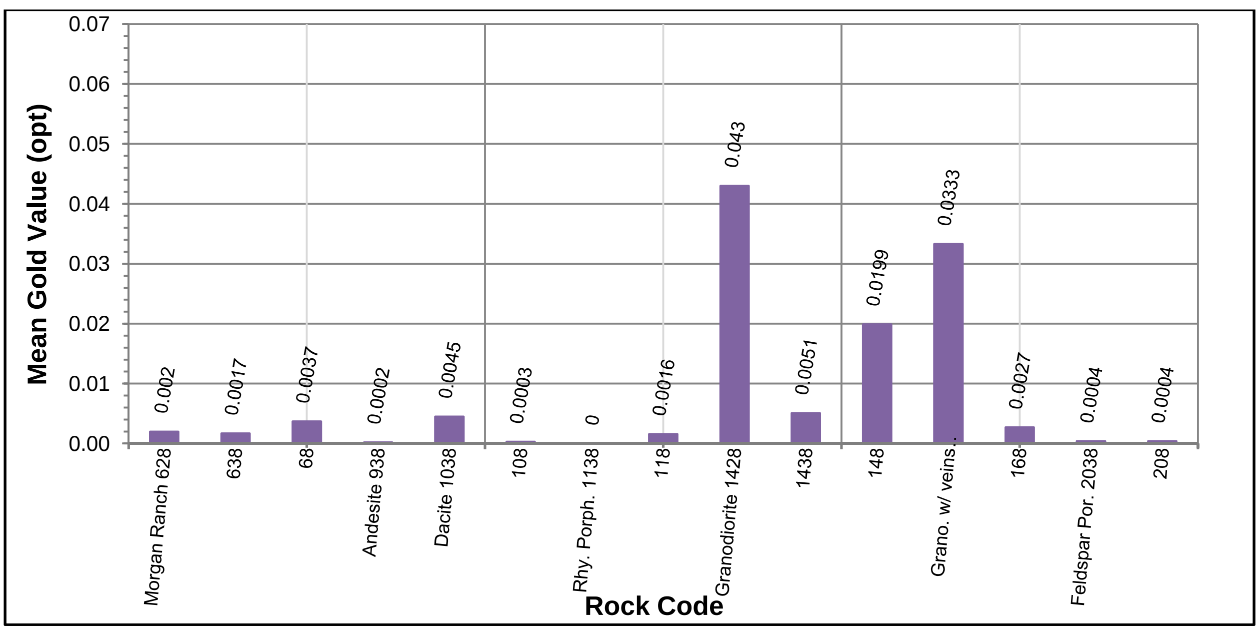


Figure 14.6: Mean Gold Value by Rock Code in Faulted Rock Types with some Clay as Fault Gouge

14.3.3.3 *Faulted Rocks with Mainly Clay as Fault Gouge*

Rock codes which contain an “9” have faults with mainly clay as fault gouge. **Table 14.10** shows selected statistics for rocks which have faults with mainly clay as fault gouge. Refer to **Table 14.4** for variance and standard deviation for all rock types. **Figure 14.7** shows the mean gold values for all assay intervals in the database which have faults with mainly clay as fault gouge.

Table 14.10: Gold Statistics for All Rock Types with Mainly Clay as Fault Gouge

Rock Code	Rock Type	# of Samples	Minimum (opt)	Maximum (opt)	Mean (opt)*
49	Rhyolite dikes and flows	1	0.0010	0.0010	0.0010
69	Morgan Ranch Fm	4	0.0010	0.0160	0.0070
119	Rhyolite porphyry dikes	27	0.0000	0.3780	0.0214
129	Aplite dikes or zones	1	0.0340	0.0340	0.0340
149	Granodiorite	241	0.0000	0.7150	0.0133
169	Granodiorite with quartz veins	27	0.0000	0.0400	0.0068
209	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar	13	0.0000	0.0040	0.0009
929	Andesite dikes with mixed oxide and sulfide	3	0.0010	0.0090	0.0050
939	Andesite dikes with pyrite	2	0.0003	0.0010	0.0007
1039	Dacite dikes with pyrite	4	0.0010	0.0020	0.0013
1139	Rhyolite porphyry dikes with pyrite	3	0.0010	0.0010	0.0010
1239	Aplite dikes or zones with pyrite	1	0.0010	0.0010	0.0010
1429	Granodiorite with mixed oxide and sulfide	25	0.0010	0.7820	0.0386
1439	Granodiorite with pyrite	568	0.0000	0.7200	0.0062
1629	Granodiorite with quartz veins with mixed oxide and sulfide	2	0.0070	0.0342	0.0206
1639	Granodiorite with quartz veins with pyrite	27	0.0000	0.5490	0.0257
2039	Feldspar porphyry dikes; granite K-spar porphyry with conspicuous pink & white feldspar with pyrite	10	0.0010	0.0020	0.0011
* See Table 14.4 for variance and standard deviation		1,178			

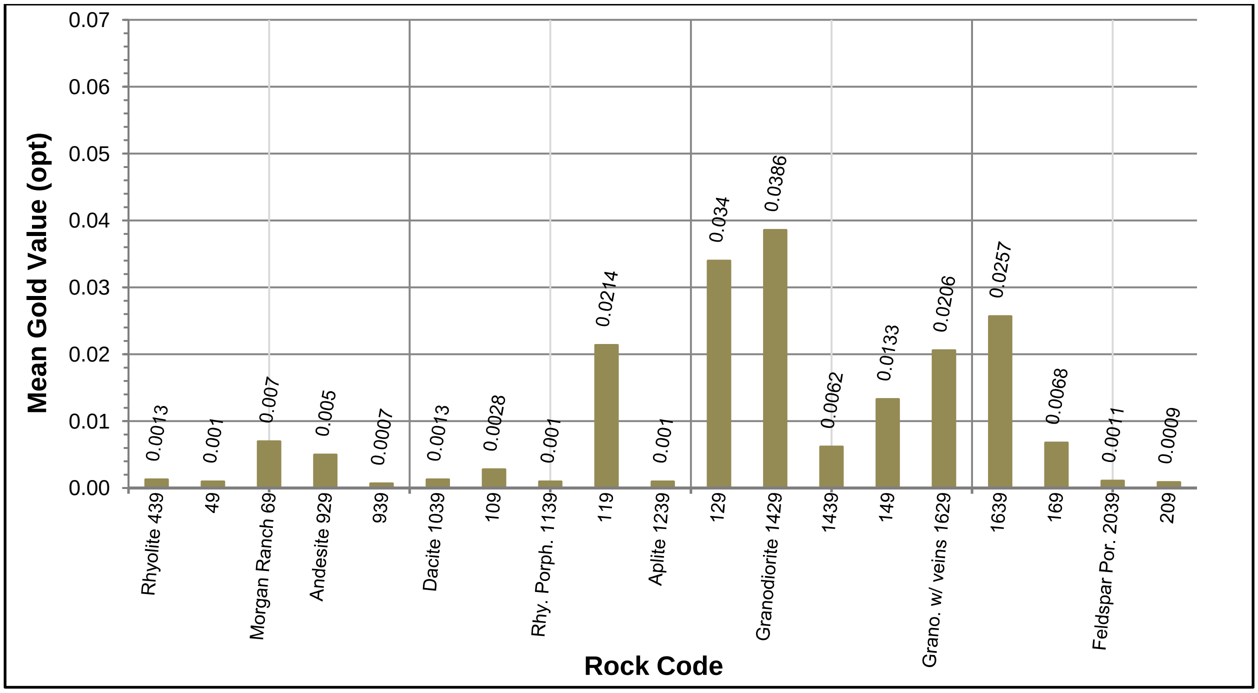


Figure 14.7: Mean Gold Value by Rock Code in Faulted Rock Types with Mainly Clay as Fault Gouge

14.3.4 Weighted Statistics and Discussion of Rock Codes

The unweighted statistics of rock codes and mean grades do not readily convey the importance of certain rock types in the Wilson and Wheeler deposits. Some rock codes have higher grade than other rock codes, but the higher grade may be based upon relatively few samples. For example, Rock Code 21 is feldspar porphyry with quartz veins with an average grade of 0.0493 opt Au (See **Table 14.4**), but there are only 4 intervals in entire database of 14,767 intervals which are Rock Code 21. So the influence on the resource estimate of these four higher grade intervals is minor.

Figures 14.8A and **14.8B** show the weighted average of gold values by rock code. The data was separated into two graphs because there are too many rock codes to display on one graph. Both graphs have the same maximum value on the Y-axis for easy comparison between graphs. All rock codes that begin with a number between 1 and 12 (**Figure 14.8A**) have relatively few samples. Regardless of average grade in those rock codes, the over impact on the resource model is minor.

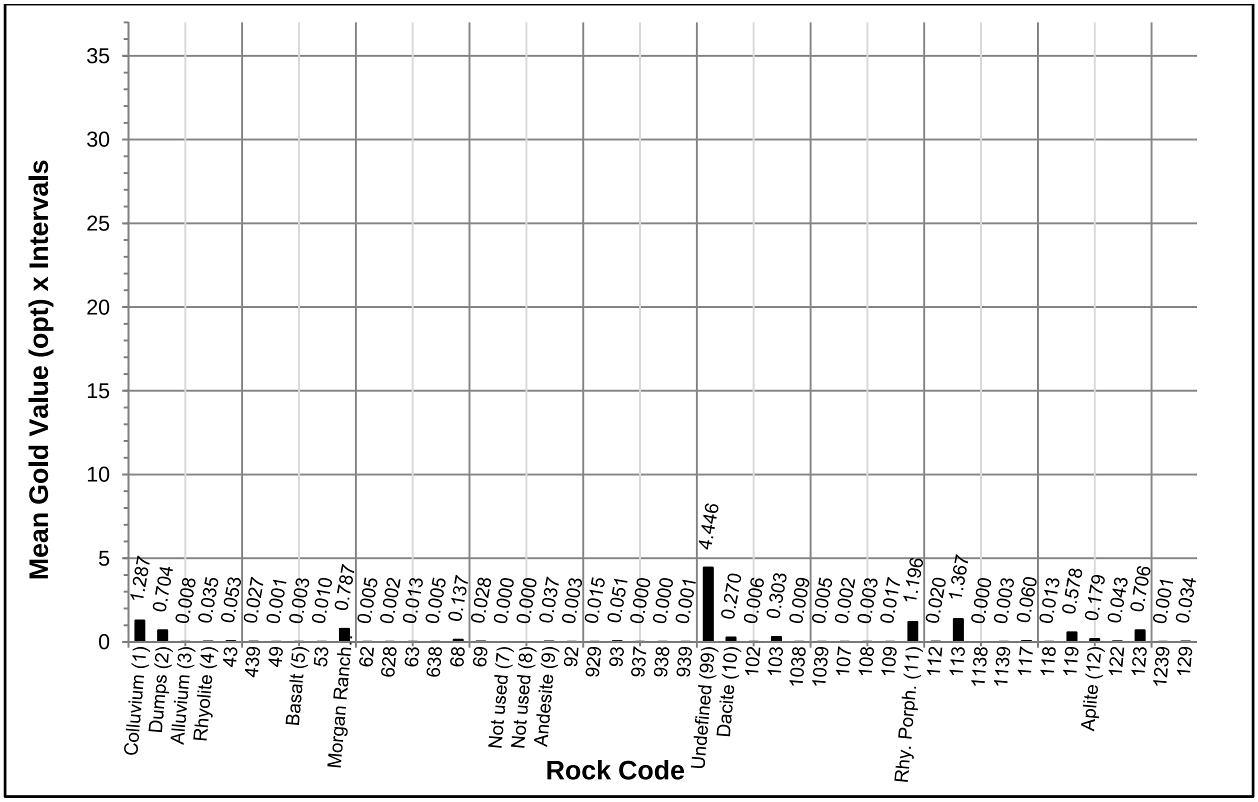


Figure 14.8A: Weighted Average of Average Gold Grade Times Number of Intervals for Rock Codes 1 Through 12

Certain rock codes shown in **Figure 14.8B** (Rock Codes 13 through 21) have significant impact on the resource model. Weighted averages of mean grade times number of intervals for granodiorite (Rock Codes 14 and 16 and all derivations thereof) are the highest in the database and therefore have the most significant impact on the resource model.

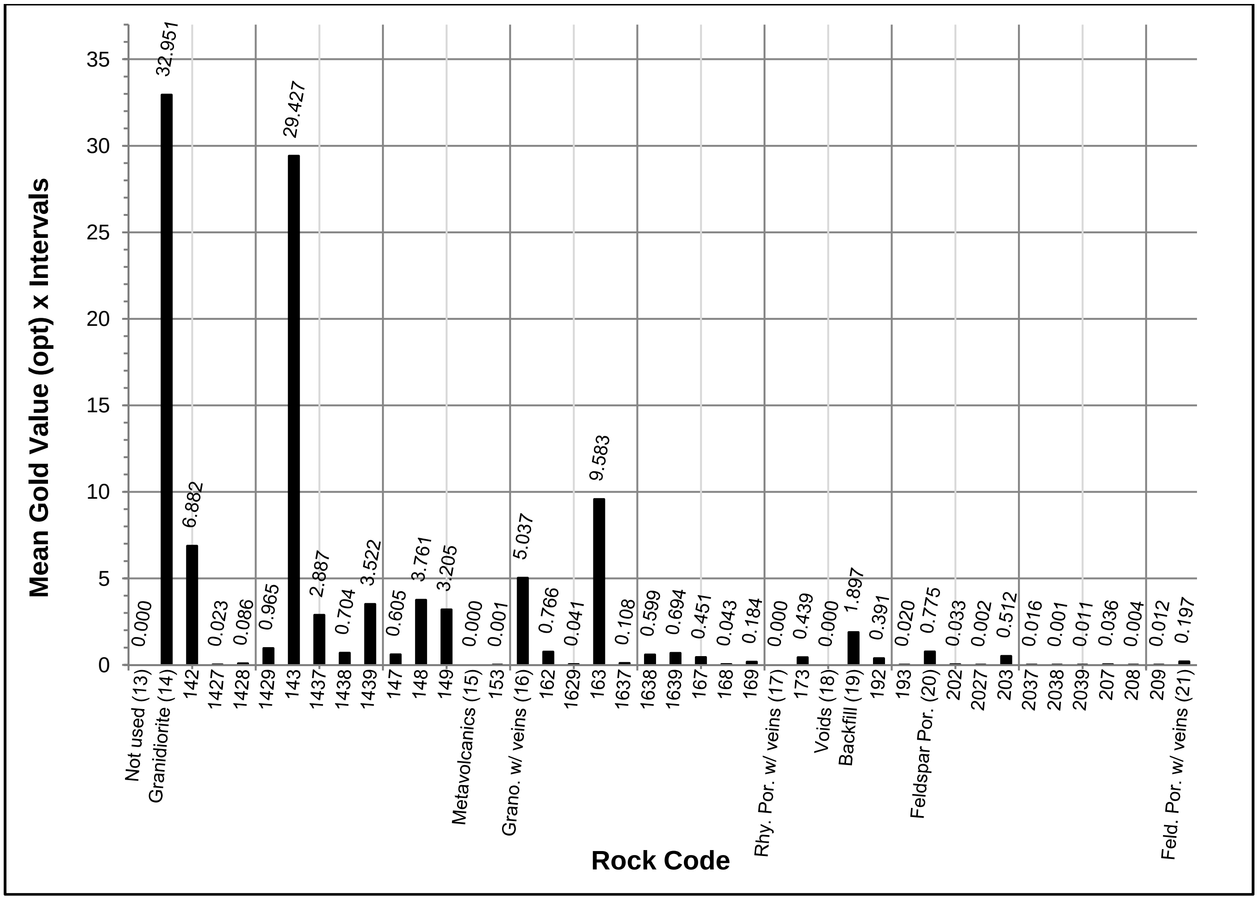


Figure 14.8B: Weighted Average of Average Gold Grade Times Number of Intervals for Rock Codes 13 Through 21

14.3.5 Creation of 3-D Rock Model

The 3-D rock model was generated based on two sets of section maps provided by Lincoln geologists. There were 33 sections for the Wheeler deposit (600N through 2450N) looking NNW. There were 29 sections for the Wilson deposit (2469200E through 2470550E) looking west. Twelve separate rock types are identified in these sections, including Qcol(1), Dump(2), Qal(3), Tb(5), Tc(6), And(9), Dac(10), Rp(11), Apl(12), Grd(14), Tv(15), and Fp(20).

The majority of the sections were spaced 50 feet apart. The end sections in each set were projected out 25 feet, while the internal sections were projected half way to the next section.

3-D blocks that were within 7.5 block widths (7.5 feet) of an underground channel sample were recoded as 99. 3-D blocks that were within 25 horizontal feet of a sample coded as void (code 18) were recoded as 18. 3-D blocks that were within 25 horizontal feet of a sample coded as backfill (code 19) were recoded as 19. A combined total of 2167 blocks were recoded as either channel, void, or backfill. These blocks were treated as sterile waste blocks.

14.3.6 Rock Codes which were Excluded from the Resource Estimate

Rock Code 1 represents colluvium and slope cover, talus and overburden while Rock Code 3 represents alluvium, gravel, sand and silt stream deposits. During modeling of grade, blocks containing these rock codes were excluded from the resource because both types of rocks are recent as compared to the mineralization.

Rock Code 2 is historic dump material which remains on the surface in various places around the Pine Grove property. Historic dumps are planned to be crushed and placed on the heap leach pad, but they are not considered to be part of the in-situ resource. Blocks coded as dump material were not modeled and were treated as sterile waste.

Rock Code 6 is the Morgan Ranch Formation (conglomerate, sandstone, clay and limestone). The Morgan Ranch Formation is Tertiary in age and unconformably overlies the Mesozoic intrusive igneous host rocks. For this reason, all blocks with Rock Code 6 were treated as sterile waste.

Rock Code 9 (andesite dikes which may be a fine-grained phase of granodiorite) was also excluded from the resource. These dikes intruded the host rocks after mineralization and are therefore will not carry grade. Blocks coded 9 were treated as sterile waste.

Rock Code 18 represents voids which were encountered during drilling. The voids are most likely old workings from historic mining. Blocks of Rock Code 18 were treated as sterile waste.

Rock Code 19 is backfill. Old workings at Pine Grove were often backfilled with waste during historic mining operations. As modern drilling encountered such backfilled areas, the sample intervals were identified and logged in the drill log as backfill. Samples from backfilled areas were analyzed along with all other intervals. The backfill has relatively high average grade (0.0542 opt Au) in 35 intervals with Rock Code 19. However, backfill material is by definition not in-situ, so it is not appropriate to use any blocks which are coded 19 in the current resource estimate. Therefore, the blocks coded as backfill were treated as sterile waste.

Rock Code 99 represents underground drifts that have been channel sampled. Since there are no assay certificates available for these channel samples, they could not be used in the resource estimate. Blocks that were coded as 99 were treated as sterile waste.

3-D block count for all excluded blocks is summarized by Rock Code in **Table 14.11**.

Table 14.11: Blocks Excluded from the Resource Estimate by Rock Code

Rock Code	Number of Blocks
1	16570
2	1338
3	6240
6	153497
9	1850
18	1314
19	377
99	476
Total	**181662**

14.4 Modeled Area Descriptions

The southwest corner of the block model is located at 14,552,100 ft North, 2,468,600 ft East (Nevada State Plane West Zone NAD83 feet) with an elevation of 6,100 feet (See **Figure 14.9**). The modeled area has an orientation of north-south (0°) and contains 380 rows, 600 columns and 250 levels. Each block has the following dimensions (x,y,z): 10 feet (3.05 m) per row, 10 feet (3.05 m) per column and a block height of 10 feet (3.05 m). See **Table 14.12** for a summary of parameters used in the resource model.

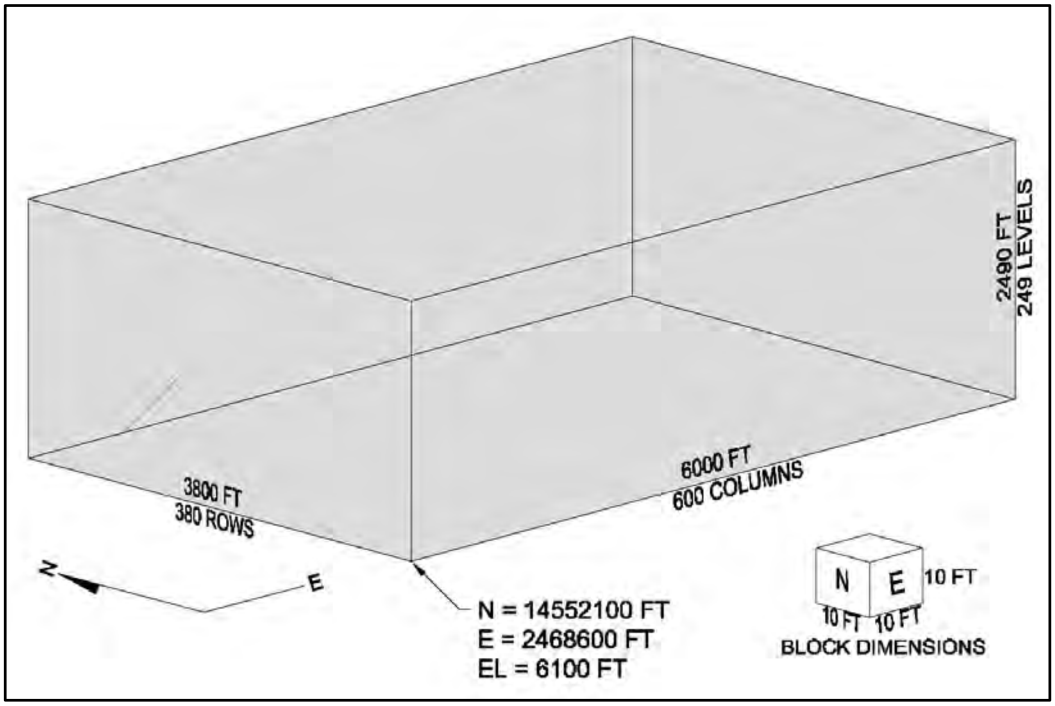


Figure 14.9: Block Model Limits

Table 14.12: Block Parameters for the Pine Grove Model

Block size (ft)			Dip	Rake	Orientation	Number of Rows	Number of Columns	Number of Levels
X	Y	Z						
10	10	10	0°	0°	0°	380	600	250

The number of drillholes used in the model totals 261 holes totaling 15,472 sampled intervals. The total footage of drilling involved in the resource estimate is approximately 78,577 ft (23,579 m). **Figure 10.1** shows collar locations and drillhole traces of the drillholes used in the model.

14.5 Capping of High Grades

Review of the sample cumulative frequency curve as well as decile analysis show that the capping of high grade values is not warranted. However, the influence of each of the six assay values that are greater than 1.0 opt, which average 1.72 opt, was limited to a distance of fifty feet.

14.6 Bulk Density

For all rock types in the Pine Grove estimate, Telesto used a weighted average density of 2.13 tons/yd^3 (12.63 ft^3 per ton). This is the weighted average of individual rocks types that were tested for density to calculate a global value for specific gravity. These tests were conducted by McClelland Labs in 2009 and consisted of 8 specific rock types. **Table 13.6** shows density tests which were performed by Lincoln. No independent bulk density testing was performed by Telesto.

14.7 Cross Sections through the Modeled Areas

Figure 14.10 shows the location of a cross section created by Telesto which is generally perpendicular to the strike of the Wilson deposit and is considered to be representative of the resource. The Wilson cross section is shown in **Figures 14.11**. In the cross section, gold intercepts are shown as color-coded numbers on the right side of the drillhole trace. **Figure 14.12** shows the location of a cross section which is generally perpendicular to the strike of the Wheeler deposit and is considered to be representative of the resource. The Wheeler cross section is shown in **Figures 14.13**.

Figure 14.10: Cross-Section Location – Wilson Deposit

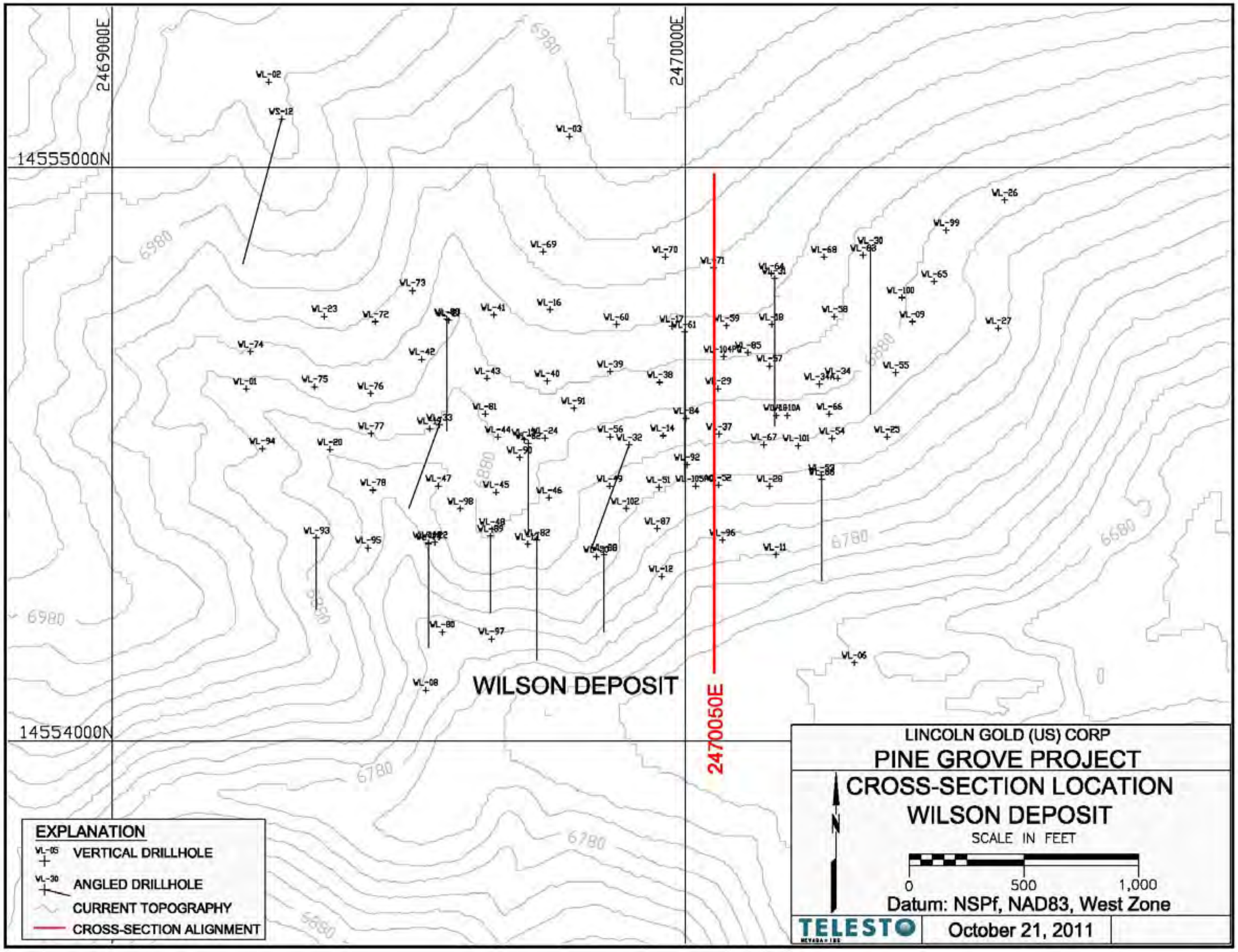


Figure 14.11: Drill Hole Gold Intercept Cross-Section – Wilson Deposit

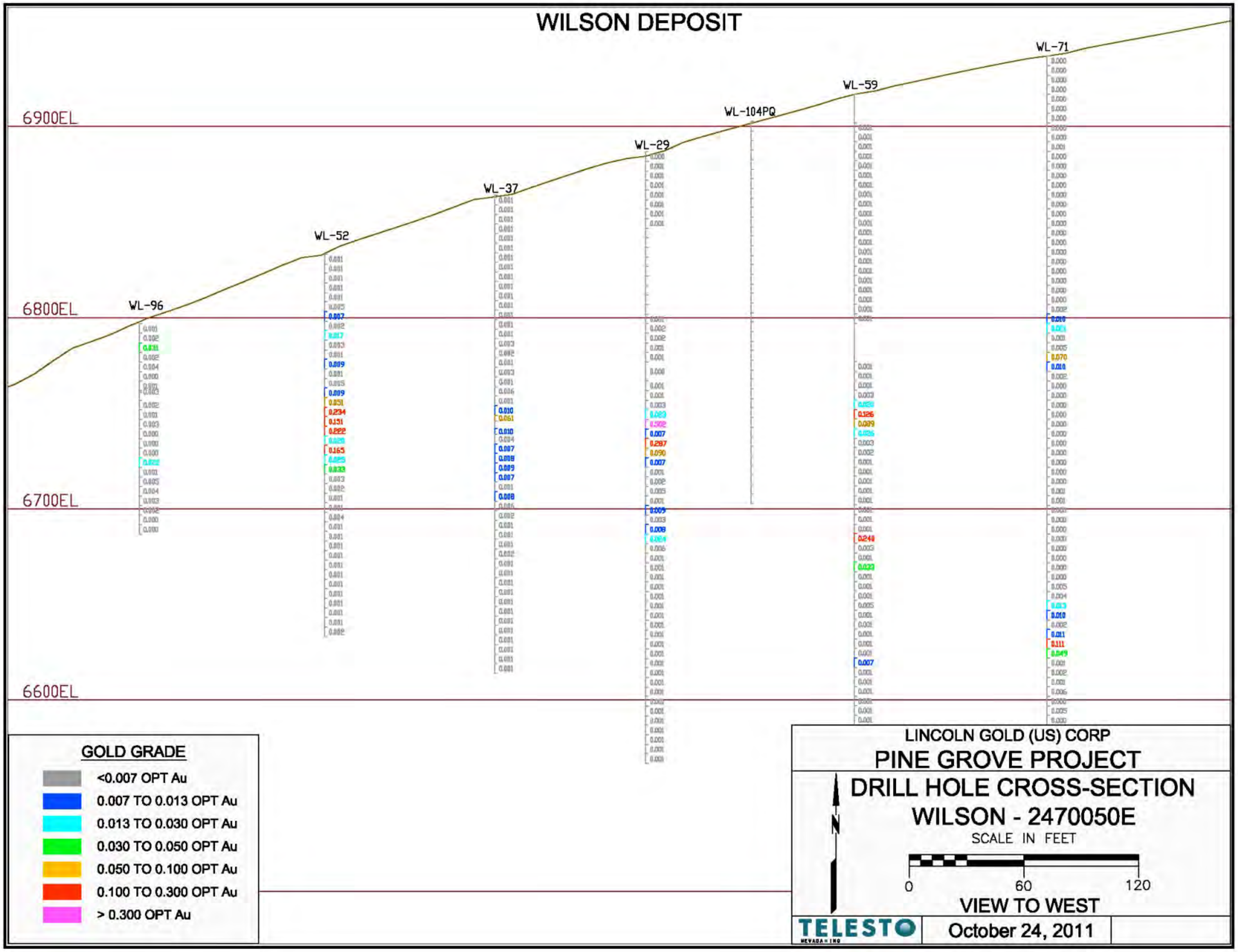


Figure 14.12: Cross-Section Location – Wheeler Deposit

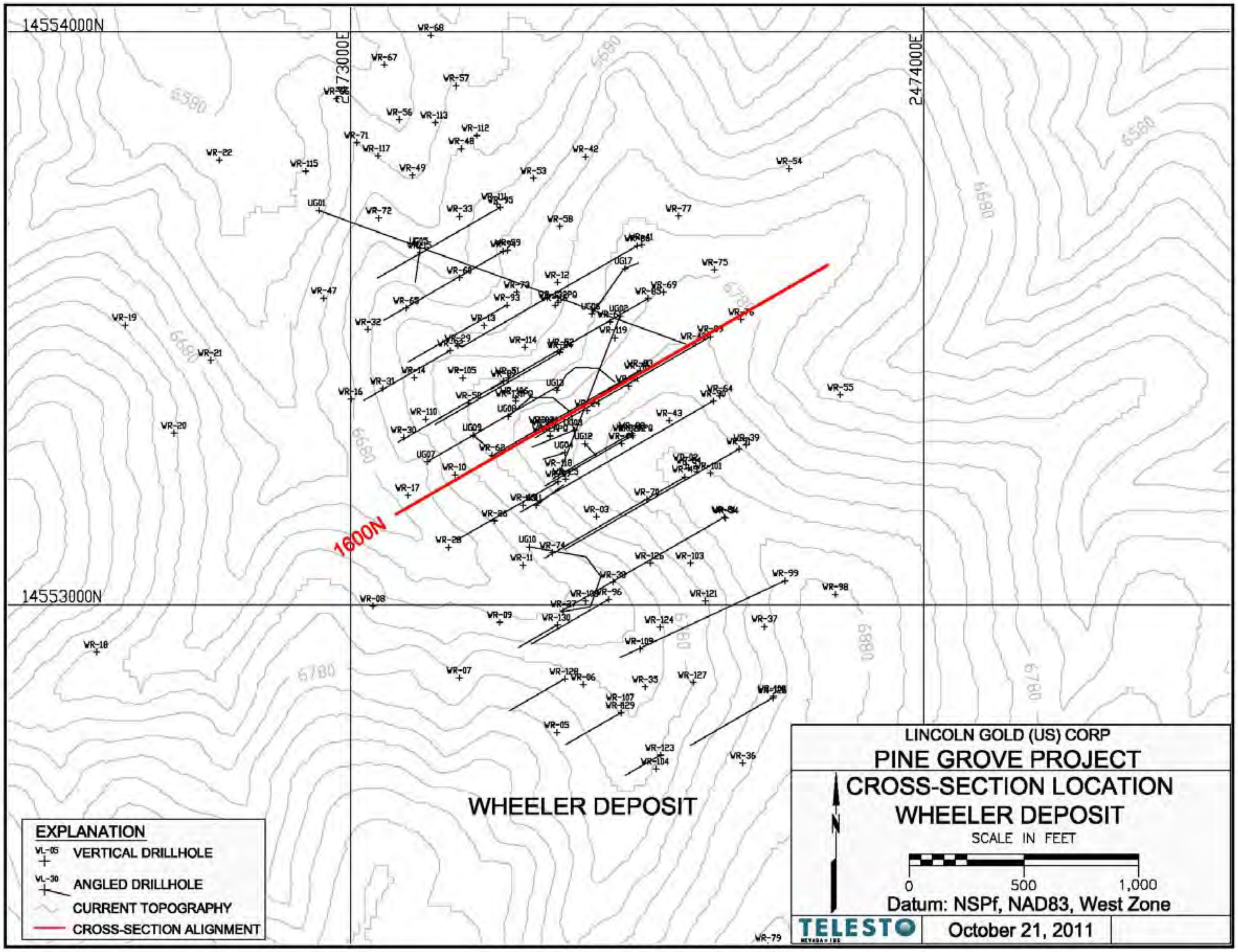
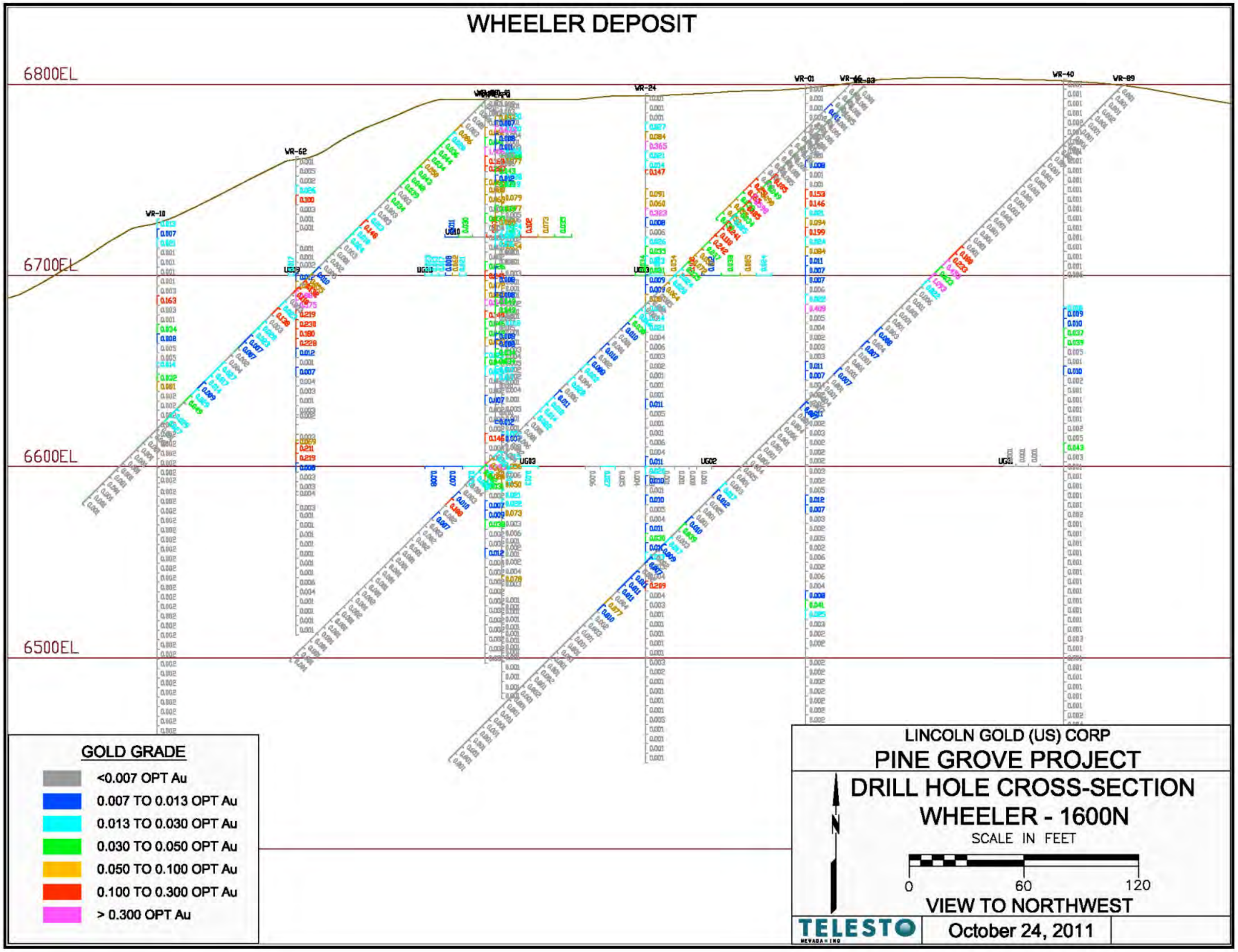


Figure 14.13: Drill Hole Gold Intercept Cross-Section – Wheeler Deposit



14.8 Block Model

Prior to modeling, a new set of simplified rock codes for the sample intervals was generated. The encoded values for oxidation state and structure were removed from the original Pine Grove code, leaving only the main lithology code. For example, code 1427 which represents granodiorite in mixed oxide/sulfide with fracturing becomes code 14.

The granodiorite unit contains the majority of the gold mineralized intercepts. Ninety-six percent of the mineralized tons above cutoff in the resource are from granodiorite. Rhyolite porphyry (code 11) contains just over two percent of the tons, and the only other rock types containing gold above the cutoff are dacite dikes (code 10), Aplite (code 12), and Feldspar Porphyry (code 20). The last three rock types combined represent less than two percent of the resource. Basalt Dikes (code 5) and Metavolcanics (code 15) contained insufficient data points to model and as such, do not contain any mineralized material in the model.

A study of the grade profiles across the contacts between granodiorite and the other four mineralized lithology types showed that there is a discontinuity at each boundary. As such, the five gold bearing rock units were modeled separately, using only assay intervals that matched the lithology code of the 3-D rock type being modeled. The **Table 14.13** summarizes which lithology units were modeled, and the source of assay data for each.

Table 14.13: Modeled Lithologies Summary

3-D Rock Code	Assay Interval Codes Used to Model 3-D Rock	Description
10	10	Dacite Dikes
11	11	Rhyolite Porphyry Dikes
12	12	Aplite Dikes
14	14,16	Granodiorite
20	20	Feldspar Porphyry

14.8.1 Search Parameters

The inverse-distance squared method was applied in the modeling process. Based on the results of the geostatistical analysis of drillhole data for the predominate granodiorite rock types (14 and 16), search distances were established to estimate grade in the block model for each category of resources. The search orientations for the current estimate are shown in **Table 14.14**.

Table 14.14: Block Model Search Parameters

Category	Dip Direction	Dip Angle	Search Distances (ft)			Min. # of Samples	Min. # of Drillholes
			Primary	Secondary	Tertiary		
Wilson Measured	225	30	50	29.4	17.6	3	2
Wilson Indicated	225	30	100	58.8	35.3	2	1
Wilson Inferred	225	30	150	88.2	52.9	2	1
Wheeler Measured	30	45	50	33.3	16.7	3	2
Wheeler Indicated	30	45	100	66.6	33.3	2	1
Wheeler Inferred	30	45	150	100	66.7	2	1

14.8.2 Cross Section Examples of the Block Model Rock Codes

One cross section each from Wilson and Wheeler were selected to show the details of the block model rock codes. Cross section 2470050E, which bisects the Wilson area, and cross section 1600N in the Wheeler area are presented as **Figure 14.14** and **Figure 14.15**, respectively. In the cross sections, rock codes are displayed on the left of each drillhole trace. The geologic boundaries used for the geologic modeling are also shown on the cross sections.

Figure 14.14: Geologic Model – Wilson Deposit

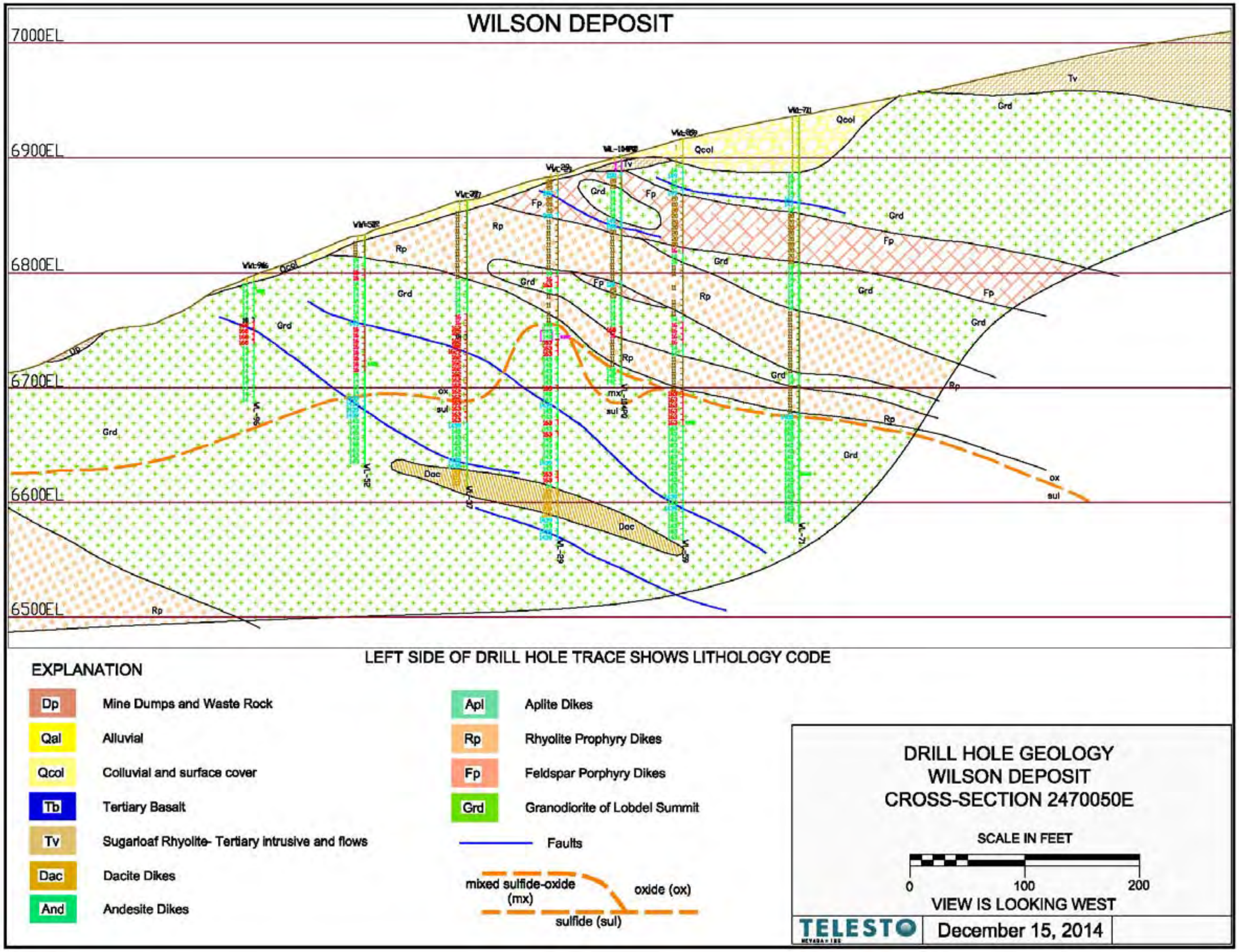
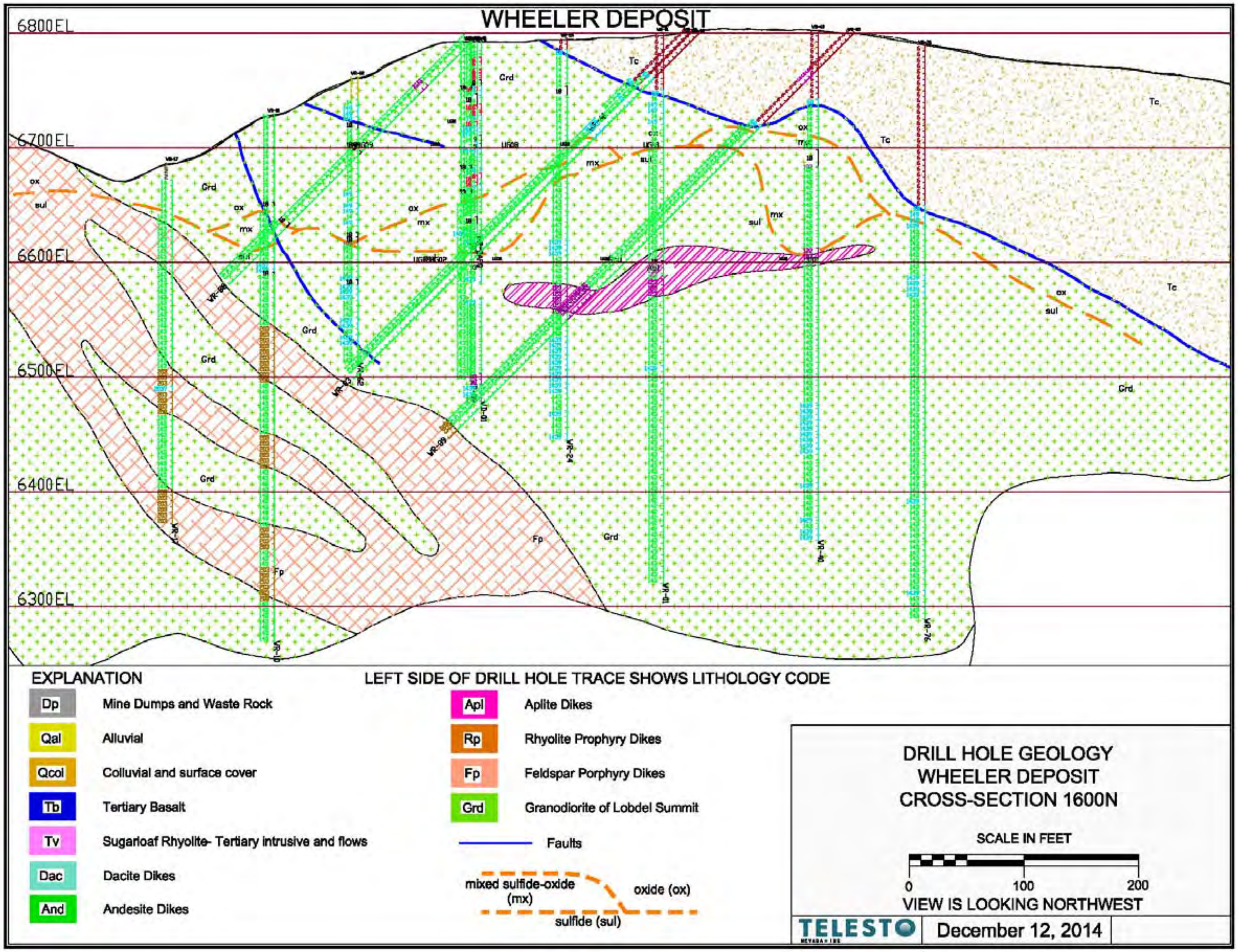


Figure 14.15: Geologic Model – Wheeler Deposit



14.8.3 Cutoff Grade

Cutoff grades used to estimate the in-situ resource and the resources within the designed pit shells are based on an analysis of gold price, mining costs and recoveries. Assumptions were made for items like strip ratio, mining costs, processing costs and recovery percentages. These assumptions were based on published or internally calculated rates for this and/or other similar mining operations in the general area and test work performed on Pine Grove samples. Each aspect of costs is discussed separately in the following sections with a summary below.

Table 14.15 summarizes the parameters for which the appropriate cutoff grade was determined, and on which the floating cone analysis was based

Table 14.15: Parameters Used in Pit Design and Cutoff Grade Determination

Parameter	Value	Comment
Estimated Yearly Mineralized Material Tonnage	1,000,000	Strip ratio 4:1
Gold Price	$1,425/oz	60% 3-year previous average / 40% 2-year projected forward – Aug. 23, 2011
Mining Cost	$2.33/ton ore $2.33/ton waste	Estimated from similar-sized Nevada gold mining operation
Crushing Costs	$4.83/ton ore	Based on costs developed for Pine Grove
Leaching Costs	$ 3.50/ton ore	Based on costs developed for Pine Grove
Processing Costs	$6.46/ton ore	Based on costs developed for Pine Grove
G & A	$3.68/ton ore	Based on costs developed for Pine Grove
Process Recovery	75% for all ore	Based on weighted average recovery for all test work

Mining Costs

Mining costs include $2.25 per ton for the mining contractor. Mining costs also include the Lincoln labor costs for mining staff, which are about 8 cents per ton mined.

Crusher Costs

Costs were estimated for crushing material to 80% passing $^3/_8$ inch. The material is then agglomerated and conveyed to a radial stacker. Crushing costs are estimated to be $4.83/ton.

Leaching Costs

Leaching costs were estimated $2.96 per ton of mineralized material. This includes an average sodium cyanide application rate of 2.5 lbs. per ton of mineralized material.

Process Reagent and CIC Plant Supply Costs

Processing costs were estimated to be $6.46 per ton of mineralized material. The cost for reagents and column cell operation are estimated to be $3.01/ton. .

General & Administrative (G & A) Costs

G & A costs are estimated based Mining Cost Service and other Telesto sources and are estimated to be $3.68/ton.

Recovery

The recent cyanide leach test work of five column and 45 bottle roll tests from the Wheeler and Wilson deposit provide the bulk of the metallurgical test data used, with a weighted average gold recovery value of 77%, if crushed to 80% passing $^3/_8$ inch and heap leached for 150-170 days. In this report, a gold recovery value of 75% was used for all mine planning. As a general rule, a feed requiring crushing to a nominal ¾ inch (19 mm) or finer will need agglomeration, even if clayey constituents are not present." (McClelland, 1988)

Resources are reported using a cutoff grade of 0.014 opt gold. A grade-tonnage curve for measured and indicated gold resources is presented in **Figure 14.16**. Cutoff grade vs. average grade above cutoff for gold measured and indicated resources is shown as **Figure 14.17**.

14.8.4 Grade Model Cross Sections

Figure 14.18 is a cross section showing the details of the grade model in the Wilson resource area. **Figure 14.19** shows block gold values in the Wheeler resource area. Gold grade in the blocks and in the drillhole intercepts are color coded according to the explanation on each cross section.

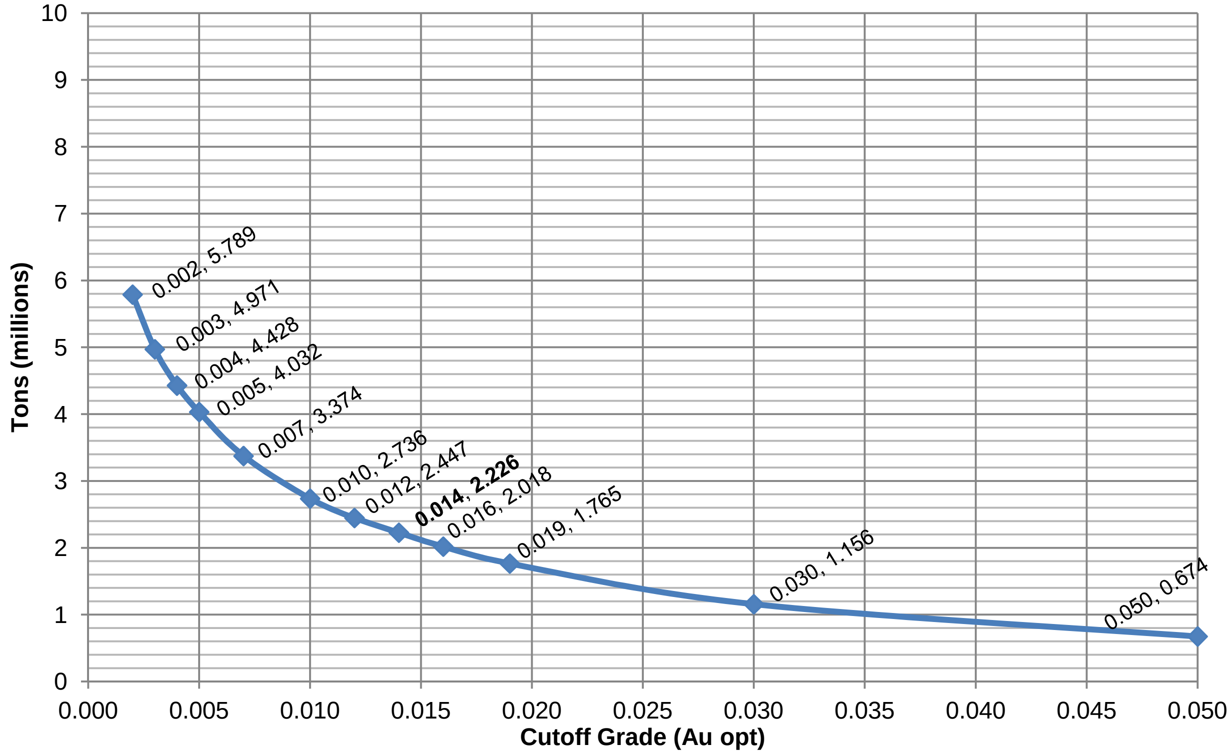


Figure 14.16: Grade-Tonnage Curve for Measured and Indicated Gold Resources

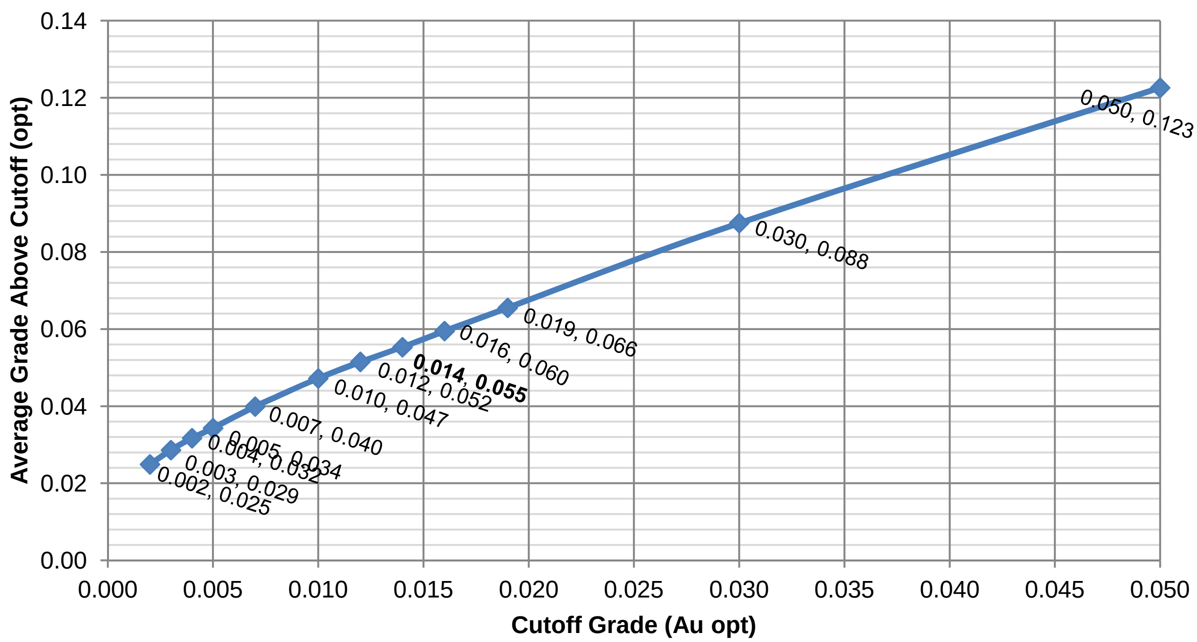


Figure 14.17: Cutoff Grade vs. Average Grade Above Cutoff for Measured and Indicated Gold Resources

Figure 14.18: Wilson Block Model Cross-Section

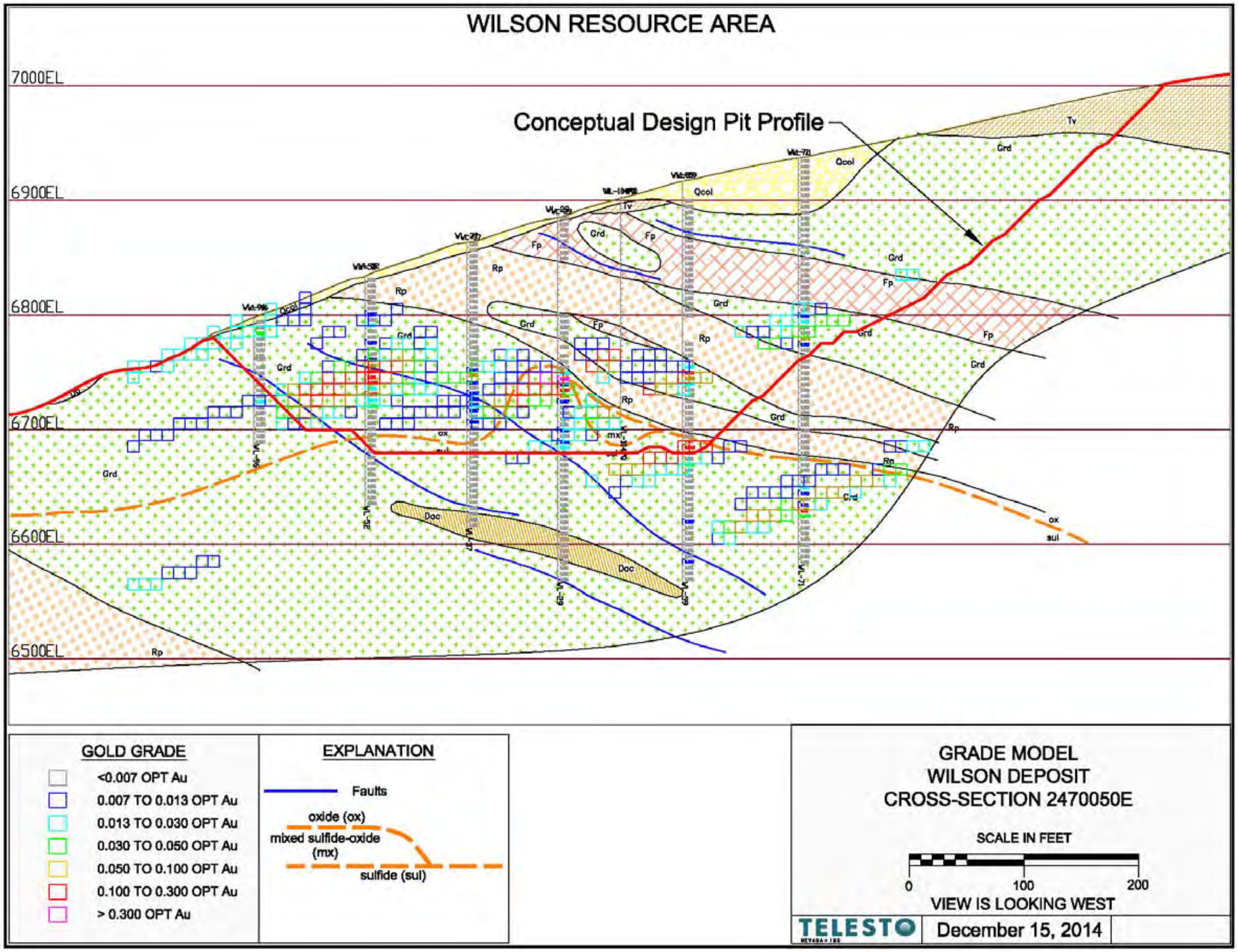
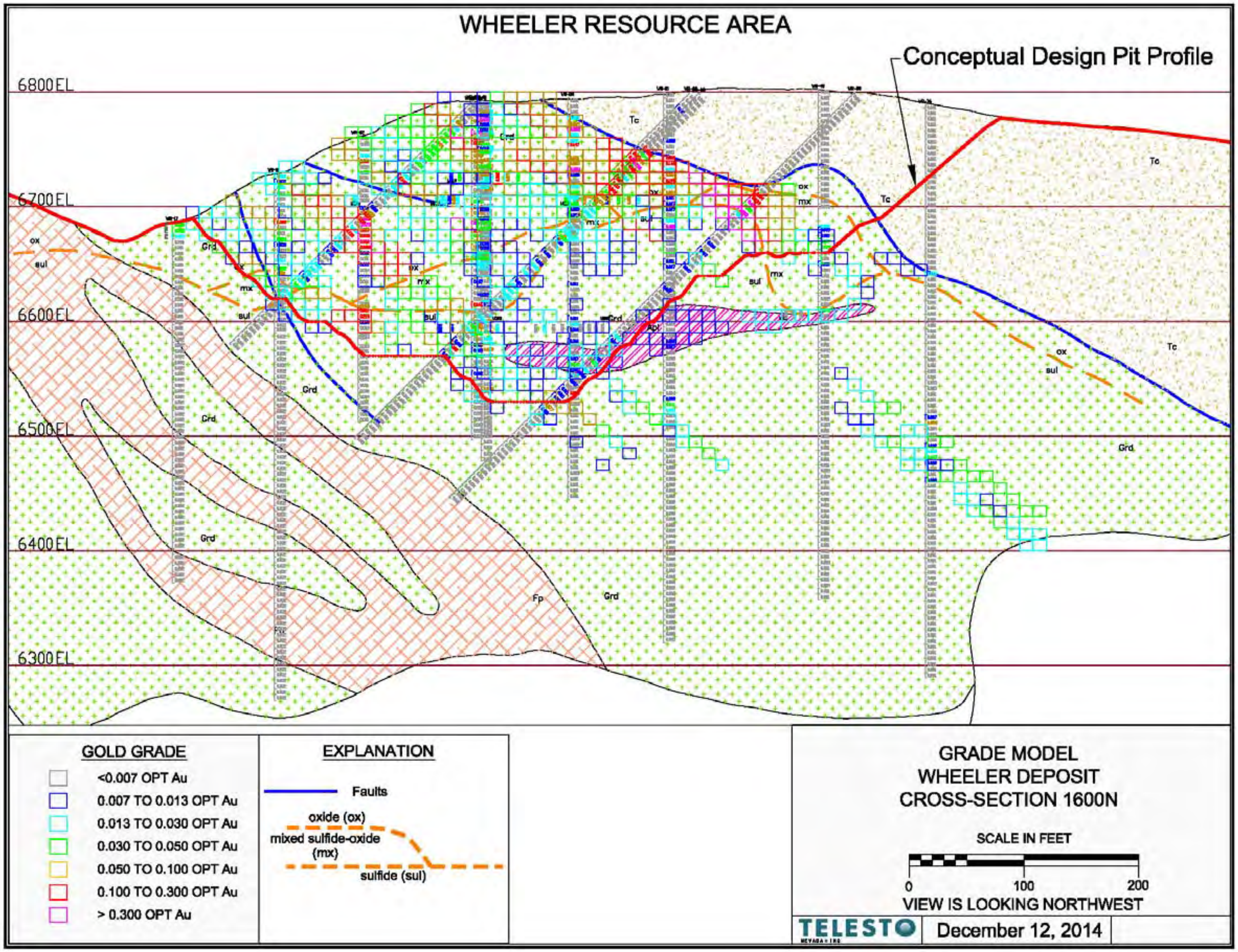


Figure 14.19: Wilson Block Model Cross-Section



14.9 Gold Resources

The resulting resources reported herein for Pine Grove were classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definitions. Resources are reported as measured, indicated and inferred.

In order to comply with the CIM definition of “reasonable prospects for economic extraction,” the following tables report only those blocks contained within a designed pit shell as Mineral Resources.

Telesto has reported the Pine Grove Resource at three different cutoffs. 0.019 opt Au is the calculated break-even cutoff. The break-even cutoff is the gold grade which, when processed, will generate just enough revenue to just cover all operating costs (mineralized material mining, mineralized material processing, and G&A). 0.014 opt Au is the calculated internal break-even cutoff. The internal break-even cutoff is the gold grade, which, when processed will generate just enough revenue to cover the mineralized material processing costs.

The third cutoff grade, 0.007 opt Au, is below the internal break-even cutoff. Telesto has found that using this lower cutoff will generate the best project NPV, when compared to using either of the break-even cutoffs. This can be explained by the fact that the production rate for mineralized material is primarily limited by the mining capacity. At the higher cutoffs, the amount of mineralized feed material coming from the mine cannot keep the crushing plant going at maximum capacity. In fact, at the higher cutoffs, there are short periods where the mine cannot produce any mineralized feed material.

By lowering the cutoff, the mine is able to produce an amount of mineralized material to feed the crushing plant which is closer to the maximum designed capacity. Since much of the plant operating cost is a fixed value, the cost of processing the added lower grade mineralized material is minimal, and the net result is an increase in yearly revenues. Therefore, the 0.007 cutoff case has been chosen as the preferred cutoff scenario in the examination of the potential economics. The fixed costs associated with the crusher are certainly an area requiring further analysis in a Pre-Feasibility Study. This might well impact future economic cutoff grades.

14.10 Total Combined Gold Resources

Table 14.16 shows the total combined measured and indicated resources at Pine Grove; **Table 14.17** shows the total combined inferred resource that is within the Pine Grove designed pits.

Table 14.16: Total Measured and Indicated Gold Resources at Pine Grove

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Measured	2,356	2,137	0.007	0.240	0.041	1.42	97,300	3,026,300
Indicated	1,017	923	0.007	0.240	0.037	1.25	37,200	1,155,600
Measured + Indicated	3,373	3,060	0.007	0.240	0.040	1.37	134,500	4,182,000

At 0.014 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Measured	1,580	1,433	0.014	0.480	0.057	1.95	89,700	2,788,500
Indicated	647	587	0.014	0.480	0.052	1.78	33,600	1,044,900
Measured + Indicated	2,227	2,020	0.014	0.480	0.055	1.90	123,300	3,833,400

At 0.019 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Measured	1,256	1,139	0.019	0.651	0.067	2.30	84,200	2,619,800
Indicated	509	462	0.019	0.651	0.062	2.11	31,300	974,700
Measured + Indicated	1,765	1,601	0.019	0.651	0.065	2.24	115,600	3,594,600

Notes:
- Mineral Resources are reported using a long term gold price of $1425/oz.
- Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
- Cutoff Grade 0.014 opt Au. This is considered the “best case” cutoff.
- Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
- Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.17: Total Inferred Gold Resources at Pine Grove

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	160	145	0.007	0.240	0.041	1.41	6,600	204,100

At 0.014 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	88	80	0.014	0.480	0.067	2.29	5,900	183,000

At 0.019 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	64	58	0.019	0.651	0.086	2.94	5,500	170,500

Notes:
- Mineral Resources are reported using a long term gold price of $1425/oz.
- Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
- Cutoff Grade 0.014 opt Au
- Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
- Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

14.11 Wilson Gold Resources

Table 14.18 shows the estimated measured and indicated resource at Wilson; Table **14.19** shows the estimated inferred resource at Wilson.

Table 14.18: Wilson Measured and Indicated Gold Resources

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade		Ounces	Grams
					Gold (opt)	Gold (g/t)		
Measured	894	811	0.007	0.240	0.035	1.20	31,400	975,800
Indicated	620	562	0.007	0.240	0.035	1.20	21,600	672,900
Measured + Indicated	1,514	1,373	0.007	0.240	0.035	1.20	53,000	1,648,800

At 0.014 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade		Ounces	Grams
					Gold (opt)	Gold (g/t)		
Measured	570	517	0.014	0.480	0.049	1.69	28,200	875,700
Indicated	389	353	0.014	0.480	0.050	1.71	19,400	603,700
Measured + Indicated	959	870	0.014	0.480	0.050	1.70	47,600	1,479,300

At 0.019 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade		Ounces	Grams
					Gold (opt)	Gold (g/t)		
Measured	436	396	0.019	0.651	0.060	2.04	25,900	806,800
Indicated	301	273	0.019	0.651	0.060	2.05	18,000	558,800
Measured + Indicated	737	669	0.019	0.651	0.060	2.04	43,900	1,365,600

Notes:
- Mineral Resources are reported using a long term gold price of $1425/oz.
- Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
- Cutoff Grade 0.014 opt Au. However, the “best case” economic potential of the project was examined using a cutoff grade of 0.007 opt Au due to the fixed costs associated with the preferred crusher throughput and timing of material to feed the crusher.
- Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
- Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.19: Wilson Inferred Gold Resources

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	154	140	0.007	0.240	0.042	1.43	6,400	200,200

At 0.014 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	85	77	0.014	0.480	0.068	2.33	5,800	180,000

At 0.019 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	62	56	0.019	0.651	0.087	2.99	5,400	167,900

Notes:
- Mineral Resources are reported using a long term gold price of $1425/oz.
- Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
- Cutoff Grade 0.014 opt Au
- Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
- Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

14.12 Wheeler Gold Resources

Table 14.20 shows the estimated measured and indicated resource at Wheeler; **Table 14.21** shows the estimated inferred resource at Wheeler.

Table 14.20: Wheeler Measured and Indicated Gold Resources

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade		Ounces	Grams
					Gold (opt)	Gold (g/t)		
Measured	1,462	1,326	0.007	0.240	0.045	1.55	65,900	2,050,500
Indicated	397	360	0.007	0.240	0.039	1.34	15,500	482,700
Measured + Indicated	1,859	1,686	0.007	0.240	0.044	1.50	81,500	2,533,200
At 0.014 opt Au cutoff	**Tons (000s)**	**Tonnes (000s)**	**Gold**					
			Cutoff Grade (opt)	**Cutoff Grade (g/t)**	**Average Grade**		**Ounces**	**Grams**
					Gold (opt)	**Gold (g/t)**		
Measured	1,010	916	0.014	0.480	0.061	2.09	61,500	1,912,800
Indicated	258	234	0.014	0.480	0.055	1.89	14,200	441,300
Measured + Indicated	1,268	1,150	0.014	0.480	0.060	2.05	75,700	2,354,100
At 0.019 opt Au cutoff	**Tons (000s)**	**Tonnes (000s)**	**Gold**					
			Cutoff Grade (opt)	**Cutoff Grade (g/t)**	**Average Grade**		**Ounces**	**Grams**
					Gold (opt)	**Gold (g/t)**		
Measured	820	744	0.019	0.651	0.071	2.44	58,300	1,813,100
Indicated	208	189	0.019	0.651	0.064	2.20	13,400	415,900
Measured + Indicated	1,028	933	0.019	0.651	0.070	2.39	71,700	2,229,000

Notes:
• Mineral Resources are reported using a long term gold price of $1425/oz.
• Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
• Cutoff Grade 0.014 opt Au However, the "best case" economic potential of the project was examined using a cutoff grade of 0.007 opt Au due to the fixed costs associated with the preferred crusher throughput and timing of material to feed the crusher.
• Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
• Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.21: Wheeler Inferred Gold Resources

At 0.007 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	6	5	0.007	0.240	0.021	0.72	100	3,900

At 0.014 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	3	3	0.014	0.480	0.032	1.09	100	3,000

At 0.019 opt Au cutoff	Tons (000s)	Tonnes (000s)	Gold					
			Cutoff Grade (opt)	Cutoff Grade (g/t)	Average Grade Gold (opt)	Average Grade Gold (g/t)	Ounces	Grams
Inferred	2	2	0.019	0.651	0.041	1.41	100	2,600

Notes:
- Mineral Resources are reported using a long term gold price of $1425/oz.
- Rounding of tons and contained gold results in apparent differences in totals and are in accordance with reporting guidelines.
- Cutoff Grade 0.014 opt Au
- Contained metal estimates remain subject to factors such as mining dilution and process recovery losses.
- Mineral Resources that are not mineral reserves do not have demonstrated economic viability.

15.0 MINERAL RESERVE ESTIMATES

There are no mineral reserves at Pine Grove.

16.0 MINING METHODS

The term "ore" is used in this section of the PEA to describe the conceptual mining methods proposed by this PEA in commonly used mining terms. The term "ore" generally implies that sufficient technical feasibility and economic viability studies have been completed to classify the material as mineral reserve. A Qualified Person has not done sufficient work to classify the mineral resource at Pine Grove as current mineral reserve and the issuer is not treating the mineral resource as mineral reserve.

The reader is reminded that the PEA is based on the Project resource model which consists of material in Measured, Indicated and Inferred classifications. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability.

16.1 Contract Mining

Lincoln will be soliciting bids from firms capable of providing contract mining services for the Pine Grove in the event that more detailed economic studies have been conducted and a decision to develop the property into a mine is made. Final contractor selection will be made once all permits are received.

The mine work schedule will be set by the contractor to meet Lincoln's production goals and is flexible. Typically, one 10-hour shift per day, 5 days per week would be used. The mining operation will be idle on weekends allowing for planned equipment maintenance activities.

The mine contractor will do both the preproduction work and site earthworks prior to beginning actual production.

16.2 Mining Schedule

A conceptual mining schedule based on an annual production goal of 1.0 million tons of mineralized material to the crusher. There is some legacy dump material from the old underground workings on the property that may be included in future schedules but lacks information at this time. The life of the project spans seven years including preproduction. Details regarding the assumptions used to generate the mining schedule are given in **Table 16.1**.

Table 16.1: Heap Leach Pad Design Details

Parameter	Unit	Comment
Leach tons per year	~1,000,000 tons	
Mine life	4 years	
Leach life	5 years	
Leach cycle	170 days	
Initial leach cycle	100 days	70% recovery at 100 days
% water lost per year	10%	Loss to evaporation
Solution to ore ratio	2.5	Tons of water per tons of ore, life-of-heap
Bench height	20 ft	
Width of each cell	851 ft	
Length of each cell	1,370 ft	
Gallons per minute	~900 gpm	Barren flow
Angle of repose	38.0°	
Liner length	1,407 ft	
Liner width	887 ft	
Total liner area	1,248,450 ft^2	
Lift toe to crest	25.6 ft	This measurement is a horizontal setback
Number of lifts	4	

A mining schedule was generated by Telesto based on resources within the conceptual designed pit phases using the following parameters and guidelines:

- Contract mining operations, 5 days per week, one shift per day;
- Crushing operations 5 days per week, two shifts per day; one weekend maintenance shift
- Average total annual mineralized material production of approximately 1.0 million tons.

Hydraulic excavators and rubber-tired front-end loaders were chosen as primary loading units. The loading units were matched to the contractor specified 100-ton haul trucks. This equipment is a good match for the size of the conceptual pits. Initial pit development may be performed using same equipment fleet as specified for production mining.

In general, backfilling of the Wheeler pit is considered economically and environmentally appropriate. Since the Wheeler Pit would conceptually be mined first, it would probably be backfilled with waste from the Wilson pit. As mining progresses, a minor quantity of fill material may be required on a bench by bench basis to provide temporary ramps in areas with difficult access. Access ramps to the upper levels of the pits would mainly be internal to the pits and would be mined out as the pit progresses downward. The detailed conceptual mine schedule for the currently identified mineral resources within a conceptually designed pit is summarized by year in **Table 16.2**.

A set of conceptual annual mining plans was developed based on the desired production schedule. A conceptual general facility arrangement for the Pine Grove Project showing the pre-mining topography, ultimate pit configurations, haul roads and processing facilities is presented as **Figure 16.1**. **Figure 16.2** and **Figure 16.3** show the conceptual design pits generated for the Wheeler and Wilson Deposits, respectively.

Figure 16.1: Conceptual General Facility Arrangement

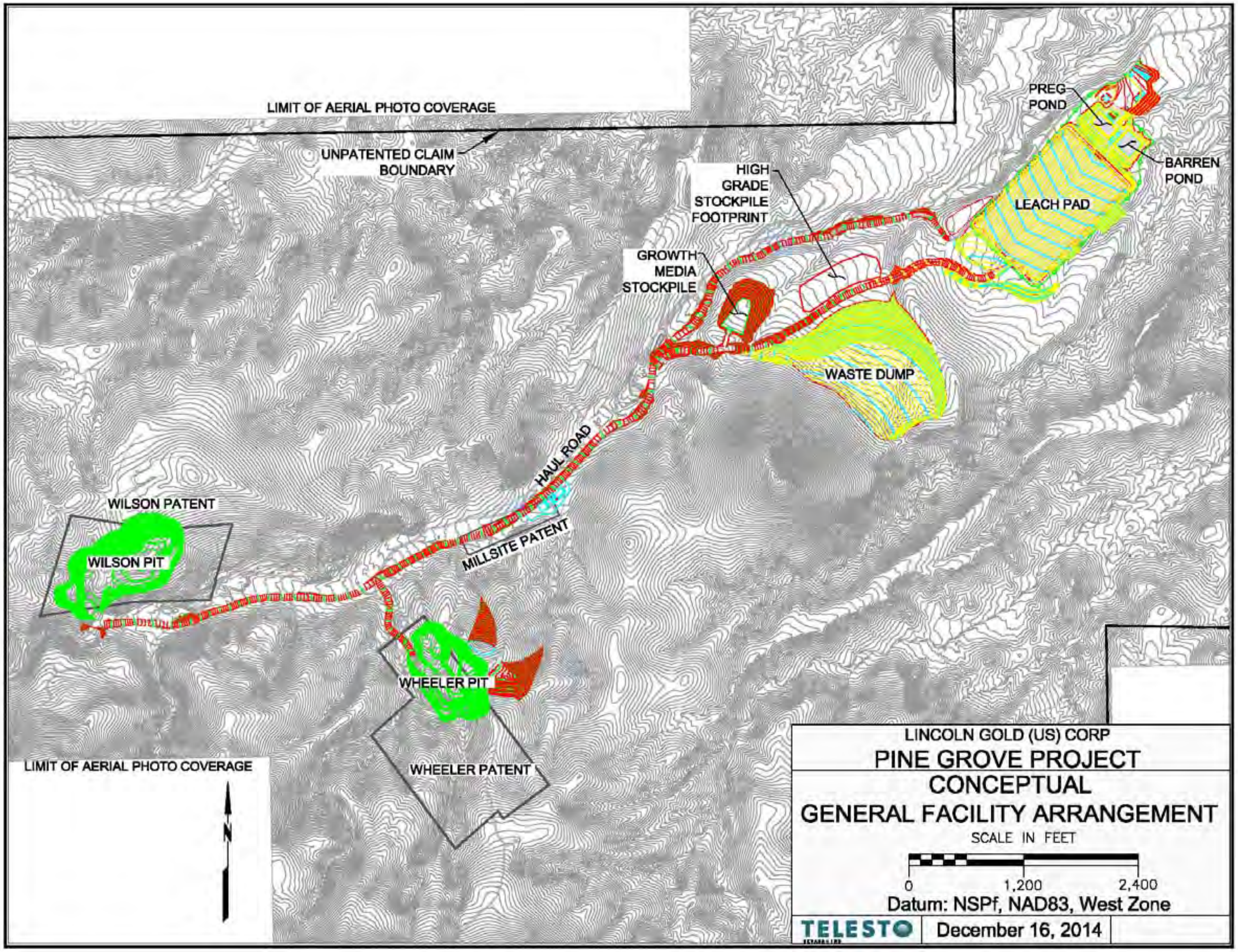


Figure 16.2: Conceptual Pit Design– Wheeler Deposit

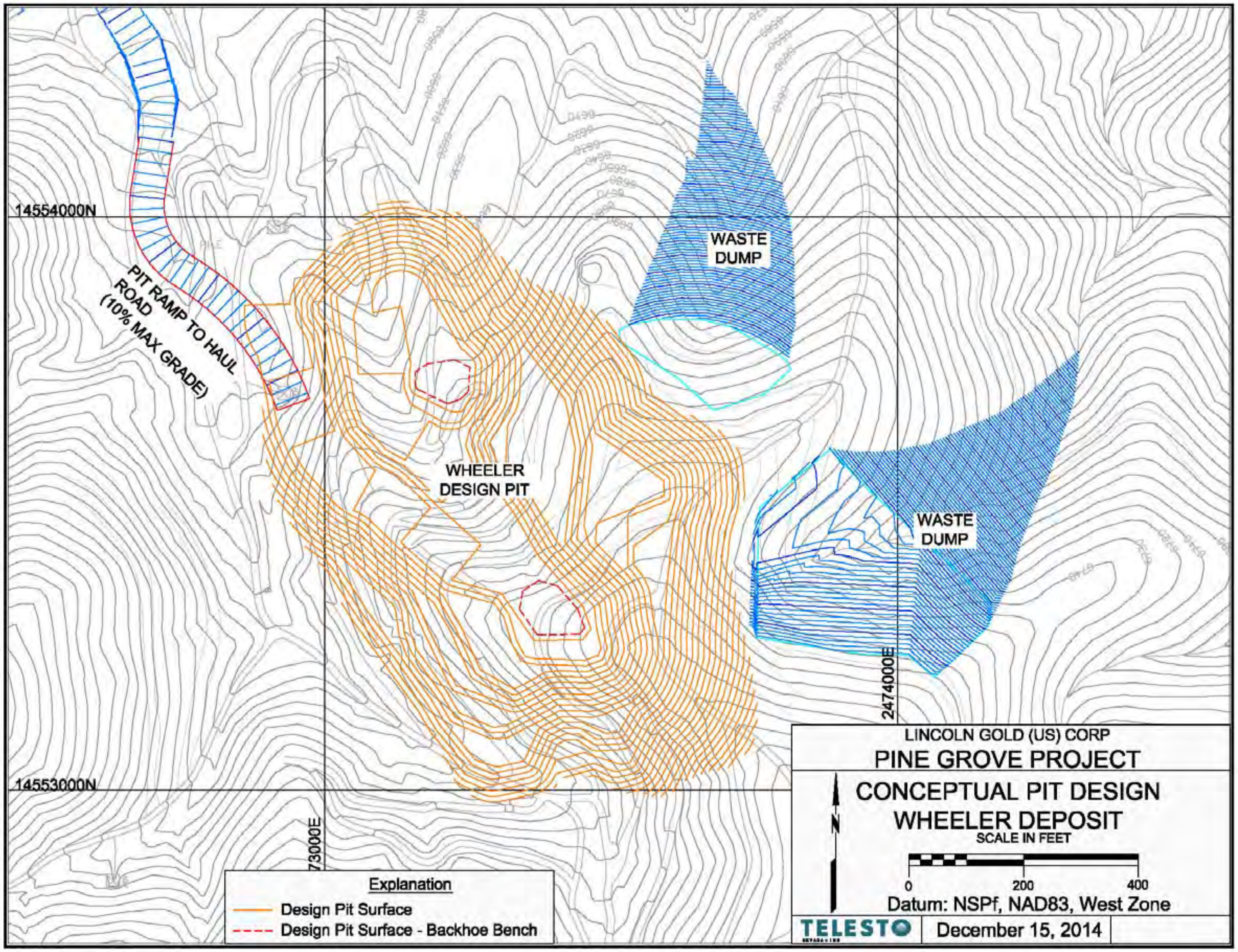


Figure 16.3: Conceptual Pit Design – Wilson Deposit

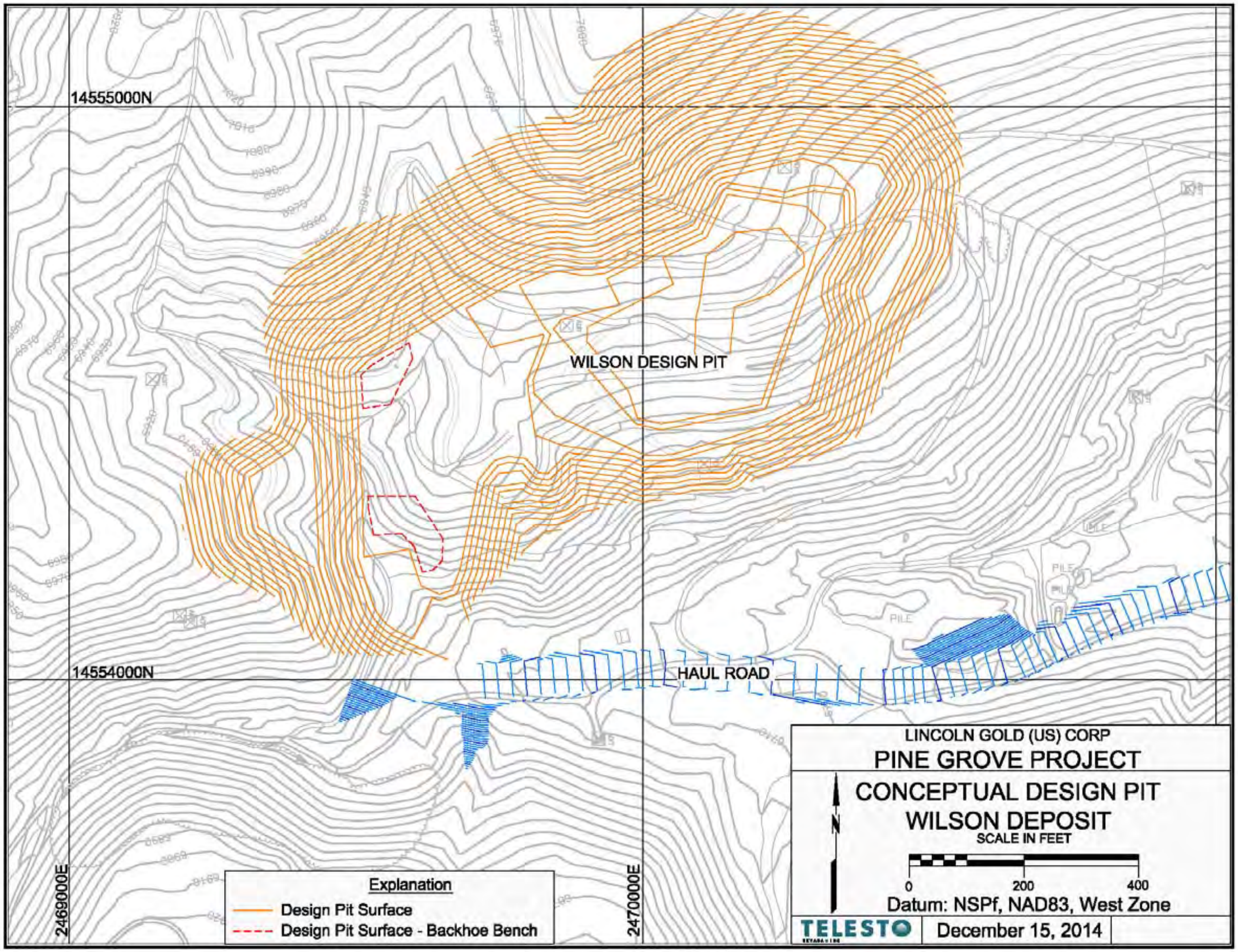


Table 16.2 shows the schedule for the currently identified mining resources. Pre-production would begin when all permits are issued and last a year. Leach pad construction would occur simultaneously. The mining contractor delivers mineralized material to a crusher stockpile throughout the mine life and during pre-production, thus the variances in "Mineralized Material to Crusher" and "Mineralized Material Mined". Lincoln personnel operate the crusher and feed mineralized material from the stockpile.

Pre-production includes:

- the development of the main haul roads to the pits, crusher and waste dumps
- development of haul roads within both the Wheeler and Wilson Pits to access the upper waste benches and waste dumps
- stripping of waste overburden in the Wheeler Pit on the upper benches until enough mineralized material is available to assure a continuous feed to the crusher stockpile
- stripping of some waste in the Wilson Pit to reduce the future strip ratio so that a relatively continuous mineralized material supply is available to the crusher.

The Wheeler Pit would conceptually be mined out at the beginning of Year 3. The Wilson Pit would be completed during the third quarter of Year 4. Crusher feed is assumed to be 80,000 tons per month. Contractor production will be about 275,000 total tons per month for the first twelve months of mining and then will increase to approximately 300,000 total tons per month through the remainder of the mine life.

Table 16.2: Conceptual Pine Grove Mine Schedule

New Ore	Year						Project Total
	Pre-production	1	2	3	4	5	
Combined Production							
Ore to Crusher (kt)		960	960	960	494	0	3,374
Grade (opt)		0.045	0.036	0.038	0.040	0.000	0.040
Au (oz)	–	43,418	34,355	36,890	19,857	0	134,520
Ore Mined(kt)	130	1,133	834	1,062	215	0	3,374
Waste (kt)	2,992	2,314	2,837	2,119	213	0	10,475
Total (kt)	3,122	3,447	3,671	3,181	428	0	13,849
Wheeler							
Grade (opt)	0.047	0.047	0.037	0.094	0.000	0.000	0.044
Au (oz)	2,316	50,452	26,403	2,357	0	0	81,527
Ore Mined(kt)	50	1,079	705	25	0	0	1,860
Waste (kt)	451	50,452	1,744	28	0	0	3,926
Wilson							
Grade (opt)	0.026	0.023	0.018	0.039	0.034	0.000	0.035
Au (oz)	2,049	1,265	2,364	41,331	7,340	0	53,010
Ore Mined(kt)	80	54	129	1,061	215	0	1,514
Waste (kt)	2,541	614	1,094	2,225	213	0	6,557

Note: Rounding of tons in monthly sequence creates some variances with total resource numbers.

17.0 RECOVERY METHODS

The reader is reminded that the PEA is based on the Project resource model which consists of material in Measured, Indicated and Inferred classifications. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability.

17.1 Introduction

The facilities for the Pine Grove Project will be designed to process up to 1.5 million tons of mineralized material to the crusher annually, with an average of 1.0 million tons scheduled in the mine plan. The sources of mineralized material consist of approximately 1.9 million tons from the Wheeler area, 1.6 million tons from the Wilson area. Potentially there is additional mineralized material available from the old underground waste stockpiles, waste dumps and tailings but there is not enough information available to include them at this time. Lincoln would ship loaded carbon from the column cells to an offsite refinery. The major operations required to process the material include:

- Crushing, screening, agglomeration, and heap stacking
- Heap leaching
- Carbon Adsorption

The material will be leached with a dilute sodium cyanide solution on a heap leach pad. Gold will be recovered from the collected pregnant leach solution in activated carbon columns.

Figure 17.1 shows a conceptual schematic flow diagram of the process facilities.

17.2 Site Layout Considerations

The carbon columns are located on the site layout adjacent to the pregnant solution pond and are down gradient from the heap leach pad as shown on **Figure 17.2**.

17.3 Process Description

17.3.1 Design Criteria

Key design criteria are estimated to be:

- Operating Schedule 365 days/year
- Heap Stacking Schedule 80,000 tons/month or 3,300 tons/day
- Gold Production 27,000 ounces/year
- Solution Flow rate 350 to 650 gpm

Figure 17.1: Process Flow Diagram

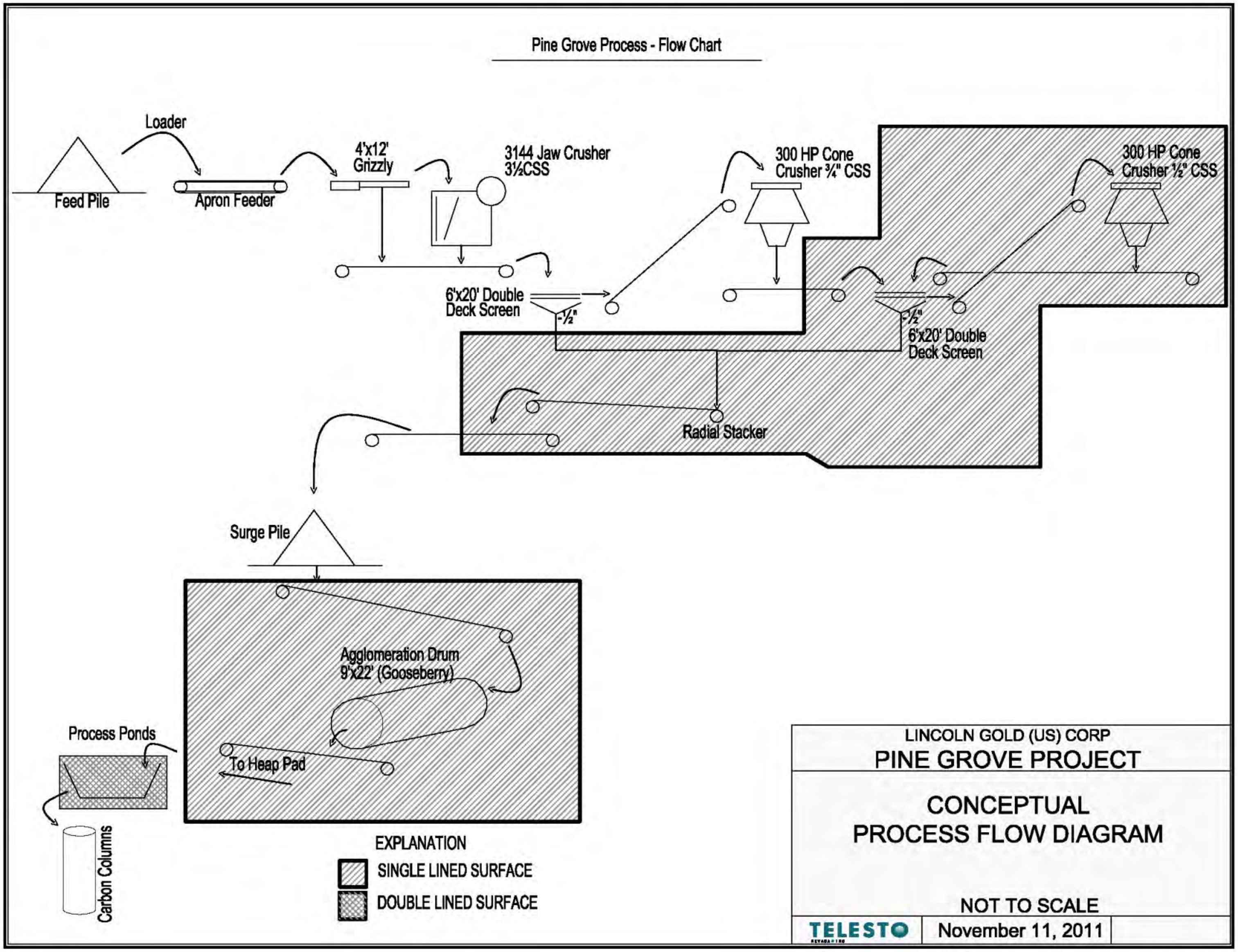
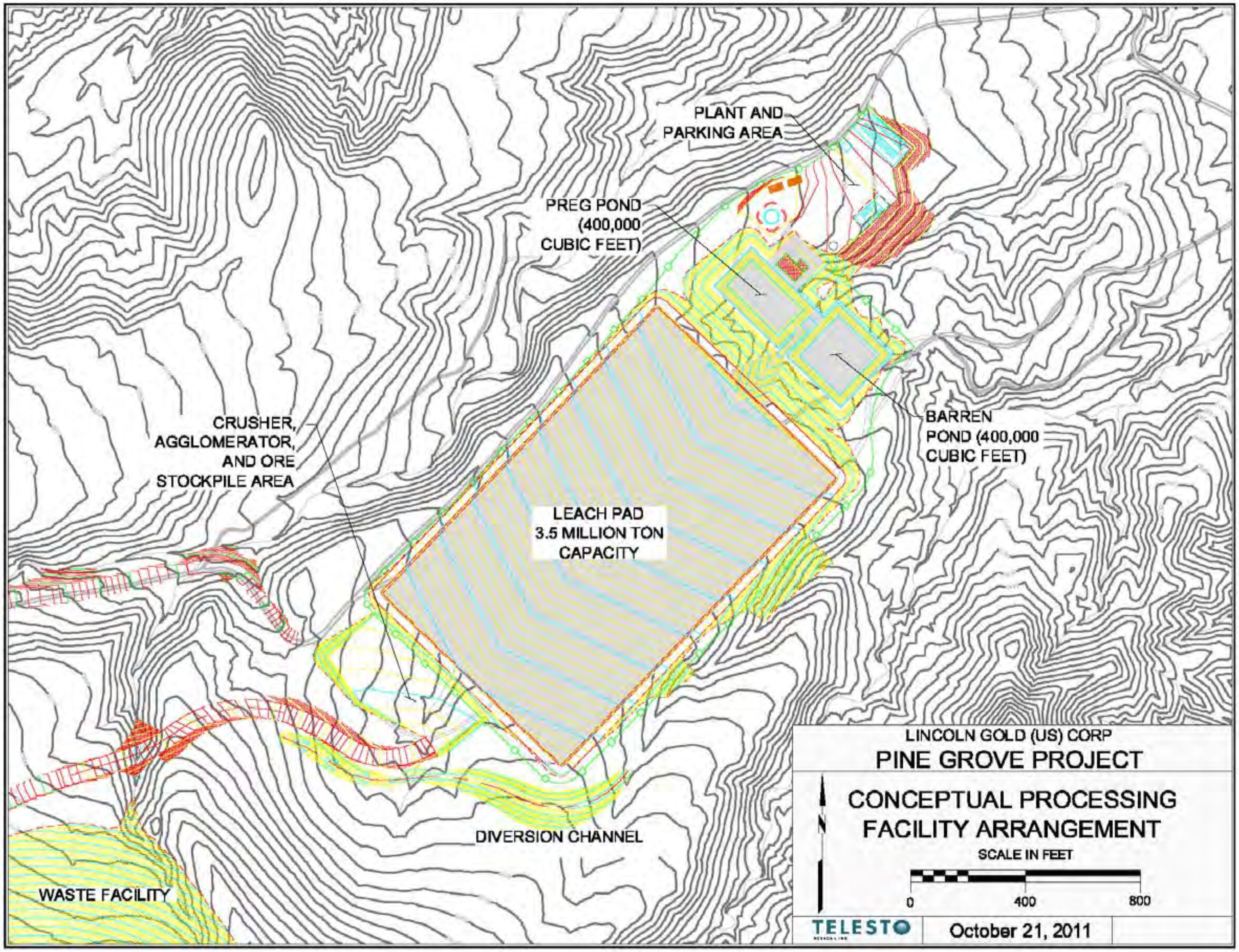


Figure 17.2: Conceptual Processing Facility Arrangement



17.4 Crushing, Screening, and Agglomeration

Mineralized material will be crushed in a primary crusher followed by screening and secondary crushing in an open circuit to achieve a size of 80 percent less than $-^{3}/_{8}$ inch.

After crushing, the mineralized material will normally be transferred onto a conveyor for agglomeration in a rotary drum and then conveyed for stacking onto the heap leach pad. A bypass stockpile can be used if the crushing and screening plant is not operating. A front end loader will transfer the material from the stockpile onto the overland conveyor when needed.

A series of overland and grasshopper conveyors transport the material to the heap leach pad for stacking. The agglomeration step will use cement or lime and a dilute cyanide solution. Mineralized material will be stacked in 20-foot lifts on the heap leach pad.

17.4.1 Heap Leaching

Mineralized material will be leached with a dilute cyanide solution on the lined heap leach pad. Barren cyanide solution from the column cells is pumped over the heap material with drip tubes fed by pipes from a barren solution distribution network. After the solution percolates through the material, the pregnant leach solution is collected in a series of pipes and ditches that drain to the pregnant solution tank.

An intermediate, or recycle, solution will also be applied to the fresh mineralized material. This solution, which contains dilute cyanide and lower grade gold, will be recycled back to a part of the heap that has already reached the end of its initial leach cycle. This recycle solution will continue to leach residual gold values from the material and increase the concentration of gold in the effluent. The leach application rate averages 0.004 gpm/ft^2, while the total flow (barren and recycle) to all areas of the pad averages 400 to 500 gpm. All solution flows and values are to be monitored for metallurgical accounting purposes.

17.4.2 CIC Circuit

The pregnant solution is pumped to a CIC circuit. In this circuit, the pregnant solution is contacted with activated carbon to recover the gold by adsorption. The CIC plant consists of one train of five columns. Carbon is advanced in a direction counter current to solution flow.

Solution that discharges from the last column overflows to the barren solution tank. Liquid sodium cyanide, fresh water, and anti-scalent are added in this tank, as required, to make up a fresh leach solution, which is pumped to the leach pad for additional distribution on the heap material.

The carbon will probably be loaded with gold until it is holding approximately 100 ounces. It then will be shipped off-site for processing by a company such as Just Refiners of Reno, Nevada. Just Refiners proposed processing loaded carbon at a cost of $1,000 per ton of loaded carbon and 2% of the gold.

17.4.3 Reagents

The reagents required at the Pine Grove process facility include:

- Sodium cyanide (NaCN)
- Activated carbon
- Anti-scalent
- Lime
- Cement

Anti-scalent and liquid NaCN will be shipped to the site in recyclable containers. Appropriate storage and containment facilities will be provided for all of the reagents. Dry lime and cement will be delivered in bulk quantities by trucks equipped with pneumatic delivery pumps, off-loaded, and stored in silos. The activated carbon will arrive in bulk bags. An attrition system is included in the design to prepare activated carbon.

17.5 Recoverability

When the quantity and size distribution of free gold is determined in the deposit, a flow sheet can be developed for enhanced recovery of the higher grade material. Currently there is only one metallurgical test conducted and communicated by Lincoln Gold US Corp. to concentrate the precious metals in the material for further processing. In 1983, a sample of "black sand concentrate" designated W-2 was ground to a P_{82} of 200 mesh and leached with cyanide with gold recovery of 99.2% and 99.5% of silver with a fire assay head of 41.92 gold and 3.80 silver oz/ton (Clem, 1983). "No description of the origin of the concentrate sample was provided. Consequently, interpretation of those results is difficult." (McPartland, 2009)

No other metallurgical data on content and free gold quantity as per size was provided, except 28 screen fire assays. "Head screen analysis results showed that the Wilson composite 1¼" and $^3/_8$" feeds contained 0.069 and 0.062 oz Au/ton ore, respectively, and that the Wheeler 1¼" and $^3/_8$" feeds contained 0.054 and 0.043 oz Au/ton ore. The Wheeler Surface composite ($^3/_8$" feed) contained 0.109 oz Au/ton ore. Thus, contained gold values were not evenly distributed throughout the various size fractions. Size fraction assays tended to be "spotty", possibly indicating the presence of free-milling, particulate gold values. Further testing would be required to confirm this observation." (McPartland, 2011)

The calculated head analysis from the Wheeler Composite (McPartland, 2011) Bottle Roll Test was 0.059 oz Au/ton which was significantly different than the three direct assays of 0.064, 0.066 and 0.070 oz Au/ton and the $^3/_8$" calculated column head of 0.048 and the 1¼" calculate column head of 0.047 oz Au/ton.

The calculated head analysis from the Wilson Composite (McPartland, 2011) Bottle Roll Test was 0.104 oz Au/ton which was significantly higher than the three direct assays of 0.062, 0.064 and 0.084 oz Au/ton and the $^3/_8$" calculated column head of 0.064 and the 1¼" calculate column head of 0.064 oz Au/ton.

Lincoln has not provided any additional metallurgical testwork, or reports on concentrating the potential free gold fraction indicated by spotty assays determined in the reviewed metallurgical testwork. Without concentration metallurgical testing on a representative sample, with good metallurgical accounting, no milling or concentration process data, or projected gold recovery values can be designed or inferred upon at this time. The potential exists for enhancing a gold recovery system by the use of gravity and/or flotation concentration on a ground high grade fraction of the properties material to collect the potential larger free gold fraction.

Generally the Wheeler mineralized material was readily amenable to simulated heap leaching when crushed and agglomerated to a P_{80} of 1¼” and $^{3}/_{8}$” feed sizes. In addition, the Wheeler deposit contains sufficient copper that will be co-extracted via cyanide heap leaching, thus resulting in high cyanide consumption. The Wheeler deposit does not appear to have significant concentrations of Hg, but this metal needs to be tested and tracked in future metallurgical testwork.

17.5.1 Wheeler Surface Composite

- Gold recovery rate was moderate. A longer leaching cycle would not significantly improve precious metal recoveries.
- Cyanide consumption was high. Column cyanide consumption for the $^{3}/_{8}$" feed was 6.60 lb NaCN/ton mineralized material.
- The lime added to the mineralized material charge before leaching, 3.6 lb lime per ton mineralized material, was sufficient for maintaining protective alkalinity throughout the leaching cycle.
- The copper extracted during leaching was 0.2 lb Cu/ton ore, which was equivalent to 17% of the total contained copper. Column test pregnant solution copper concentrations increased to as high as approximately 300 ppm (mg/L) during leaching (McPartland, 2011).

Generally the Wheeler Surface mineralized material was amenable to simulated heap leaching when crushed and agglomerated to a P_{80} of $^{3}/_{8}$” feed size. In addition, the Wheeler Surface deposit contains sufficient copper that will be co-extracted via cyanide heap leaching, thus resulting in high cyanide consumption. The Wheeler Surface deposit was not tested for Hg, so this metal needs to be tested for and tracked in future metallurgical testwork.

17.5.2 Wilson Column Composite

- Gold recovery rates were moderate and increased with decreasing feed size. A longer leaching cycle would not significantly improve precious metal recoveries. The Wilson deposit composite was not amenable to simulated heap leaching treatment at the 80% -1¼" feed size. Individual screen fraction gold extraction from the 1¼” column cyanide leach tests showed less than 0.01% recovery for rock sizes greater than 1”, 54.3% gold extraction for rock 0.75 to 1”, 3.6% gold extraction for the rock size 0.75 to 0.5”, 82.7% gold extraction for 0.50 to 0.375” size, and a weighted average of 76.6% gold extraction for minus 3/8” material which represents 25.5% of the column tail mass. Thus crushing finer does achieve a higher gold recovery for the Wilson deposit composite via cyanide column leaching.

- Cyanide consumptions were high for both column feed sizes. Cyanide consumptions for the 1¼" and 3/8" feeds were 4.40 and 5.95 lb NaCN/ton mineralized material, respectively.
- The lime added to the ore charges before leaching, 4.6 lb lime per ton mineralized material, was sufficient for maintaining protective alkalinity throughout the leaching cycle.
- Copper extractions obtained at both feed sizes were 0.2 lb Cu/ton ore (100 ppm), which was equivalent to 15% of the total contained copper. Copper concentrations in the column test pregnant solutions increased to as much as approximately 400 ppm (mg/L) during leaching. These concentrations are considered to be high enough to complicate down-stream solution recovery and refining processes solution recovery in a commercial heap leach circuit (McPartland, 2011).
- The gold extraction column metallurgical test work on the 80% -1¼" feed size (**Figure 13.4**) achieved a recovery of 37.5 % after 140 days, but the leach curve was still climbing, thus it is inferred that the extraction could improve with additional leaching time with sufficient reagents. This metallurgical inference should be quantified with additional, longer column leach time prior to an economic analysis of a higher heap leach recovery.

Generally the Wilson mineralized material represents metallurgical challenges of low gold recovery, unless crushed and agglomerated to a P_{80} of 3/8" or finer. In addition, the Wilson deposit contains sufficient copper that will be co-extracted via cyanide heap leaching, thus resulting in high cyanide consumption, which could limit the gold extraction in the micropores of the leaching material. Plus, the Wilson deposit has Hg with the potential to be co-extracted with the precious metals via cyanide leaching, and this Hg extraction will need to be addressed in the process flow sheet and resulting plant design and capitalization. Hg co-extraction with precious metals measurements will be required to determine the effect of cyanide leaching on Hg via heap leaching or tank leaching prior to finalizing a flow sheet for design and construction.

The recovery values utilized for this study were based on results from column leach tests conducted or reviewed by McClelland Laboratories as shown in **Table 17.1**. The head grades as shown are based on reported average fire assays and the actual head grade (calculated head) from the test work. There is some scatter in the data which makes the evaluation of the data difficult, and may show the presence of free gold. Screen fire assays on nine drill hole samples of a grade less than 0.04 opt gold showed 29.5% of the gold reported to not pass a 100 µm screen. No viable metallurgical testwork is currently available on other dump or tailings material. No recovery discounts for column leach testing was integrated into the recovery table, but reflects actual reported metallurgical test data.

Table 17.1: Metallurgical Cyanide Leach Testing

McPartland, 2009-2011 Data			Wheeler Composite	Wheeler Surface	Wilson Composite	Wilson Surface	Test Method
Head Grade: Au oz/ton			0.056	0.094	0.072	0.058	Avg. Fire Assay
Au oz/ton			0.0475	0.08	0.064	0.0635	Avg. Calculated Head
P_{80} Size:	mm	inch					
10	2.0	0.079	78.0%		76.9%		Bottle Roll-CN Test
$^3/_8$”	9.5	0.375	87.5%	85.0%	62.5%		Column Test
½”*	12.5	0.500				78.0%	Column Test
1¼”	31.5	1.250	74.5%		37.5%		Column Test

* - This sample is a nominal ½ inch not a P_{80} of ½”, Clem, 1983.

The data displayed in **Table 17.1** assumes that the composites from the 140 drill hole composites formed by McClelland Lab with direction from Lincoln are representative of the type and quantity of tons from the Wheeler and Wilson Property that may be mined. If the drill samples are not representative of the type, grade, composition and quantity of tons that will be processed from the deposits, then new, fresh, representative metallurgical drill samples will need to be obtained for metallurgical testing. Plus, the 1983 composite sample for cyanide column testing of the Wilson Surface dump is also assumed to be representative of the tons in the dump. The reported size fractions from McPartland, 2011 are indeed the proper size distributions and the ½” nominal size fraction for the Wilson Surface sample is significantly close to the P_{80} of ½”. It is also assumed that all the samples were preserved upon drilling, storage, and metallurgical processing to preserve the integrity of the sample and prevent the sample from natural oxidation and aging. In addition, it is assumed that sufficient reagents and laboratory conditions did not limit the metal extractions.

In general, it takes about a month from the time the material is mined until it is placed on the heap leach pad, ripped, plumbed, solution added with the resultant pregnant solution breakthrough to the recovery system. Results from the load/permeability testing conducted on the column leach residue displayed a measured hydraulic conductivity at a simulated 100 m staked height was 5.33 x 10^{-2} cm/sec (McPartland, 2011). This equated to solution movement through the heap at about 150 ft/day, which is very high in comparison to about 4 ft/day experienced in the Carlin Trend heaps on crushed and ROM ore. This high solution flow value can be attributed to ideal agglomeration conditions in the laboratory setting and should be evaluated operationally on the heap. Solution management (solution application, ore moisture, rest cycles, water balance, evaporation, precipitation) will need to be analyzed and operationally integrated to optimize pregnant grade to the precious metal recovery system, and minimize solution inventory.

The leach kinetics curve derived from the column test data showed a leach cycle of 140 to 166 days. The information is displayed in **Table 17.2**. From the cyanide column test data on the assumed representative composite sample, a leach cycle of 160-170 days per cell would be ideal. The weighted average (contained ounces per deposit) gold recovery was 77.13%, derived from the drill hole samples that were composited, crushed, agglomerated and cyanided column leached at a P_{80} of $^3/_8$ inch.

Table 17.2: Column Leach Kinetic Recovery Data, Percentages of Recovery vs. Fire Assay

	Wheeler Composite		Wheeler Surface	Wilson Composite	
Days	3/8"	1¼"	3/8"	3/8"	1¼"
0	0%	0%	0%	0%	0%
Breakthrough	17.3%	17.2%	12.9%	2.0%	2.5%
5	36.5%	27.9%	37.1%	7.0%	5.9%
10	52.5%	40.0%	55.0%	23.4%	10.5%
15	59.0%	46.0%	61.0%	31.1%	13.3%
20	62.9%	49.8%	64.8%	36.3%	15.5%
40	70.0%	58.3%	70.8%	47.0%	21.4%
70	77.7%	66.2%	77.6%	55.2%	28.9%
100	82.7%	70.4%	81.4%	59.5%	34.1%
140	86.7%	73.8%	85.0%	62.5%	37.5%
141	87.5%			62.5%	
146					
164			85.0%		37.5%
166		74.5%			

18.0 PROJECT INFRASTRUCTURE

18.1 Site Access

The site is readily accessed via East Walker Road. East Walker Road is a well maintained, all-weather, gravel road, intersected off Nevada Highway 208 south of Yerington. The maintenance of the road is performed by Lyon County. The on-site access road from East Walker Road will be improved and widened to 28 feet and will be suitable for over-the-road delivery truck and employee traffic.

18.2 Site Improvements

Limited site improvements are required for mine development. The site plan calls for ancillary facilities, including portable maintenance and warehouse structures and modular administrative offices to be located near the site entrance. All facility support buildings will be modular.

The primary site road will be upgraded from the existing site road from the front entrance gate. A lay down area is planned near the crushing plant and main office.

18.3 Office, Change House and Surveying

The office will be a modular unit, approximately 3,000 square feet, suitable for administrative and management functions, toilets, and a conference room. All visitors and vendors will check in at this location.

The change house will be a modular unit erected in the vicinity of the office and process facilities. The space will include provisions for showers and toilets, and will have a small lunchroom, which can also serve as a meeting area for staff meeting and training sessions.

18.4 Water Supply Distribution

As of the writing of this report, Lincoln has yet to establish a working well head that would service the proposed working site. To date, the water for drilling has been pumped from an existing pond in Pine Grove Canyon.

Lincoln is exploring the possibility of drilling a water well in the valley adjacent to the intersection of East Walker Road and the site access road. Water rights for this new well will need to be acquired. Once a well head is established, it is expected that the well field will connect to a 40,000 gallon storage tank servicing the mine and process facilities via a buried steel pipeline.

18.5 Power

NV Energy is the power company for Nevada. However, it would not be economically feasible to bring power from their transmission lines. Power for the site will be provided by two 1500 KW generators housed in mobile containers. One generator will be located near the crushing facilities and the other near the process and office facilities. Transmission lines will be constructed to distribute the power locally to facilities and to the water well located near Walker Road. Detailed power transmission and usage studies will need to be completed for a feasibility level study.

18.6 Crushing Plant

The crushing operation will consist of the following equipment:

- A primary, secondary and possibly tertiary crusher
- Conveyors
- An agglomerator
- A radial stacker

18.7 The Process Facility

The process facility consists of the following major systems and equipment:

- CIC
- Barren, Recycle and Pregnant Solution Pumps
- Reagents Storage and Distribution System
- Process and Storm Water Ponds
- Building

18.8 Haul Roads

Haul roads will be 70 feet wide outside the pits and 70 feet wide inside the pits with a 10 percent maximum grade. Haul roads will be maintained on a regular basis by the mining contractor and water trucks will be used as part of the dust control measures.

18.9 Perimeter Roads

Fourteen-foot wide perimeter or secondary access roads will be designed around the leach pad. Diversion channels will been sited to convey runoff from the perimeter road, adjacent cut slopes, and upland catchment areas around the leach pad and to prevent this runoff from running onto the leach pad and into the leach circuit.

The south pad perimeter road is 28 feet wide and is considered to be the main access road to the leach pad, pond and plant site.

18.10 Fire Protection

Portable, hand held fire extinguishers will generally be provided to all facilities in accordance with National Fire Protection Association (NFPA) – 13 (2002), NFPA – 14 (2002), NFPA – 122 (2002), International Building Code (IBC) (2003), and the material safety data sheets for the process reagents and consumables. A study to determine if the on-site facilities need additional protection has yet to be done.

18.11 Fencing and Access

Visitors and vendors will check in to the main office entrance located outside of the site fencing. Areas around the heap leach pad, process ponds and cyanide use facilities will be fenced.

Additional fences may be required to limit livestock and discourage human access to the active mine area. Pits will be fenced subsequent to final reclamation.

18.12 Propane

Individual tanks will supply propane gas to buildings that require it for heating purposes.

18.13 Site Radios

Site communications system will be installed and will be a satellite based system through a system provider. There will be handheld units, equipment mounted units, and base units for the radios.

18.14 Site Phones

Phones will be installed in all office facilities to provide communications on site. They will either be cell phones for a wireless provider or if there is no cell service, may be a satellite based system.

18.15 Plant Operation and Instrumentation

The recovery plant, reagent system, pumps and valves will be operated by leach pad operators. Some actions, such as cement or lime dosage, will automatically follow a manually entered set point based on the belt scale.

The recovery plant design will incorporate flow meters for the pregnant solution line coming into the recovery plant. The adsorption system will have samplers for the incoming pregnant solution and at the end of the adsorption system for sampling the barren solution.

Site flow meters, level gages, pressure gages and belt scale(s) will all tie into monitoring networks. There will be a terminal in the operational building as well as in the main office for alarms and condition monitoring.

18.16 Plant Services

18.16.1 Mobile Equipment

Mobile equipment for process plant services will consist of two pick-up trucks, one extended reach fork-lift truck for reagent handling, and two utility all terrain vehicles for servicing the heap and process area.

18.16.2 Building

A pre-engineered metal building will be provided by the contractor once it has been sized to meet operational needs. The building will include a 2-ton jib hoist, insulated walls and roof, a 14-foot x 14-foot overhead door and two man-doors. The interior of the building will be well lit to allow around-the-clock operations.

Two modular buildings may be located at the building site for lunch room and meeting space for plant operators and for administration of the recovery facilities.

18.16.3 Assay/Metallurgical Laboratories

Lincoln plans to construct an assay laboratory off-site, possibly in the town of Yerington.

18.17 Heap Leach Pad and Pond Design

18.17.1 Introduction

Telesto performed the final design of the Pine Grove heap leach facility under the supervision of John Welsh, P.E. The design for the heap leach facility will be part of the PoO for USFS as well as the zero discharge permit and Reclamation Permit Application for a Mining Operation for the NDEP.

18.17.2 Heap Leach Pad Grading Plan

The leach pad will cover an area of approximately 800,000 square feet and has been sited just east of Sugarloaf Mountain. Prior to pad grading, the area will be cleared and stripped of vegetation and topsoil.

18.17.3 Heap Leach Pad Liner System

The leach pad liner system consists of the following components from top to bottom:

- 12-inch protective layer with pregnant solution collection piping
- 60-mil single-side, textured High-density polyethylene (HDPE) geomembrane (primary liner) with textured side down
- Leak detection system
- 12-inch prepared subbase ($k < 1{\times}10^{-5}$ cm/sec secondary liner)
- Natural foundation soils (topsoil removed)

The primary liner for the leach pad will consist of a 60-mil, single-side, textured HDPE geomembrane. The textured side will be placed “down” so that it is in contact with the sub-base. NDEP regulations governing design, construction, operation, and closure of mining operations shall be followed in the design and construction of the leach pad. A quality assurance and control program is proposed during construction to provide high-quality installation of the liner system as required in Nevada Administrative Code (“NAC”) 445A, 439.

A 12-inch-thick protective layer of crushed rock will be placed over the geomembrane to protect the liner from damage by vehicles or conveyors working within the leach pad limits or during material loading.

A summary of the Heap Leach Pad (HLP) design criteria is shown in **Table 18.1**.

Table 18.1: Heap Leach Pad Design Criteria

Item	Design Criteria
Area/Capacity: Total Area/Capacity	 1,165,960 sf / 3.5 million tons
Solution Collection System (as per NAC 445A.438)	Series of 4" diameter perforated ADS and 8" diameter N-12 solid plain-end ADS pipe surrounded in drainage aggregate
Leak Detection System	8' x 8' x 2' deep sump within pregnant and barren/stormwater ponds
Heap Configuration: Nominal Lift Height (Settled) Bench Width Individual Lift Slope Maximum Heap Height Factors of Safety: Static Pseudostatic	 20 ft 14 feet (as required to produce a final overall heap slope of 3H:1V, 18^o) 1.28H:1V (38^o) 80 feet 1.3 (minimum) 1.03 (minimum)
Production Rate	Range 2,000 – 4,000 tons/day (336 days/year)
Dry Density	100 pcf
Specific Gravity	2.2-2.6 (2.4 used in analyses)
Gradation	Minus $^3/_8$ inch
Heap Setback from Pad Perimeter Berm	6 feet (minimum) plus perimeter berm slope of minimum 2 feet
Pad Perimeter Berm Height	2 feet to 20 feet
Access Road Width	Minimum 30 feet
Pad Liner System: (as per NAC 445A.434 and .438)	From bottom to top: 12-inch prepared subbase ($k < 1\times10^{-5}$ cm/sec) 60 mil HDPE geomembrane ($k < 1\times10^{-11}$ cm/sec) 18-inch protective layer Drainage layer comprises solution collection pipework, 18-inch thick drainage aggregate placed over leach pad with additional cover over solution collection pipework
Surface Water Diversion Channels: Permanent Channel Design Storm/Armor Maximum Side Slopes Freeboard	In-place after reclamation – 100-yr/24-hr storm peak flow/riprap 2H:1V(24^o) 1 foot minimum
Culverts Design Storm	25-yr/ 24-hr storm

18.17.4 Process Component Monitoring System

A PCMS will be constructed beneath the solution pipe channel liner system. The PCMS will terminate at the end of the solution pipe channel where any conveyed solution in the PCMS will discharge into the PCMS sump and be pumped back to the solution pipe channel. The PCMS consists of a 4-inch diameter perforated corrugated polyethylene tubing (CPT) (Type SP) pipe placed in an HDPE geomembrane-lined trench backfilled with drainage aggregate. Monitoring and sampling of any collected solution would occur at the PCMS sump.

18.17.5 Loading Plan

Loading plans and schedules were prepared assuming approximately 3.5 million tons of heap material will be loaded on the leach pad. The facility has the design capacity to store 3.5 million tons of heap material when stacked to a maximum height of 80 feet, assuming a material density of 100 pcf.

Loading of the leach pad will commence with a Lift 1, a 20-foot thick lift. When Lift 1 is completed, loading on Lift 2 will commence and continue through Lifts 3 and 4. The pad is large enough that leaching on a portion of the previous lift can continue while the next lift is being loaded.

18.17.6 Recycle/Storm Water Pond LCRS

A summary of the recycle/storm water pond design criteria is shown below in **Table 18.2**.

Table 18.2: Recycle and Storm Water Pond Design Criteria

Item	Design Criteria
Barren/Storm Water Pond: Capacity to 2-feet below crest Freeboard Notes	 400,000 cubic feet (cf) 2 feet Stores the average process solution volume and the 100-yr/24-hr storm volume for Phases 1 and 2
Pregnant Solution Pond: Capacity to 2 feet below crest Freeboard	 400,000 cf 2 feet
Seismic Data (IBC): Design Horizontal Ground Acceleration Magnitude of Design Earthquake Event	 0.243 g (horizontal free-field ground acceleration) M = 7.2
Factors of Safety: Static Pseudostatic	 1.5 1.03
Barren/Storm Water and Pregnant Solution Pond Liner Systems (as per NAC 445A.434 and .438)	From bottom to top: 6-inch prepared subgrade 60-mil HDPE secondary geomembrane ($k < 1\times10^{-11}$ m/sec) Leak detection and collection sump 80-mil HDPE dimpled geomembrane ($k < 1\times10^{-11}$ cm/sec)

18.17.7 Water Balance

The HLP water balance is summarized in **Table 18.3**. The water balance will be verified during the detailed engineering phase.

Table 18.3: Heap Leach Pad Water Balance Input Parameters

Item	Design Criteria
Leach/Load Cycle	45 days load/place pipeworks
Project Life	7 years (residual leaching and reclamation not included)
Solution Application Rate	0.004 gpm/sf
Pregnant Solution Flow Rate	750 – 1,250 gpm
Total Solution Flow Rate	750 – 1,250 gpm
Area Under Leach Method of Application Mineralized material Properties: Dry Density Natural Moisture Content Agglomerated Moisture Content Moisture Content During Leach Residual Moisture Content	290,000 sf (maximum) Drip emitters, Sprays during process fluid stabilization 100 pcf 4% (assumed) 12% (assumed) 13 to 15% (assumed) 5% (assumed)
Average Velocity of Solution Flow Through Heap at 0.004 gpm/sf: Under Leach During Draindown Unleached Area	 20 feet/day (calculated) 2.9 feet/day (calculated) 2.9 feet/day (calculated)
Residual Leach	1 pore volume
Reclamation Completed	Cover completed first. Ponds will be needed another year for evaporation of the final draindown.
Minimum Operational Pond Volume	2 af (pregnant solution directed to the pregnant solution pond, recycle solution immediately pumped back to leach pad)
Emergency Draindown Time	24 hours (emergency backup generators and pumps are available)
Design Storm Event	100-yr/24-hr storm (5.33 inches)
Curve Number	60 (estimated)
Antecedent Moisture Condition (AMC)	II
Carbon Column Throughput	750 – 1,250 gpm
Barren Line Capacity	1,250 gpm
Recycle Line Capacity	1,250 gpm

18.17.8 Monitoring Wells

Currently, four new monitoring wells approximately 300 feet deep are planned for the site.

The monitoring plan will insure that the wells will be installed such that water quality can be monitored up gradient and down gradient of the HLP and ponds. The initial monitoring plan for all wells requires monthly sampling (if possible).

18.17.9 Maintenance and Fuel Storage Facilities

The maintenance facilities will be located near the open pit property areas (Wilson and Wheeler). The facilities will consist of a modular truck shop and are part of the mining contractor's deliverables. Above-ground diesel and gasoline fuel storage tanks with proper spill containment (110 percent of stored volume) will also be provided by the contractor.

Warehousing for the mine will be accomplished with 20-foot or 40-foot sea containers. The containers will be adjacent to the truck shop and will include shelving space for hardware-type material. Additional containers for Lincoln's process parts inventory will be situated convenient to the process work areas.

18.17.10 Explosives Storage

All materials required for the blasting program will be stored on site in federally approved magazines. Blasting materials inventory, supplies and permits will be the responsibility of the mining contractor.

18.17.11 Stockpiles and Waste Dumps

The existing mineralized dumps and stockpiles will be mined during startup and will allow gold production to ramp up more quickly. A working stockpile will be maintained at the crusher site as required to maintain continuous crusher production.

Future waste dumps are planned during the life of the project will be designed using a density factor of 1.5 tons per cubic yard on 50-foot high bench intervals with angle of repose dump faces. Production mine trucks will use end-dump methods to place the waste rock with bench setbacks incorporated into each lift to produce an overall average slope of 18 degrees (3H/1V).

19.0 MARKET STUDIES AND CONTRACTS

19.1 Markets

Gold is sold through commercial banks and metal dealers. Sales prices are obtained based on World spot or London fixes and are easily transacted.

The final product for the Pine Grove mine will be carbon from the CIC loaded with gold. Because of the high copper present in the in-situ resources, the process solutions will be run with high cyanide concentrations. This should reduce the collection of the dissolved copper onto the activated carbon. The loaded carbon will be shipped to a refiner such as Just Refiners in Reno, NV. These contracts are negotiated on a short term basis but will probably have a cost of refining of approximately $1,000 per ton of loaded carbon and a percentage (possibly 2-3%) of the gold recovered from the carbon.

Once the mine has established an operating history with the refiner, payment of typically 90% of the estimated shipment value will be forwarded to the Lincoln's account at the commercial bank that manages the gold sales for the Company. Lincoln's Chief Financial Officer (CFO) will manage the account as a source of immediate funds or gold and silver can be kept in inventory.

19.2 Contracts

No contracts are finalized or in place at this time.

Lincoln intends to utilize contractors during the construction phase of the project and a contractor for mining once the mine enters production.

Equipment contracts will include the supply and construction of the processing facility and rental contracts for the mobile equipment used at the crusher and leach pad. There will also be a rental contract for the generators needed to supply power for the site.

Construction of the leach pad and the process ponds will be provided by contract. The total leach pad area to be constructed is approximately 1,100,000 square feet and the ponds have a total capacity of 800,000 gallons.

Additional contracts will include those for transporting the loaded carbon to the refiners and a contract with the refiners.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Lincoln Gold U.S. Corporation (Lincoln Gold) has retained the services of JBR Environmental Consultants, Inc. (JBR) to assist with the environmental permitting of the Pine Grove Project. The project includes proposed exploration and mining on patented mining claims (i.e. private land). Lincoln Gold also owns adjacent lode mining claims on National Forest System (NFS) lands managed by the United States Forest Service (USFS). Proposed exploration and production will include the development of deposits on both private and NFS lands; thus, discussion herein related to permitting and environmental compliance reflects the requirements for development on both private and NFS lands.

20.1 Required Permits and Statutes

Development of the project patented claims is regulated by the State of Nevada. The regulatory permitting requirements of the state are primarily administered by several bureaus of the Nevada Division of Environmental Protection (NDEP). The NDEP bureaus likely to have regulatory oversight of the project include the Bureau of Mining Regulation and Reclamation (BMRR), the Bureau of Water Pollution Control (BWPC), and the Bureau of Air Pollution Control (BAPC). These bureaus work cooperatively to ensure mining activities in Nevada are compliant with the Clean Water Act (CWA), the Clean Air Act (CAA), and several other federal and state statutes. The potential permits and plans that each NDEP bureau will potentially require and the statute mandating each permit are listed below. The potential permits are based on the activities proposed by Lincoln Gold at this time, and on communication with NDEP during a meeting in which Lincoln Gold and JBR introduced the project. During the meeting, NDEP staff indicated that they felt there were no limitations or uncertainties in obtaining these permits.

20.1.1 Bureau of Mining Regulation and Reclamation (BMRR)

- Water Pollution Control Permit – required by Sections 445A.300 through 445A.730 of the Nevada Revised Statutes (NRS) and Sections 445A.350 through 445A.447 of the Nevada Administrative Code (NAC)
- Notice of Intent (NOI) for exploration (disturbance less than 5 acres) – required by Sections 519A.160 of the NRS and 519A.410 of the NAC
- Reclamation Permit (disturbance more than 5 acres) – required by Sections 519A.010 through 519A.405 of the NRS and Sections 519A.120 through 519A.345 of the NAC.

20.1.2 Bureau of Water Pollution Control (BWPC)

- Notice of Intent for Storm Water Discharges under a National Pollutant Discharge Elimination System (NPDES) General Permit (Storm Water Permit) and associated Storm Water Pollution Prevention Plan (SWPPP) – required by the CWA and Sections 445A.300 through 445A.730 of the NRS
- Spill Prevention, Control and Countermeasures Plan (SPCC Plan) – required by the CWA

20.1.3 Bureau of Air Pollution Control (BAPC)

- Facilities Operating Permit (Air Quality Permit) – required by the CAA (42 USC §7401 et seq.) and by Nevada air quality rules and regulations (Chapters 445B of the NRS and 445B of the NAC)
- Surface Area Disturbance Permit and Dust Control Plan – required by the CAA and by Nevada air quality rules and regulations

In accordance with state law, Lincoln Gold must post reclamation surety with NDEP before development of the project patented claims would be authorized. Reclamation liabilities are determined by the Nevada Standardized Reclamation Cost Estimator (SRCE). The SRCE was created in accordance with the guidelines developed during the implementation of the Nevada Standardized Unit Cost Project, a collaborative effort by the NDEP, Bureau of Land Management (BLM) - Nevada State Office, and the Nevada Mining Association. The SRCE utilizes standardized reclamation calculation methods, data, and procedures to estimate the cost of reclaiming a mine site as if a third-party contractor for the State of Nevada is performing the reclamation. Cost data is maintained and updated to account for deflation, inflation, and other contingencies. Once a cost is calculated and a reclamation surety is posted, the amount of the surety must be reviewed at least once every three years thereafter to determine if it is still adequate for reclamation costs with inflation considered. The NDEP accepts several instruments for reclamation surety, including surety bonds, cash, certified checks or bank drafts, irrevocable letters of credit, and certificates of deposits.

Development of the project patented claims must also comply with the Lyon County Zoning Ordinance. The project patented claims are presently zoned as Rural Residential (PRR5). The Zoning Ordinance (Title 10, Chapter 3) states that a Special Use Permit (SUP) is required for mining activities in areas that are zoned PRR5. In accordance with the Zoning Ordinance, Lincoln Gold must apply for and obtain a SUP before mining could commence on the project patented claims. Under normal conditions, issuance of a SUP may require up to 180 days from the date the application is filed.

The USFS administers exploration and mining on NFS lands under mining regulations defined in Chapter 36 of the Code of Federal Regulations, Part 228, Subpart A (36 CFR 228 Subpart A). In accordance with 36 CFR 228 Subpart A, future exploration and mining on the project unpatented claims will require Lincoln Gold to submit a Plan of Operations (PoO) for review by the USFS, Bridgeport Ranger District. It is likely the PoO will be developed by revising the NDEP Reclamation Permit to include additional information required by the USFS. Therefore, the PoO will include the activities proposed on the unpatented and patented claims, and will serve as an overall plan for the entire project. Following their review, the USFS will determine whether an Environmental Assessment (EA) or an Environmental Impact Statement (EIS) is required for compliance with the National Environmental Policy Act of 1969 (NEPA). Preliminary discussions with the Bridgeport Ranger District indicate that an EA will likely be required for the project. The EA will be prepared in accordance with USFS guidelines, NEPA, and the Council on Environmental Quality (CEQ) regulations (40 CFR 1500-1508) for implementing NEPA. Since the EA will analyze the activities proposed in the PoO, the NEPA analysis will include the activities proposed on the unpatented claims and the activities occurring or proposed on the patented claims.

The anticipated timeline for completion of an EA is 9 to 12 months after development of the PoO; however, several operational specifications are yet to be determined, such as identifying a source of electricity or water supplies for the proposed project operations. Once currently undetermined operational specifications are decided upon, the impacts of the project may become larger or affect additional lands administered by other federal agencies, such as the BLM. In such an event, the EA may require a lengthier period to complete. If BLM-administered lands are affected, the BLM would likely become a participating agency in the NEPA process. If impacts intensify, any agency involved in the NEPA process may determine that an EIS is required for NEPA compliance. The pertinent regulatory agencies, regulations, and permits that will ultimately be required for construction and operation of the proposed project will be identified when all operational procedures and specifications have been determined by Lincoln. A limited amount of exploration may occur prior to an EA, with appropriate notification and assessment, and with the approval of the USFS.

In addition to NEPA, the USFS must also ensure the project is compliant with other federal statutes, including the Endangered Species Act (ESA), the National Historic Preservation Act (NHPA), Native American Graves Protection and Repatriation Act (NAGPRA), and all applicable federal orders, directives, and regulations pertaining to the development of NFS lands. Compliance with the applicable federal statutes and regulations must be considered in the NEPA analysis. In anticipation of this, Lincoln Gold has performed consultation with state agencies to gather data pertaining to wildlife, vegetation, and cultural resources that may potentially be located within the project area. The United States Fish and Wildlife Service (USFWS), Nevada Department of Wildlife (NDOW), and Nevada Natural Heritage Program (NNHP) were consulted to determine if there are records of federally threatened and endangered species within or near the project area. Wildlife and plant surveys have been completed in portions of the project area. Biological surveys in the remaining portions of the project area are scheduled to be completed in spring 2012. Lincoln Gold has conducted a Class III Cultural Resources Assessment within the project area boundary and the findings have been submitted to the Nevada State Historic Preservation Office (SHPO) for concurrence. The anticipated EA completion time of 9 to 12 months is also dependent on the completion of any additional field surveys the USFS may determine are necessary.

A reclamation surety that is adequate for the reclamation of the entire project, which includes development of the patented and unpatented claims, must be posted before Lincoln will be authorized to proceed with activities. The reclamation liabilities for the project will be determined using the SRCE. The reclamation surety would be administered by NDEP, and would be subject to the same administrative policies and regulations as the reclamation surety for the patented claims alone, as described above.

20.2 Environmental Liabilities and Permitting

The Pine Grove District is a historic gold producing district with several underground mines. The area was mined between the 1860s and 1915. These underground workings are no longer accessible. Some old buildings and the remnants of a stamp mill are present. In 1988 the property was optioned to Teck, which performed extensive exploration at the site. Teck drilled 160 holes and bladed roads throughout the proposed production areas. Teck dropped their option in 1992. Reclamation was conducted on public lands administered by the USFS. No reclamation was performed on the patented claims, which is under the jurisdiction of the State of Nevada. It is understood that Teck had permits with the USFS for those exploration efforts; however, did not apply for permits or submit a bond with NDEP. While Lincoln Gold will not be responsible for reclamation of pre-1980 disturbances, BMRR will require Lincoln Gold be responsible for reclamation and bonding associated with Teck's exploration disturbances on the patent claims in addition to Lincoln Gold's proposed disturbances.

A Class III Cultural Resource Assessment has been conducted within the project area boundary and findings have been submitted to the SHPO for concurrence. Any resources determined to be significant by SHPO will be managed through avoidance or approved mitigation during development.

Section 404 of the CWA establishes programs to regulate the discharge of dredged or fill material into waters of the United States (U.S.), including wetlands. Activities in waters of the U.S. regulated under this program include fill for development, road crossings, and most other components typical of mining projects. A delineation was performed November 7-11, 2011, to determine whether waters are present that the U.S. Army Corps of Engineers considers jurisdictional (waters of the U.S.) and subject to the CWA. Findings of the survey have not been presented to the Army Corps of Engineers as of the date of this report. Preliminary findings by JBR show that there is no significant nexus between drainages in the Pine Grove project area and waters of the U.S. There appears to be no connection between the Pine Grove drainages and the Walker River which is east of the project area. JBR does not anticipate that the drainages at Pine Grove will fall under the jurisdiction of the waters of the U.S.

After presenting the study findings to the Army Corps of Engineers, a determination will be made whether the drainages are under jurisdiction of the waters of the U.S. From that determination, Lincoln Gold will gain an understanding of the optimal places to locate project components to minimize wetland impacts while providing for safe and efficient mine operations. If impacts are unavoidable, Lincoln Gold will be able to initiate Section 404 permitting or other similar wetland permitting with the U.S. Army Corps of Engineers.

A Gantt chart has been developed by JBR to display the expected timing of permit acquisition for the Pine Grove Project. A summary of the chart is presented as **Table 20.1**.

Table 20.1: Summary of Permitting and Environmental Tasks to be Completed (JBR)

Actions	Estimated Start	Estimated Completion
Additional Area Needed for Heap Leach Facility	Early Autumn 2011	Late Summer 2012
Development of Plan of Operations (i.e. Reclamation Permit Application)	Early Autumn 2011	Mid Spring 2013
Development of Supplemental Plans Included in the Plan of Operations	Late Autumn 2011	Early Summer 2012
Actions Supporting Development of Plan of Operations or Necessary Permits	Late Autumn 2011	Late Spring 2012
Possible Transmission Line Right-of-Way	Mid Winter 2011/2012	Mid Spring 2013
Waters of the U.S. and Wetlands	Late Autumn 2011	Late Spring 2012
Remaining Biological Requirements	Late Winter 2011/2012	Early Winter 2012/2013
Remaining Cultural Requirements	Early Autumn 2011	Early Winter 2012/2013
Prepare Environmental Assessment	Early Spring 2013	Mid Winter 2013/2014
Prepare Other/Plans in Support of Necessary Permits	Early Winter 2012/2013	Early Spring 2013
Other Permits/Notifications	Late Autumn 2012	Mid Winter 2013/2014
Bonding	Early Summer 2012	Early Summer 2013

Notes

1. Schedule includes several items that regulatory agencies have complete control over, such as application and plan review periods or public review periods. Agencies may require lengthier periods of time for the items in the schedule that they have control over.
2. Schedule assumes that regulatory agencies will require an Environmental Assessment for project rather than an Environmental Impact Statement.
3. Along with permit application and review periods, the schedule accounts for weather constraints where applicable. In general, most vegetation, wildlife, and cultural surveys must be performed when the ground is free of snow.

21.0 CAPITAL AND OPERATING COSTS

The reader is reminded that his PEA is preliminary in nature, and is based on technical and economic assumptions which will be evaluated in more advanced studies. The PEA is based on the Project resource model which consists of material in Measured, Indicated and Inferred classifications. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in this PEA will be realized. The PEA results are only intended as an initial, first-pass review of the potential project economics based on preliminary information.

21.1 Capital Costs Introduction

Initial capital costs will be primarily incurred during the year of pre-production (Year -1). See **Table 21.1** for a list of capital costs, annual revenue and cash position by year. There are also several years of costs for environmental and engineering studies and condemnation drilling prior to Year -1 which are summarized under Pre-production on this table. Payback for the capital is 32 months.

Table 21.1: Pine Grove Capital Costs, Annual Revenue and Cash Position

(000s $)	Pre-Production	Year						Project Total
		1	2	3	4	5	6	
Permitting, Bonding, Engineering	3,700	-	-	-	-	-	-	3,700
Site Power Infrastructure	50	-	-	-	-	-	-	50
Condemnation Drilling	1,000	-	-	-	-	-	-	1,000
Leach Pad	2,750	-	-	-	-	-	-	2,750
Crushing, Conveying, Agglomerating	2,709	-	-	-	-	-	-	2,709
Process Plant	1,664	-	-	-	-	-	-	1,664
Prestrip & Earthworks	9,786	-	-	-	-	-	-	9,786
Site Layout	1,590	-	-	-	-	-	-	1,590
Total Direct Capital Costs	23,249	-	-	-	-	**-**	**-**	23,249
Contingency @ 20%	4,620	-	-	-	-	-	-	4,620
Operating Capital	-	5,373	-	-	-	-	-	5,373
Total Capital	27,869	5,373	-	-	-	-	-	33,242
Annual Revenue	-	37,226	33,959	38,892	33,310	-	-	143,387
Cash Position	(27,869)	31,853	33,959	38,892	33,310	**-**	**-**	110,145

21.2 Operating Costs Introduction

Operating costs for the Pine Grove Project were estimated by Telesto utilizing Mining Cost Services information, Telesto experience with projects of similar size in the area and vendor quotes. They were based on 3 million tons of pre-production stripping and then going into full production averaging an annual new mineralized material production rate of approximately 1.0 million tons per year over a 4-year period. The estimate is based on current dollars third quarter 2011 USD and excludes escalation.

21.3 Organization

The Pine Grove Project will use a traditional organizational structure and is divided into three primary areas: mining, processing, and general services and administration (G & A). Mining operations will be performed by a contractor while Lincoln personnel will be responsible for geology, mine planning, ore control and surveying. After mineralized material is delivered to the crusher stockpile by the mining contractor, Lincoln will crush, agglomerate and stack the mineralized material on the leach pad. Lincoln personnel will be responsible for the leaching and leach operations. G & A will be comprised of a general manager, human resources, some accounting and warehousing functions, safety, and environmental reporting.

Initial Lincoln project staffing will include the General Manager, Administrative Assistant/HR, Safety Manager, warehouse person, environmental technician, accounting clerk, and leaching operators. These people will help train and recruit project staff to build the organization to a point where it can perform all the job functions an open pit, heap leach gold mine requires. All employees will be employed by Pine Grove Mining Company, a wholly owned subsidiary of Lincoln Mining Corporation.

21.3.1 General Management

The General Manager will oversee the entire operation. The Mine Engineer, Process Superintendent, and Safety Manager will report to this person. The General Manager will report to Lincoln's Operations Officer.

21.3.2 Finance and Accounting

The accounting clerk will also act as the project accountant. This person will be responsible for all finance and accounting duties and will work closely with corporate accounting personnel. The accounting clerk will report to the General Manager as required.

21.3.3 Human Resources

The Administrative Assistant will be responsible for onsite human resource activities. Lincoln plans to utilize an offsite human resources contractor for recruiting and benefits. The Administrative Assistant reports to the General Superintendent.

21.3.4 Purchasing and Materials Management

The Purchasing/Planner will be responsible for all purchasing, inventory, and material management functions for the operation. This person will support the mining and processing groups. This position will report to the Process Superintendent.

21.3.5 Public Relations

The General Manager will be responsible for local community relations activities and will coordinate with the corporate offices of Lincoln on these activities.

21.3.6 Environmental and Permitting

The Environmental Technician will be responsible for all environmental and permitting issues and requirements which includes the monthly, quarterly, and annual reporting. This person will coordinate closely with Lincoln management and others that Lincoln may retain, in order to stay compliant with all local, State, and Federal requirements. The Environmental Technician will report to the Mine Engineer.

21.3.7 Health and Safety

The Safety Manager will have overall responsibility for all health and safety requirements related to Lincoln personnel and all on-site contractors. Although the mining contractor is responsible for his employees' safety and training, the Safety Manager will review and verify all training. The safety Manager will oversee the coordination of on-site training of personnel in order to stay current and compliant with Mine Safety and Health Administration ("MSHA") requirements and will act as the lead investigator in all incidents or accidents.

21.3.8 Corporate Support

Lincoln's corporate office will support the financing and accounting functions, public relations, environmental and permitting activities, health and safety functions, as well as other technical support for the exploration, mining and processing areas.

21.3.9 Emergencies

The on-site emergency medical and security will be a contracted entity that is currently being used at other local mines. An Emergency Response Plan will be formulated with this entity outlining site-specific emergency response procedures. This plan will be coordinated with local authorities and will provide additional detailed information. The plan will address issues from chemical spills to medical emergencies and will contain an emergency contact list.

For medical emergencies, the town of Yerington, 21 miles (34 kilometers) distant, has a small hospital equipped to handle most medical emergencies. The closest major city is Reno, 100 miles distant, which has several large hospitals and a flight-for-life.

21.3.10 Compensation Plan Structure

Salaries and hourly wage rates will be commensurate with local Nevada mining industry labor rates and the Yerington job market. An allowance of 39 percent of base wages and salaries has been provided in labor cost estimates. The competitive benefit package will include provision for health insurance, holiday pay and vacation pay.

21.3.11 Training

Because of the stated preference to hire local employees, and current lack of similar local mining operations, many of the staff and hourly paid positions will require some amount of training.

21.4 Mining

It is the intent of Lincoln to utilize a mining contractor to perform all aspects of mine development, preproduction stripping, and the mining of mineralized material and waste rock from the pits, heaps, and dumps designated by the mine production schedule for the Pine Grove Project. Lincoln will also employ personnel to oversee and assist the contractor.

21.4.1 Contract Mining

The average weighted mining cost for mineralized material and waste from all pits is estimated to be $2.25 per ton, including fuel. These costs are life of mine weighted averages as mineralized material and waste costs vary according to the pit being mined. Pre-production estimates are $2.75 per ton and include the early development work for the property. Pre-productions costs are included with the capital costs. The scope of work detailed in the contract mining bid package will include the following:

- Provide all mine related equipment.
- Drilling and blasting of material and waste rock.
- Excavation and haulage of mineralized material to the crusher stockpile (or other areas designated by Owner).
- Excavation and haulage of waste rock to waste dumps or road fills designated by Owner.
- Excavation of water diversion ditches to control surface runoff in the mining area, access roads and other contractor work areas.
- Secondary breakage of all oversized (+30 inches) mineralized material from the open pits.
- Dewatering of all active pits.
- Stabilization and maintenance of pit walls by appropriate drilling and blasting procedures, face scaling, and safety berm maintenance as designated by Owner and/or state and federal regulations.
- Construction and maintenance of all haulage and access roads and ramps necessary for on-going mining operations.
- Dust suppression in all working areas of the mine, including haulage and access roads.
- Maintenance of Contractor's equipment to ensure timely availability and provide the necessary services to support safe and efficient mining operations. All equipment will be maintained to meet MSHA regulations.
- The Contractor will provide all necessary and ancillary support equipment and supplies, such as cranes, mechanics, trucks, fuel & lube trucks, tools and repair parts including:
    - Operating labor
    - Maintenance labor
    - Oil and grease
    - Tires
    - Major repairs
    - Minor repairs
    - Ground engaging parts

The mining contract specifically excludes diesel fuel from the contractor's scope. The contractor will provide fuel tanks and manage the system, but pass the fuel costs directly to Lincoln without markup.

The expected contract mining equipment fleet is shown in **Table 21.2**.

Table 21.2: Anticipated Contract Mining Equipment

Equipment	Number	Make & Model	Comment
Dozer	1	CAT D8	or similar
Dozer	1	CAT D9	or similar
Dozer	1	CAT D9R	or similar
Loader	2	CAT 992	or similar
Loader	1	CAT 988	or similar
Drill	2	IR DML	or similar
Haul Truck	7	CAT 777	or similar
Motor Grader	1	CAT 16H	or similar
Water Truck	1	CAT 773B	or similar
Pickup Truck	4	Heavy Duty	or similar
Service Truck	1	W900	or similar
Mechanics Truck	1	Ford L8000	or similar
Tire Truck	1	W900	or similar

The manpower requirements for contract mining include heavy equipment operators, light and heavy vehicle mechanics, lubricator, warehouse clerk, foreman, mine manager, and general administrative support. The total manpower for the mining contractor is expected to be approximately 35 personnel.

21.4.2 Blasting Design

The blasting program will be designed to maximize mineralized material fragmentation while minimizing damage to final pit walls. The design was developed to maintain a nominal production rate of approximately 15,000 tons of material (mineralized material plus waste) per day.

The mining contractor will perform all blasting related activities with Lincoln review and oversight. This contractor or its chosen subcontractor will supply the blasting and explosives permits as well as all the labor and equipment necessary to deliver and store explosive supplies, load and tie-in blast holes and initiate the blasts.

Drill holes will be 4-inches to 6-inches in diameter in a 15 foot-square pattern with a sub-drill of 3 to 5 feet. The mined bench height will be 20 feet. A mixture of ammonium nitrate and fuel oil will be the primary blasting agent.

21.4.3 Grade Control Procedures

The site grade control program will be conducted by Lincoln personnel and has three principal objectives:

- Ensure that the material to be mined as mineralized material meets the criteria required to be profitable and leachable;
- Provide information to the planning and operations personnel to enable them to delineate the mineralized material/waste contacts in the field prior to mining; and
- Provide additional information to improve the quality of future grade and mineralized material type forecasts to optimize gold production and maximize economic metal recovery.

To achieve these objectives, bench mineralized material control maps will be generated and updated with current assay results. These maps will be employed in mine planning for mineralized material scheduling as well as in the planning of drill patterns and blast design. Quality control procedures will be established for drill hole sampling, the handling of samples, recording of assay results, mapping of the mineralized material and waste contacts, and the communication and layout of these results in the field. As mining progresses, geological mapping will be updated to insure mine development is based on the best available information.

Drill hole samples will be assayed for total gold at an offsite laboratory with results made available within 24 hours. These results will be mapped and laid out in the field prior to excavating the broken material.

The engineering department of Lincoln will have primary responsibility for the grade control program including data collection, interpretation, mineralized material estimation, mineralized material block generation, grade control reports, and engineering implementation of digging plans and contractor supervision.

Routine reconciliation analysis of the mineralized material will be performed by comparing the block model grades with the actual drill-hole sample assays. Mineralized material/production tonnage will be reconciled by comparing the block model data to the belt scale.

21.4.4 Mineralized material Dilution

Mineralized material dilution may occur in three main areas:

- Internal dilution is sub-economic material that is included in the mineralized material block that cannot be economically separated due to the size of the significant mining unit as determined by the size of the equipment utilized;
- Contact dilution occurs when the mineralized material/waste contact cannot be clearly projected because it is irregular in nature; and

- Operational dilution results from operating practices that mix the mineralized material and waste as a result of blasting, over digging the delineated contacts or poor floor grade control.

Quality standards will be developed in each of these areas to minimize dilution of the mineralized material without negatively impacting the economics of the mine. The mining plans developed for this report assume that all of the planned ounces and tons will be mined.

21.4.5 Lincoln Mining Labor

Lincoln's manpower requirements and costs to support the mining operation are shown in **Table 21.3**.

Table 21.3: Labor Requirements, Mining

Position	Qty	Salary	Hourly	Overtime (10%)	Burden (39%)	$ / person	Annual Cost $
Mining Engineer	1	$ 110,000			$ 42,900	$ 152,900	$ 152,900
Mining Geologist	1	$ 95,000			$ 37,050	$ 132,050	$ 132,050
Ore Control Technician	1		$ 53,123	$ 58,436	$ 22,790	$ 81,225	$ 81,225
Survey Technician	1		$ 53,123	$ 58,436	$ 22,790	$ 81,225	$ 81,225
Total Mining Labor Costs	**4**						**$ 447,400**

21.4.6 Crushing, Agglomeration, and Heap Stacking

Lincoln will perform the mineralized material crushing, agglomeration, and heap stacking as designated in the material production schedule for the Pine Grove project.

A crushing cost of $4.97 per ton of mineralized material was used based on Mining Cost Service information, Telesto experience with projects of similar size in the area and vendor quotes. Costs for this area include the following:

- A crushing, screening, conveying, agglomeration facility;
- Heap stacking via overland and portable conveyors;
- Lime and cement silos, bag houses, electrical power and water;
- Maintenance of equipment to support safe and efficient crushing operations. Lincoln will provide all necessary ancillary support equipment and supplies, such as cranes, mechanics trucks, tools and repair parts;
- Operating labor;
- Maintenance labor;
- Operating supplies such as lime and cement: and
- Power costs for renting, operating and maintaining a mobile generator.

21.4.7 Crushing Labor

The manpower requirements for the crushing, agglomeration, and heap stacking includes heavy equipment operators, light mechanics, an electrician, and crushing facility operators. The crushing facilities are scheduled to be operated two shifts per day, 6 days a week, depending on tonnage requirements. The total manpower for the crushing, agglomeration, and heap stacking contractor is expected to be approximately 10 personnel. Because the work of the mechanics and electrician will primarily be associated with the crusher and its mobile equipment support, those employees are shown on the crushing labor table (See **Table 21.4**).

Table 21.4: Labor Requirements, Crushing

Position	Qty	Salary	Hourly	Overtime (10%)	Burden (39%)	$ / person	Annual Cost $
Loader Operators	2		$ 55,952	$ 61,547	$ 24,003	$ 85,551	$ 171,101
Crusher Operators	5		$ 53,248	$ 58,573	$ 22,843	$ 81,416	$ 407,081
Mechanics/Electricians	3		$ 59,384	$ 65,322	$ 25,476	$ 90,798	$ 272,394
Total Crushing Labor Costs	**10**						**$ 850,576**

21.4.8 Agglomerating Reagents

Heap leach reagent needs are shown in **Table 21.5**.

Table 21.5: Heap Leach Reagents

Reagent	units/year	lb/ton	units	$/unit	annual costs	$/ton
Cement	4,000,000	4	pounds	$0.07	$276,000	$0.28
Lime	4,100,000	4.1	pounds	$0.10	$393,600	$0.39

21.4.9 Crushing Costs

Crushing costs are shown in **Table 21.6**.

Table 21.6: Crushing Costs

	#	Units/Month	Cost/Unit	Unit/Time	$/ton
Operating Supplies & Liners	1	estimate	$ 363,000	$/year	$ 0.363
Loader (CAT 988) Rental	1	each	$ 17,325	$/mo	$ 0.017
Loader PM Maint	390	hours	$ 8.51	$/hr	$ 0.040
Loader GET	390	hours	$ 12.10	$/hr	$ 0.057
Loader - Fuel	390	hours	$ 40.43	$/hr	$ 0.189
Dozer - D-7 LGP Rental	1	each	$ 13,500	$/mo	$ 0.162
Dozer - PM Maint	208	hours	$ 6.59	$/hr	$ 0.016
Dozer GET	208	hours	$ 6.38	$/hr	$ 0.016
Dozer - Fuel	208	hours	$ 24.42	$/hr	$ 0.061
Contracted Maintenance	1	estimate	$ 2,000	$/year	$ 0.002
Mtce/Fuel Truck	1	estimate	$ 5,000	$/mo	$ 0.005

Table 21.6: Crushing Costs

	#	Units/Month	Cost/Unit	Unit/Time	$/ton
Genset Rental	1	each	$ 37,211	$/mo	$ 0.037
Genset Fuel	450	hours	$ 317.46	$/hr	$ 1.714
Genset Maintenance	1	each	$ 2,500	$/mo	$ 0.003

Because it is preferable to keep the crusher operating on a regular basis, for this economic study the operating cutoff grade was dropped to 0.007 opt Au. This insured a constant ore feed to the crusher without major time gaps associated with waiting for a higher feed grade to become available from the mine. This is an area that requires additional study.

21.5 Heap Leaching

Lincoln will perform all heap leaching activities. Leaching will be continuous and occur year round. Piping on the pad will occur five days per week. A leaching cost of $6.81 per ton of mineralized material was used based on Mining Cost Service information, Telesto experience with projects of similar size in the area and vendor quotes.

Heap Leaching costs are broken into the following primary areas:

- Labor
- Reagents
- Operating Costs

21.5.1 Labor

Table 21.7 provides a summary of the labor required to operate the heap leach.

Table 21.7: Labor Requirements, Leaching

Position	Qty	Salary	Hourly	Overtime (10%)	Burden (39%)	$ / Person	Annual Cost $
Leach Pad Operators	4		$ 47,528	$ 52,281	$ 20,390	$ 72,670	$ 290,681
Piping Crew	2		$ 37,440	$ 41,184	$ 16,062	$ 57,246	$ 114,492
Total Leaching Labor Costs	**6**						**$ 405,173**

21.5.2 Reagents

Table 21.8 provides a summary of the reagents required to operate the heap leach. The rate of sodium cyanide application for the Wheeler Pit is 1.59 lbs. per ton and for the Wilson Pit is 2.51 lbs. per ton. For costing purposes, an average application rate of 2.5 lbs. per tons was used.

Table 21.8: Heap Leach Reagents

Reagent	Units/Year	lb/ton	Units	$/Unit	Annual Costs	$/ton
Sodium Cyanide	2,500,000	2.5	Pounds	$1.15	$ 2,875,000	$ 2.88
Anti-Scale[1]	43	0.012	Pounds	$1.44	$ 62	$ 0.00

[1] Unit costs for this reagent is taken from Mining Cost Service (2010).

21.5.3 Operating Costs

Because the leach pumps use the preponderance of the power for processing and offices, processing power costs for a generator are included in this area. **Table 21.9** provides a summary of the operating costs required to operate the heap leach.

Table 21.9: Heap Leach Costs

	#	Units/Month	Cost/Unit	Unit/Time	$/ton
Genset Rental	1	each	$ 37,211	$/mo	$ 0.04
Genset Fuel	720	hours	$ 317	$/hr	$ 2.74
Genset Maintenance	1	each	$ 3,600	$/mo	$ 0.00
Piping/Drip Tubing	1	estimate	$ 38,000	$/year	$ 0.04
Maintenance Supplies	1	estimate	$ 100,000	$/year	$ 0.10

21.6 Processing

Lincoln plans to ship loaded carbon from the carbon cells to a refiner to recover the contained gold. All assaying will be done at on off-site lab operated by Lincoln. Assay costs are included with the processing costs. A processing cost of $3.50 per ton of mineralized material was used based on Mining Cost Service information, Telesto experience with projects of similar size in the area and vendor quotes. This cost also includes the refining costs.

Processing costs are broken into the following primary areas:

- Labor
- Reagents
- Consumables

21.6.1 Processing Labor

Table 21.10 provides a summary of the labor required to operate the facility.

Table 21.10: Labor Requirements, Processing

Position	Qty	Salary	Hourly	Overtime (10%)	Burden (39%)	$ / person	Annual Cost $
Process Superintendent	1	$ 110,000			$ 42,900	$ 152,900	$ 152,900
Operators	2		$ 58,656	$ 64,522	$ 25,163	$ 89,685	$ 179,370
Assay Technicians	2		$ 56,368	$ 62,005	$ 24,182	$ 86,187	$ 172,373
Total Processing Labor Costs	**5**						**$ 504,643**

21.6.2 Reagents

Reagents are those chemicals used in this operation for the column cells and the laboratory. **Table 21.11** provides a summary of these items on an annual basis. Annual costs for carbon are not shown. The plan is to load the carbon to approximately 100 ounces Au per ton of before shipping it to a refiner. Therefore carbon use will vary with the number of ounces loaded in a given timeframe. Average carbon cost will be approximately $23 per ounce of gold produced. Also not shown in the table is the percent of gold that is retained by the refiners. It is expected that it will be approximately a 2% loss.

Table 21.11: Process Reagents, Heap Leach and Column Cells

Reagent	tons/Year		Per ton	Units	$/Unit	Annual Costs
Carbon						
Caustic[1]	60,000	ton/year	0.060	pounds	$ 0.40	$ 24,000
Anti-Scale	12,000	ton/year	0.012	pounds	$ 1.44	$ 17,280
Propane	25,000	ton/year	0.150	gallons	$ 2.17	$ 54,250
Borax	6,000	ton/year	0.006	pounds	$ 1.17	$ 7,020
M&O Supplies	$ 30,000	$/year	0.609	$	$ 0.61	$ 30,000
Wear Items	$ 0	ton/year	0.100	$	$ 0.10	$ 0
Mobile Equipment	$ 30,000	ton/year	0.030	$	$ 0.03	$ 30,000
Assaying			3,000	samples/mo	$ 10.00	$ 30,000
Refining Off-Site	$ 1,000	$/ton carbon		$/ounce	$ 10.00	$ 10,000
M&O Supplies	$ 30,000	$/year	0.609	$	$ 0.61	$ 30,000

[1] Unit costs for this reagent is taken from Mining Cost Service (2010).

21.6.3 Processing Cost Summary

Table 21.12 provides a summary of the processing cost on an annual basis.

Table 21.12: Process Cost Summary

Description	Cost per Year	$/ton
Crushing	$4,043,801	$5.03
Leaching	$4,802,541	$5.97
Process	$2,331,874	$2.90
Total Costs	$11,178,216	$13.89

21.7 General and Administrative (G & A)

The G & A costs are shown in the Annual G & A Cost Summary **Table 21.13** below. These costs include the cost of labor for personnel (with a 39 percent payroll burden rate), property taxes, insurance, bonding, and other indirect costs.

Table 21.13: Annual G & A Cost Summary

Description	Number	Salary & Benefits	Cost per Year	$/ton
General Manager	1	$ 208,500	$ 208,500	$ 0.28
Safety Manager	1	$ 152,900	$ 152,900	$ 0.20
Purchasing and Planning	1	$ 86,187	$ 86,187	$ 0.11
Administration/HR	1	$ 70,890	$ 70,890	$ 0.09
Accounting Clerk	1	$ 59,770	$ 59,770	$ 0.08
Environmental Technician	1	$ 78,681	$ 78,681	$ 0.10
Security and Emergency Medicine		$ 262,928	$ 262,928	$ 0.35
Insurance		$ 175,000	$ 175,000	$ 0.23
Property Taxes		$ 110,000	$ 110,000	$ 0.15
Outside Services/Legal		$ 120,000	$ 120,000	$ 0.16
Communications		$ 50,000	$ 50,000	$ 0.07
G & A Supplies & Services		$ 60,000	$ 60,000	$ 0.08
Permits, Annual Maintenance		$ 150,000	$ 150,000	$ 0.20
Dry Trailer		$ 20,000	$ 20,000	$ 0.03
Contributions, Travel, Misc		$ 150,000	$ 150,000	$ 0.20
Small Vehicles		$ 100,000	$ 100,000	$ 0.13
Total G & A Costs	**6**		$ **1,854,856**	$ **2.64**

Property taxes were estimated based on a 35 percent of taxable value multiplied by the tax rate of 3.66 percent. Taxable value equals market value of land plus the cost of improvements minus 1.5 percent depreciation.

21.8 Operating Cost Summary

Accounting spreadsheets were developed based on mineralized material and waste from the annual mine plans, processing costs, and G & A costs. These costs varied annually based on mined and processed quantities.

22.0 ECONOMIC ANALYSIS

This PEA is preliminary in nature, and is based on technical and economic assumptions which will be evaluated in more advanced studies. The PEA is based on the Project resource model which consists of material in Measured, Indicated and Inferred classifications. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them. The current basis of project information is not sufficient to convert the mineral resources to Mineral Reserves, and mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in this PEA will be realized. The PEA results are only intended as an initial, first-pass review of the potential project economics based on preliminary information.

22.1 Cash Flow

Table 22.1 shows a simplified cash flow using only operating costs. Mining staff and G & A costs begin in the pre-production year. Pre-production mining is capitalized. The mining Staff and G&A costs continue into year 6 as leaching continues and reclamation begins.

Table 22.1: Pine Grove Cash Flow at $1425 Gold Price

(000s $)	Year							Project Total
	-1	1	2	3	4	5	6	
Gold Revenue	-	37,226	33,959	38,892	33,310	-	-	143,387
Ore Mining Expenditure	-	(2,550)	(1,877)	(2,389)	(483)	-	-	(7,299)
Waste Mining Expenditure	-	(5,206)	(6,384)	(4,768)	(479)	-	-	(16,837)
Mining Staff	(453)	(453)	(453)	(453)	(209)	(87)	-	(2,110)
Revenue Minus Expenditure	(453)	29,017	25,245	31,282	32,138	(87)	-	117,141
Processing Expenditure	-	12,397	12,256	12,469	8,810	2,447	-	48,378
Processing Revenue	(453)	16,620	12,989	18,813	23,329	(2,534)	-	68,763
G & A	927	1,855	1,855	1,855	1,855	1,855	-	10,202
Net Production Revenue	(1,381)	14,765	11,134	16,958	21,474	(4,389)	-	58,561

Table 22.2 on the next page shows a more detailed summary that includes additional items such as royalties, reclamation costs and the Nevada Net Proceeds Tax.

LINCOLN MINING COMPANY

		Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
GOLD PRICE	$ 1,425											
Gold Ounces Produced		-	-	-	26,124	23,831	27,293	23,375	-	-	-	100,623
Bullion Revenue												
Gold		-	-	-	37,226,192	33,959,219	38,892,071	33,309,895	-	-	-	143,387,377
Direct Operating Costs												
Mining Costs - Contractor		-	-	-	7,756,062	8,260,589	7,156,842	962,441	-	-	-	24,135,934
Mining Costs - Company		-	-	453,401	453,401	453,401	453,401	209,284	87,225	-	-	2,110,112
Crushing Costs		-	-	-	4,640,446	4,640,446	4,640,446	2,850,512	-	-	-	16,771,849
Leaching Costs		-	-	-	5,255,357	5,255,357	5,255,357	3,896,908	2,446,761	-	-	22,109,739
Process Costs		-	-	-	2,501,247	2,360,252	2,573,143	2,062,228	-	-	-	9,496,871
G&A		-	-	927,428	1,854,856	1,854,856	1,854,856	1,854,856	1,854,856	-	-	10,201,707
Direct Costs		-	-	1,380,829	22,461,369	22,824,900	21,934,045	11,836,229	4,388,842	-	-	84,826,213
Cash Cost per Ounce					860	958	804	506				843
Net Operating Costs		-	-	(1,380,829)	14,764,823	11,134,320	16,958,026	21,473,667	(4,388,842)	-	-	58,561,164
Add Operating Capital Back												-
Less Capital		-	(1,680,000)	(26,039,330)	(5,373,159)	-	-	-	-	-	3,327,500	(29,764,989)
Net Cash Flow per Year		-	(1,680,000)	(27,420,159)	9,391,664	11,134,320	16,958,026	21,473,567	(4,388,842)	-	3,327,500	28,796,175
Cash Position		-	(1,680,000)	(29,100,159)	(19,708,495)	(8,574,175)	8,383,851	29,857,518	25,468,675	25,468,675	28,796,175	28,796,175
Other Production Costs												
Royalties		305,000	55,000	55,000	3,695,641	1,930,772	798,542	196,115	-	-	-	7,036,070
Reclamation		-	-	-	-	-	-	85,000	120,000	225,000	400,000	830,000
Depreciation		-	-	-	-	-	-	-	-	-	-	-
Net Proceeds Tax	5%	-	-	-	760,539	460,177	807,974	1,067,311	-	-	-	3,096,002
Total Other Costs		305,000	55,000	55,000	4,456,180	2,390,950	1,606,516	1,348,426	120,000	225,000	400,000	10,962,071
Total Cash Cost of Production		(305,000)	(55,000)	(1,435,829)	10,308,643	8,743,370	15,351,510	20,125,241	(4,508,842)	(225,000)	(400,000)	47,599,093

STATEMENT OF CASH FLOW

Sales/Revenues		-	-	-	37,226,192	33,959,219	38,892,071	33,309,895	-	-	-	143,387,377
Direct Costs		-	-	(1,380,829)	(22,461,369)	(22,824,900)	(21,934,045)	(11,836,229)	(4,388,842)	-	-	(84,826,213)
Other Costs--Royalties, Net Proceeds, etc.		(305,000)	(55,000)	(55,000)	(4,456,180)	(2,390,950)	(1,606,516)	(1,348,426)	(120,000)	(225,000)	(400,000)	(10,962,071)
Cash Flow From Operations		(305,000)	(55,000)	(1,435,829)	10,308,643	8,743,370	15,351,510	20,125,241	(4,508,842)	(225,000)	(400,000)	47,599,093
Add Operating Capital Back		-	-	-	5,373,159	-	-	-	-	-	-	5,373,159
Capital Expenses		-	(1,680,000)	(26,039,330)	(5,373,159)	-	-	-	-	-	3,327,500	(29,764,989)
Free Cash flow		(305,000)	(1,735,000)	(27,475,159)	10,308,643	8,743,370	15,351,510	20,125,241	(4,508,842)	(225,000)	2,927,500	23,207,263
									14,091,956	NPV	5%	
										IRR	23%	

Table 22.2: Economic Summary

22.2 Sensitivities

Table 22.3 shows the project sensitivity to gold price, operating cost and capital cost. This is then shown graphically in **Figures 22.1** and **22.2**.

Table 22.3: Pine Grove Project Sensitivity

% of Base	NPV (000's 5%)	IRR	Price $/oz Au
Vary Gold Price			
80%	-$5,325.0	-2%	$1,140
86%	$468.2	6%	$1,225
90%	$4,216.1	11%	$1,280
95%	$8,985.4	17%	$1,350
100%	$14,092.0	23%	$1,425
105%	$19,198.6	29%	$1,500
110%	$23,964.7	35%	$1,570
116%	$29,411.8	41%	$1,650
120%	$33,497.0	46%	$1,710
130%	$43,028.1	57%	$1,850
140%	$53,240.0	67%	$2,000
Vary Operating Cost			
80%	$27,153.1	40%	$1,425
85%	$23,887.8	36%	$1,425
90%	$20,622.5	31%	$1,425
95%	$17,357.2	27%	$1,425
100%	$14,092.0	23%	$1,425
105%	$10,826.7	19%	$1,425
110%	$7,561.4	15%	$1,425
115%	$4,296.1	11%	$1,425
120%	$1,030.8	6%	$1,425
Vary Capital Cost			
80%	$19,371.0	35%	$1,425
85%	$18,051.2	32%	$1,425
90%	$16,731.5	29%	$1,425
95%	$15,411.7	26%	$1,425
100%	$14,092.0	23%	$1,425
105%	$12,772.2	21%	$1,425
110%	$11,452.4	19%	$1,425
115%	$10,132.7	17%	$1,425
120%	$8,812.9	15%	$1,425

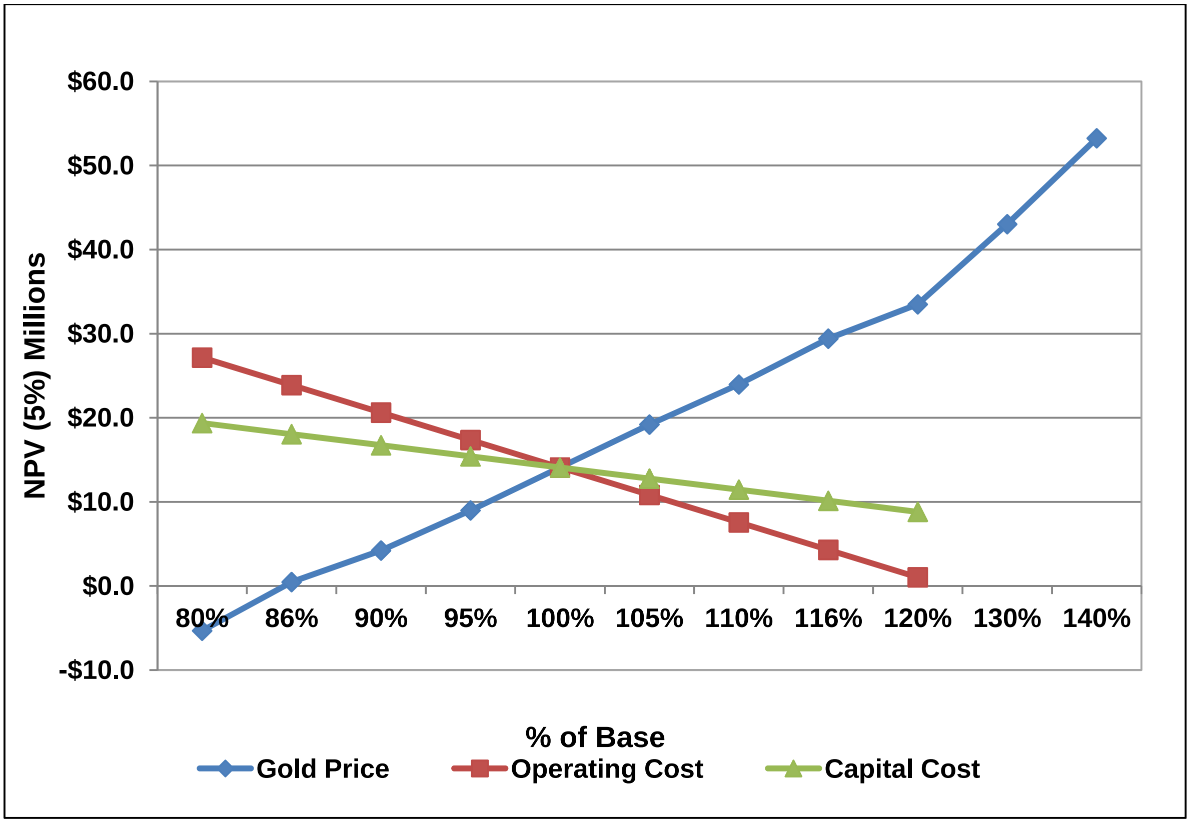


Figure 22.1: Pine Grove Project Sensitivity (NPV 5%)

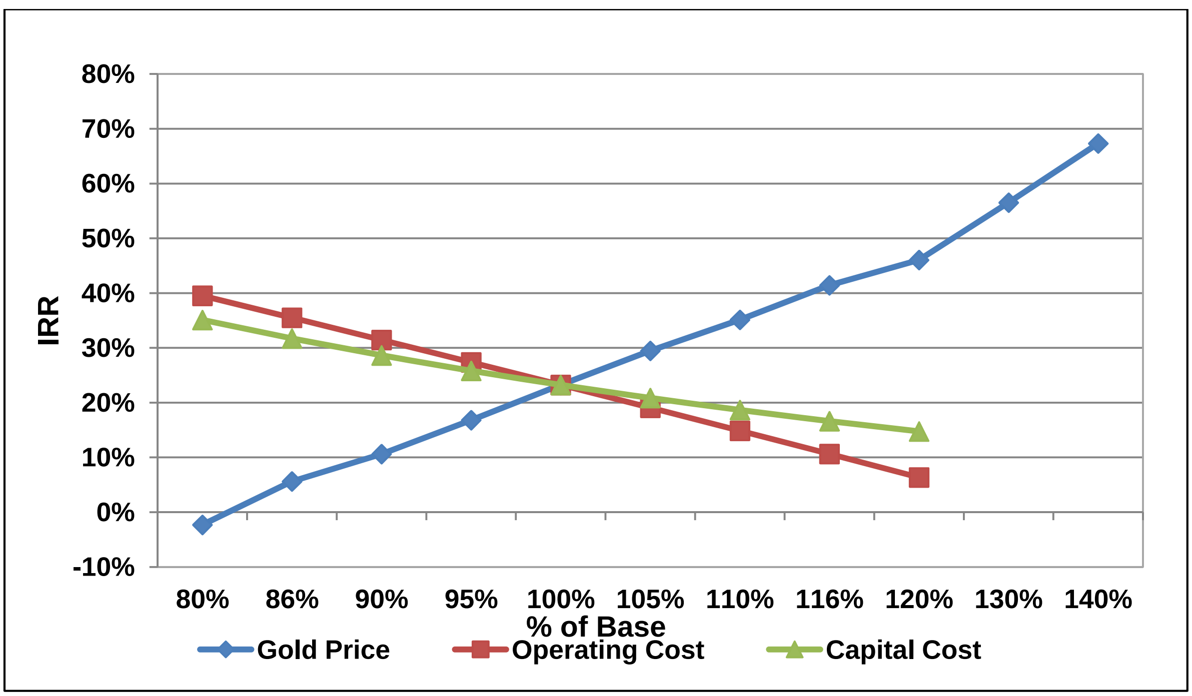


Figure 22.2: Pine Grove Project Sensitivity Internal Rate of Return

22.3 Royalties, Taxes

Table 22.4 shows a summary of the royalty payments to various entities. Although it is shown on the table, no part of the Harvest Claim Group Holdings fall into the area within the designed pits. A more detailed discussion of royalties and advanced royalty payments is included in section 4.3.

Table 22.4: Pine Grove Cash Royalties Included in Economic Analysis

Entity	Royalty	Royalty Calculated
Wheeler Mining Company (Wheeler Patented Claims)	NSR Production 3-7%	7%
Lyon Grove LLC (Wilson Patented Claims)	NSR Production 2.5%	2.5%
Lyon Grove LLC - Paid on Lincoln Claims - 6 mile area	NSR Production 5%	5%
Cavanaugh (Cavanaugh Claim Group)	NSR Production 1.5%	1.5%
Votipka (Harvest Claim Group)	NSR Production 5%	5%

Payments for the Nevada Net Proceeds Tax were included in the economic analysis. It is an ad valorem property tax assessed on minerals mined or produced in Nevada when they are sold. This tax is separate from, and in addition to, any property tax paid on land, equipment and other assets. It was calculated at a rate of 5%.

Some local property taxes were also included in the economic analysis. No attempt was made to calculate additional taxes based on income and loss.

23.0 ADJACENT PROPERTIES

Only one former mine is truly adjacent to Pine Grove. The Rockland Mine was active between 1870 and 1941, producing more than $500,000 and possibly as much as $7,000,000 of gold and silver at present prices (Hodgson, 1966). By 1870, a 10-stamp mill had been erected at Rockland but it burned down a few months afterwards (Lincoln, 1923). A 5-stamp mill was constructed for the Rockland Mine in 1902 and a 15-ton cyanide plant was added later. In 1907, a 60-ton dry crushing and leaching plant was constructed. The mine closed because of the Government Order suspending operations at the onset of World War II.

The ore was hosted in and near silicified rhyolite and diorite. Because mining ceased due to government order and not because the ore supply ran out, Hodgson estimated that as much as 500,000 tons of mineable ore was still available. Estimated value of the ore was $40 per ton in 1966.

The Rockland Mine is considered to be part of the Pine Grove Mining District in some reports (i.e. Lincoln, 1923, and Couch and Carpenter, 1943). However, because the style of mineralization is different than the style of mineralization in the historic Wilson and Wheeler mines, the Rockland Mine is no longer considered to be part of the Pine Grove District. Production statistics for the Pine Grove District (Rockland and Wilson) were reported by Couch and Carpenter (1943), as shown in **Table 23.1**.

Table 23.1: Pine Grove District Production by Year (Rockland and Wilson Combined)

Year	Tons	Gross Yield	Year	Tons	Gross Yield
1870	663	$ 27,912	1888	232	$ 3,984
1871	2,113	$ 35,279	1889	200	$ 1,131
1872	1,500	$ 24,097	1894	688	$ 13,625
1873	1,347	$ 21,381	1896	899	$ 21,603
1874	1,451	$ 25,246	1897	247	$ 5,363
1875	1,702	$ 32,050	1899	–	$ 28,386
1876	666	$ 16,288	1900	–	$ 9,885
1877	3,015	$ 85,310	1915	8,635	$ 69,618
1878	2,060	$ 48,411	1916	20,372	$ 140,556
1881	–	$ 75,590	1917	14,090	$ 82,897
1883	–	$ 6,437		**59,983**	**$ 778,734**
1887	103	$ 3,685			

Note: Table 24.1 is adapted from the Pine Grove table on page 92 of Couch and Carpenter (1943).

Couch and Carpenter (1943) also list production of properties reporting a total of $5,000 or more. The Pine Grove District is summarized in **Table 23.2**.

Table 23.2: Production of Properties Reporting a Total of $5,000 or more in the Pine Grove District

Property	Year	Tons	Gross Yield	
Dolores-Rockland	1870-1878	2,747	$	74,503
Interstate Mining & Development Co.	1933-1934	443	$	43,037
Midas Mine	1870-1878	336	$	8,883
Pittsburg-Dolores	1915-1917	42,597	$	263,071
Wheeler Mine	1873-1878	6,038	$	165,911
Wilson Mine	1870-1899	7,623	$	220,368
			$	**778,734**

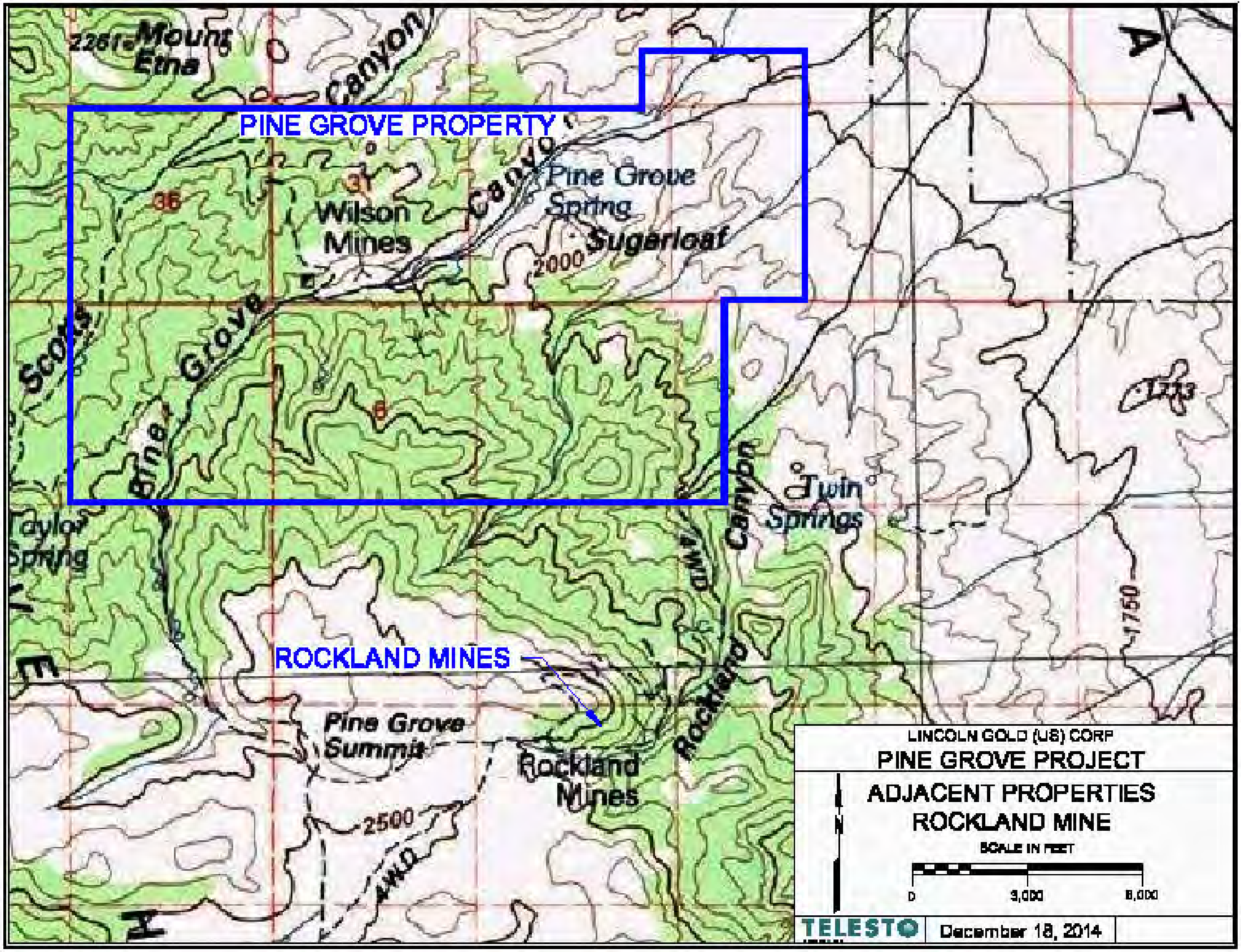
Note: Table 24.2 is adapted from the Pine Grove portion of the table on pages 94 and 95 of Couch and Carpenter (1943).

In 1981, the Rockland Mine was evaluated by J.P. Elwell Engineering, Ltd. of Vancouver, B.C. on behalf of Red Ledge Mining Corp. (Red Ledge) of Grass Valley, California (Elwell, 1981). However, it is not known if Red Ledge ever followed up on the exploration recommendations.

Lyon County has other mines which have produced in the past, but there are no other mines except Rockland which are adjacent to Pine Grove.

The Qualified Person has not verified the information contained in these publically disclosed reports and therefore the information in the reports is not necessarily indicative of the mineralization at the Pine Grove Project.

Figure 23.1: Adjacent Property – Rockland Mine Map



24.0 OTHER RELEVANT DATA AND INFORMATION

The Qualified Person knows of no other data or information as of the effective date of this report which is relevant or material to the Pine Grove Project.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 Database Integrity

Much work has been done to review and verify the integrity of the information reported in the electronic drillhole database provided by Lincoln. Rigorous comparison of the data has demonstrated that, in the opinion of the geology Q.P., the data is acceptable for the preparation of the resource estimate reported herein.

25.2 Gold Resources within a Designed Pit Shell

The Pine Grove Project does not contain mineral reserves as defined by CIM standards. This study is preliminary in nature and has used Measured and Indicated resources in the determination of the pit design. The reader is cautioned that inferred resources are considered too speculative geologically to have economics applied and there is no certainty that the economic results can be achieved. Only measured and indicated categories within the pit shells have been used in developing production schedules and preliminary cash flow analyses. Using a gold cutoff grade of 0.014 opt (0.481 g/t):

- A measured resource within a designed pit shell is estimated to contain 1,580,000 tons (1,433,000 tonnes) at an average grade of 0.057 opt (1.95 g/t) gold.
- An indicated resource within a designed pit shell is estimated to contain 647,000 tons (587,000 tonnes) at an average grade of 0.052opt (1.78 g/t) gold.
- The measured and indicated resource within a designed pit shell contains 2,227,000 tons (2,020,000 tonnes) at an average grade of 0.055 opt (1.90 g/t) gold.
- An inferred estimated resource contained within the designed pits is 88,000 tons (80,000 tonnes) at an average grade of 0.067 opt (2.29 g/t) gold.

25.3 Metallurgy

The potential recovery values from a future heap leach extraction process of material from each deposit are derived from drill core samples that are assumed to be representative of the deposit. In addition, it is also assumed that the drill core samples did not significantly age from the time of collection until metallurgical compositing and testing, which could affect the projected heap leach recovery values. The five recent cyanide leach column tests from composites of the 2008 drill campaign of material from the Wheeler and Wilson deposit provide the bulk of the 2010 metallurgical test data used in this report, with a weighted average gold recovery value of 77%, if crushed to 80% passing $^{3}/_{8}$ inch and heap leached for 150-170 days. A recovery value of 75% of the fire assay gold grade is estimated for this heap leach process if the mineralized material is crushed and agglomerated at 80% minus $^{3}/_{8}$ inch, and reflects the recovery basis for this preliminary economic analysis. Thus a discount of 2% of the gold recovery is incorporated into this report, which is typical for estimating field recovery from column test work.

The high reagent usage normally can be reduced in actual operations, but due to the presence of substantial leachable copper in the mineralized material, and various operational techniques used to optimize precious metals recovery, the reagent used and reported from the cyanide column leach tests have been discounted 41.5% (Cousins, 2009).

25.4 Geology

Statistical analysis of drillhole data has shown that there is a strong correlation between grade and structural preparation of host rocks, in particular, faults and fractures.

25.5 Economics

The preliminary economic assessment of Pine Grove concluded that the project has the potential to be economically viable based on the assumptions that were made. The project, at \$1425 per ounce gold price has an IRR of 23% and an NPV (5%) of \$23.2 million. Pre-production work will take a year to complete and the 2.9 million tons of waste stripping will be necessary to ensure a constant mineralized material feed to the crusher through the life of the mine. Maintaining a constant feed to the crusher is a consideration that deserves additional study. In order to maximize the crusher economics, the feed cutoff was dropped to 0.007 opt Au. This action improved the overall economics but may not be the best course of action.

26.0 RECOMMENDATIONS

26.1 Drilling and drillhole database

- Future angled drillholes with a potential for significant deviation should have downhole surveys performed.
- Ensure future drillhole collars are surveyed and location entered into drillhole collar database.
- Continue to document chain-of-custody and QA/QC protocols for all samples collected from future drilling programs.
- Infill and offset drilling around the Wilson deposit should be pursued.
- Deep drilling east of the Wheeler pit area should be performed to test for possible fault-displaced, down-dip resources.
- Review the potential for offset drilling around the Wheeler deposit.
- Perform condemnation drilling in proposed waste dump areas and under proposed leach pad.
    - In the proposed heap leach pad area, drill in an offset diamond pattern that matches the anisotropy of the variography for condemnation.
- Drill several deep holes (>1,000 feet) in the Wheeler deposit to condemn the area for backfilling with Wilson waste.

Estimated cost for drilling and drillhole database work: $ 300,000

26.2 Geotechnical

- Perform slope stability and rock quality testwork (RQD) to determine pit stability. Based on results, pit slopes may need to be redesigned.
- Bulk density testing should be done on all rock types representative of both waste and mineralized material.

Estimated cost for geotechnical work: $ 321,000

26.3 Exploration

- Expand drilling along structural trends and known outlying anomalous intercepts.
- Sample and survey existing waste dumps and any other surface areas of interest to determine suitability for heap leaching.

Estimated cost for exploration work: $ 200,000

26.4 Environmental Testing

- Acquire representative samples of waste and mineralized material for static and kinetic testing for rock characterization. These tests will be required for future permitting regarding acid rock drainage and the release of potentially harmful constituents into the environment.
- An analysis of material that would be used for final closure cover and growth media on the heap and waste dump.
- Establish monitoring wells upgradient and downgradient to establish baseline water quality data prior to development.

Estimated cost for environmental testing work: $ 60,000

26.5 Water Supply

- Determine water supply potential utilizing monitor well drilling information.
- Establish primary and secondary wellheads to support water demands for the Project.
- Sample all wells for baseline groundwater conditions.
- Model the water table to establish ground water gradient regime.
- Acquire additional water rights if necessary.

Estimated cost or water supply work: $ 122,000

26.6 Metallurgical and Processing Testwork

The following recommendations are made resulting from the review of the available metallurgical testing made available by Lincoln.

- Determine the effects of weather conditions and water requirements on-site (Clem, 1983).
- Collect representative fresh mineralized material samples from each mineralized material type for further testing.
- Conduct Mineral Liberation Analyses (MLA) on a split of the sample to identify mineralogy of mineralized material and gangue and the size of the minerals and representative mass fraction.
- Identify and track the concentrations of [As] in the samples for metallurgical testing as well as track the [As] in solutions.
- Track the sulfur in the metallurgical samples, both sulfide and sulfate. Try to correlate the [SS] vs. gold recovery and ratio in the samples.
- Determine the concentrations of Hg in each composite head sample on further metallurgical leach testing and follow up tracking of Hg in the pregnant solutions and tails, thus performing a metallurgical balance for Hg in the prospective process.

- Determine the concentrations of Co in each composite head sample on further metallurgical leach testing and follow up tracking of Co in the pregnant solutions and tails, thus performing a metallurgical balance for Co in the prospective process.
- Cyanide shake tests on all fire assay sample with a head grade of >0.01 ozAu/ton. Measure dissolved gold and copper on shake tests.
- Screen Fires on high grade (>0.05 ozAu/ton)
- Pilot Plant or bench testing test for free gold to determine the size and percentage of gold that is freely liberated for gravity concentration/flotation)
- Identify the reason for the large variation in lime demand due to metallurgy and mineralogically.
- Conduct agglomeration tests to optimize cement addition.
- Representative samples of high grade material with screen fire or pulp and metallic fire assays need to be conducted to determine the quantity and size distribution of free gold.
- Representative samples of high grade material need for gravity and flotation concentrate tests with bottle roll testing on concentrate and tails to be conducted. Data need to be collected on size distribution, grade for feed and tail samples, metallurgical balances on each unit process, reagent concentration and kinetics on the concentration and leaching processes.
- Develop a water balance for the location to determine the quantity of water needed for the mine, which includes the extraction process, dust control for the mine and crushing facility.
- Develop the wells on the property that will be used for the extractive process.
- Additional column tests should be done in the future to better determine actual precious metal recovery rates using on-site water on a representative sample and may lead to optimization of the reagent usage and forecast economics.
- Examine the veins and veinlets, gold bearing structures in regards to mine bench height to optimize the ore control and ore grade reporting to the extractive process. "Veins are a few mm to less than a m thick and are hosted by an extensional shear zone. Veins are strongly structurally controlled, are sheeted, and are oriented parallel to the shear zone host. Veins contain native gold, pyrite, chalcopyrite, and pyrrhotite. Gangue consists of quartz, and sulfides comprise less than 10 percent of the veins." (Stone, 2007)
- A.A. for copper
- Investigate the potential to recover the copper from process streams and determine if this process is economic. Investigate the potential to recover the copper from process streams and determine if there is a more economic metal recovery process. The Merrill Crowe process theoretically can recovery copper in addition to gold and silver, but the cost of Zn is high for copper recovery.

- Appropriate metallurgical drillhole samples should be obtained for future column testwork.
- Collect representative samples of the waste dumps to determine recovery (Macy, 1987).
- A sampling program is recommended for the identified tailings deposits. If possible, this sampling can be used to generate samples for preliminary metallurgical testing. Preliminary metallurgical testing recommended for these samples includes the same scope of bottle roll testing recommended for the drill core composites, with tracking of mercury along with gold and silver (McPartland, 2009).

Estimated cost for metallurgical test work: $ 150,000

26.7 Permitting

- Wait until spring, 2012 to perform archeological and biological studies, after submission of the PoO.
- Identify water source for the project.

Estimated cost for permitting work: $307,000

Total Estimated Costs for Recommended Work:

Drilling and Drillhole Data Base	$ 300,000
Geotechnical	$ 321,000
Exploration (to be determined)	$ 200,000
Environmental Testing	$ 60,000
Water Supply	$ 122,000
Metallurgical and Processing Testwork	$ 150,000
Permitting	$ 307,000
TOTAL:	**$1,460,000**

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