MINE DEVELOPMENT ASSOCIATES

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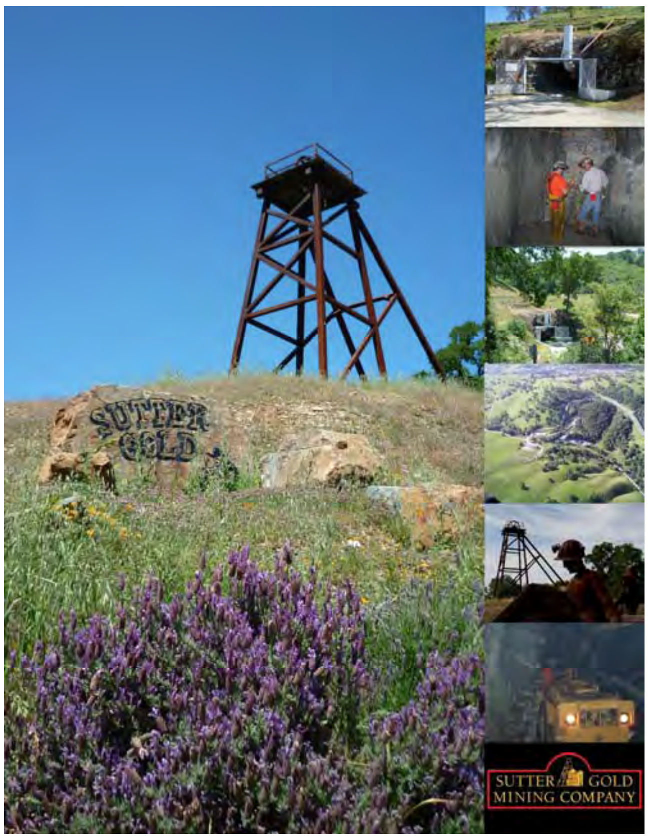
Updated Technical Report on the Lincoln Mine Project, Amador County, California



Prepared for

Sutter Gold Mining Inc.

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CONTENTS

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TABLES

FIGURES

APPENDICES

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

Mine Development Associates (“MDA”) has prepared this updated technical report on the Lincoln Mine project, located in Amador County, California, at the request of Sutter Gold Mining Inc. (“SGM”). The purpose of this report is to:

1. Update the 2011 PEA study on the Lincoln-Comet deposit to reflect work completed by SGM between 2012 and 2014;

2. Estimate a resource for the Keystone deposit;

3. Review the impacts of the 2012 SGM drilling on the 2011 Lincoln-Comet resource estimate prepared by MDA in accordance with the Canadian Securities Administrators’ National Instrument 43-101 (“NI 43-101”).

This report was written in accordance with disclosure and reporting requirements set forth in the Canadian Securities Administrators’ National Instrument 43-101(“NI 43-101”), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum’s “CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines” (“CIM Standards”) adopted by the CIM Council on May 10, 2014. .

The Lincoln Mine project is located in central California, about 45mi east-southeast of Sacramento in western Amador County between the towns of Amador City and Sutter Creek. The project lies in the foothills of the western slope of the Sierra Nevada in the central part of the historic Mother Lode gold belt. The property currently consists of 47 individual parcels that include 711.08 acres for which SGM owns or leases mineral rights and 173.21 acres of surface rights. SGM controls the properties through outright ownership as a result of purchase and through lease agreements. In addition to a number of historic producing mines, the project contains two deposit areas which will be the primary focus of this Technical Report: the Lincoln-Comet and the Keystone.

The Lincoln-Comet and Keystone deposits are Mother Lode-style gold deposits hosted within near-vertical, 1 to 4ft-wide mesothermal quartz veins. The Lincoln-Comet resource occurs over a 3,000ft strike length and has an 800ft downdip extent. SGM proposes to exploit the Lincoln-Comet resource using underground mining methods that include cut-and-fill stoping and access through multiple declines. Gold recovery will be primarily through gravity separation, though a flotation circuit will recover the remaining fine gold and also remove the arsenic from the mine tailings.

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SGM commenced construction of a 210 ton per day mill on the Lincoln mine property during 2012, and completed about 3,300 ft of underground development on the Lincoln-Comet deposit during the period of late 2012 through early 2014. A total of about $22 million was spent on the project during this period. Approximately 3,100 tons of low grade material has been stockpiled near the mill. About 1,000 tons of low-grade material was processed during 2013-2014, but the rod mill produced excessive fines and the rate of processing material through the grinding circuit was much lower than expected. Gold recovery was very low due to the excessive fines. SGM has developed plans to mitigate these problems which are incorporated in this PEA study.

The Keystone resource area lies 2,000ft north of the Lincoln-Comet resource. The current resource occurs within two distinct veins that have a maximum 1,200ft strike length and an 800ft down-dip extent. Both veins are associated with historic underground development.

The Lincoln Mine project is subject to federal, state, and local environmental regulations and permitting requirements. SGM has obtained the major permits and approvals needed for the development of the Lincoln-Comet resource area, including an environmental review, conditional use permit, water discharge requirements, surface mining permit, and reclamation plan and bond. SGM has obtained numerous other operating permits and approvals also needed for the project, with additional operating permits and approvals still to be obtained.

The project coordinates are truncated California State Plane – Zone 2 coordinates using the NAD 27 datum.

1.2 Geology and Mineralization

The Lincoln Mine property lies in the western foothills of the Sierra Nevada, which are underlain by Carboniferous and Jurassic metasedimentary and metavolcanic rocks with numerous intrusions of basic and ultrabasic rocks, many of which are serpentinized. These metamorphic rocks form the basement into which the huge Sierra Nevada granodioritic batholith that dominates the range was intruded during Jurassic to Cretaceous time.

The historic Mother Lode is a 120mi long, 1 to 4mi wide, system of linked or *en echelon* gold-quartz veins and mineralized schist that is hosted by Jurassic metamorphic rocks of the western foothills. The most productive portion of the Mother Lode was a 10mi long portion in Amador County, within which the Lincoln Mine property makes up a 3.2mi long segment between Amador City and Sutter Creek. The Melones fault zone, separating Paleozoic rocks on the east from Jurassic rocks on the west, is a regional structure located about 0.5mi east of and parallel to the Mother Lode. In Amador County, the Gold fault zone hosts all of the large productive mines and is a braided corridor of high strain that is a branch from the Melones fault zone.

The Lincoln Mine property is underlain by northwest-striking, steeply dipping metasedimentary and metavolcanic rocks of the Late Jurassic Mariposa Formation that lie west of the Melones fault zone. Most of the current Lincoln Mine resource is hosted by basaltic to andesitic metavolcanic flows and tuffs, with the southeastern part of the resource lying within an overlying metavolcaniclastic and epiclastic unit within the Mariposa Formation. Nearly all significant gold mineralization on the Lincoln Mine property is related to deformation or dislocation along contacts between metasedimentary and metavolcanic rocks where the contacts are faulted. The Comet and Lincoln mineralized vein zones trend

N30°W and generally dip steeply west at an average of 70° while the Keystone vein zones also trend N30°W but dip to the east at an average of 60°.

The gold-quartz mineralization on the Lincoln Mine property is of orogenic (mesothermal) type, in which structurally controlled gold mineralization occurs as vein quartz filling dilatant zones and as sulfidized replacements in altered wall rocks. Gold-quartz-ankerite veins, generally 1 to 4ft in width, cut the Mariposa Formation and are controlled by shear zones. Gold emplacement and localization within the quartz veins/structures is primarily related to late shearing of the quartz. Zones of barren quartz can occur where the late shearing is absent. The veins contain free gold and 1 to 2% accessory sulfides. In the Lincoln-Comet deposits, about 20% of the gold occurs as coarse grains up to 1/8in. in size.

Gold mineralization in the project area also occurs as “sulfide replacement mineralization” hosted within hydrothermally altered metavolcanic rocks. The sulfide replacement mineralization, known historically as “gray ore” in the Mother Lode belt, consists of strongly altered metavolcanic rocks cut by varying amounts of thin quartz veining. This type of mineralization makes up approximately 5% by volume of the Lincoln-Comet mineralization. The alteration consists of complete or nearly complete replacement of the metavolcanic rocks by ankerite, albite, and sericite plus 2 to 3% fine-grained sulfides. The sulfide replacement mineralization generally has higher total sulfide content but lower average gold grades than the quartz veins. The sulfide replacement mineralization at the Lincoln Mine property universally occurs where vein structures bend, propagating vein splits, and can constitute large-tonnage, bulk-minable deposits with maximum widths of 20ft at Lincoln-Comet.

1.3 Exploration and Mining History

The Lincoln Mine project lies within the most productive portion of the historic Mother Lode, which has produced 13.6 million ounces in lode gold since 1849. Eight major past-producing Mother Lode mines lie within the Lincoln Mine project area. Together they accounted for 3.4 million ounces of gold production until 1953, or about 25% of the entire Mother Lode gold production (Table 1.1). Several reports indicate that none of these eight mines had closed due to lack of “ore” but rather because of complex property ownership, litigation, insufficient land for tailings disposal, lack of capital, or regulations of World War II. Several reportedly had large tonnages of mineralization in place at the time they closed. Despite this historic production from old mines on the current Lincoln Mine property, there seems to have been little historic mining and exploration in the Lincoln-Comet resource area, probably due to the absence of surface vein outcroppings in this area. However, substantial mining activity took place along the strike of the Lincoln and Comet veins, both north and south of the resource area.

Modern exploration of the Lincoln Mine property effectively began in 1983, when Callahan Mining Corporation (“Callahan”) began acquiring properties in the vicinity of the inactive Lincoln mine. Through drilling by Callahan and by a joint venture of Callahan and Pancana Minerals, Inc. (“Pancana”), a resource was found in the Lincoln zone that represented the first significant new gold discovery not related to past-producing mines that had been made in the Mother Lode gold belt since the 1940s. Callahan’s drilling also defined a gold resource along the Medean vein near the Keystone mine located north of the Lincoln mine.

Meridian Gold Company (“Meridian”) bought the Lincoln-Comet property in 1987 and the Keystone property in 1988 from Callahan-Pancana. Meridian’s drilling from 1987 to 1990 continued to define the Lincoln resource and discovered the Comet zone and a deeper zone in the Keystone 5 vein beneath the

Comet. In 1989, Meridian began underground development to explore the Comet zone by driving the 2,885ft-long Stringbean Alley decline.

Table 1.1 Summary of Historic Gold Production from Major Mines of the Lincoln Mine Project Area

Mine (North to South)	Reported Gold Production ($)	Calculated Gold Production (ounces)	Years of Production	Notes
Bunker Hill	$5,142,382	250,000	1853-1934	1
Original Amador	$3,500,000	169,500	1852-1937	2
Keystone Consolidated				
Keystone	$18,778,000	889,300	1851-1920; 1935-1942	3
South Spring Hill	$1,953,749	94,600	1878-1883; 1883-1894	4
Medean	$156,093	7,500	1894-1899	4
Talisman	$402,000	20,000	1854-1876; 1879-close	4
Wabash		8,000		5
Lincoln Consolidated				
Lincoln	$2,200,000	106,500	1851-close	6
Wildman & Mahoney	$3,270,269	158,200	1851-1887; 1887-1894; 1894-1901	7
Central Eureka (including Old Eure	$36,000,000	1,672,000	1852-1952	8
TOTAL	**$71,402,493**	**3,375,600**		

Notes:

1. Reported Production is from Logan (1934) and calculated production is based on $20.65/oz average gold price
2. Reported production from Carlson & Clark (1954) and Clark (1970) and calculated production assumes $20.65/oz
3. Reported production from Keystone Mine Vertical Cross Section (1929) and Carlson & Clark (1954); calculated production assumes $20.66/oz through 1920 and $34.42/oz beginning in 1935 (averages)
4. Reported production from Keystone Mine Vertical Cross Section (1929); calculated production assumes $20.66/oz average
5. Payne (2008)
6. Reported production from Bowen & Crippen (1948, Carlson and Clark (1954) and Clark (1970); calculated assumes averge $20.65
7. Reported Production is from Logan (1934) and calculated production is based on $20.65/oz average gold price
8. Reported form Carlson & Clark (1954), Wagner (1970) and Clark (1970); calculated production assumes 90% at 20.67/oz and 10% at 34.42/oz

FMC Gold Company purchased Meridian in 1990 and later that year sold the Lincoln and Keystone properties to Seine River Resources Inc. ("Seine River"), who was in a joint venture with U.S. Energy Corp. Crested Corp. joined the joint venture later in 1990. In 1994, U.S. Energy and Crested Corp. acquired 100 percent interest in the venture and incorporated Sutter Gold Mining Company to operate the project. Sutter Gold Mining Company is the wholly owned subsidiary of Sutter Gold Mining Inc. In 2004, SGM leased the Central Eureka mine property, adding a significant extension to known mineralized zones at the south end of their property. In 2009, SGM leased the Original Amador mine and Bunker Hill mine properties, adding a significant extension to known mineralized zones at the north end of their property

Modern exploration since 1983 on the Lincoln Mine property has been limited to the gold-quartz veins in the area of the historic Spring Hill, Talisman and Medean shafts north of the Stringbean Alley decline and in the Lincoln and Comet zones.

Much of SGM's initial work on the property involved permitting, which continues along with site design and property consolidation. In 2006-2007, SGM drilled in both the Lincoln-Comet and Keystone areas. Since then, work has continued on structural geology interpretations and development of an exploration model for the project as well as compiling a database of fire assays from drill core and underground development samples.

In 2008-2009, SGM collected a bulk sample of the Lincoln-Comet mineralized zones that was sent to McClelland Laboratories Inc. ("McClelland") for gravity/flotation testing. Rougher tailings generated from the sample at McClelland were sent to Golder Paste Technology and utilized for paste backfill testing. SGM conducted an underground sampling program in 2009 that included face sampling, long-hole percussion drilling into the face, channel sampling on the backs and the faces, muck sampling from the LHD buckets, and bulk sampling of each round.

During the period of 2012 through 2014 the company constructed a 210 ton per day mill, a backfill plant, and completed about 3,300 ft of development. About 1,000 tons of material was processed by the plant, prior to stopping development and processing due to mill issues. During 2014, a number of metallurgical consultants visited the site and suggested revisions to the mill to improve the grinding circuit and to reduce excessive fines in the circuit. The company has received proposals to make the suggested revisions, which are incorporated in this report.

1.4 Drilling and Sampling

This technical report only describes modern drilling since Callahan's acquisition of the project in 1983. Exploration and definition drilling from 1983-2013 of 9 reverse circulation ("RC") and 272 diamond core holes (from surface and underground) has totaled approximately 112,754ft (see Table 1.2). This total includes the 2 surface RC holes, 87 surface core holes, and 107 underground core holes used in the Lincoln-Comet resource estimate and the seven RC and 23 surface core holes used in the Keystone resource estimate.

Table 1.2 Modern Drilling on the Lincoln Mine Project

Company	Date	Area	Type	# of Holes	Depth (ft)	Total # of Holes	Total Depth (ft)
Callahan	1983-1984	Keystone	RC	7	4,416	24	13,147
	1983	Keystone	core	2	1,467		
	1983	Lincoln-Comet	RC	2	466		
	1984-1985	Lincoln-Comet	core	13	6,798		
Callahan-Pancana	1986	Lincoln-Comet	core	15	9,742	15	9,742
Meridian	1987-1990	Lincoln-Comet	core	59	30,334	144	57,788
	1990	Lincoln-Comet	core U/G	74	18,273		
	1988-1989	Keystone	core	11	9,182		
Sutter Gold	2006	Lincoln-Comet	core U/G	33	9,127	43	19,546
	2006-2007	Keystone	core	10	10,419		
Project Drilling (used in resource estimates)	**1983-2007**	**all**	**RC**	**9**	**4,882**	**226**	**100,222**
			core	**110**	67,940		
			core U/G	**107**	27,400		
Sutter Gold	2012	Lincoln-Comet	core	26	10,244	55	12,532
			core U/G	29	2,288		
Total Project Drilling	**1983-2012**	**all**	**RC**	**9**	**4,882**	**281**	**112,754**
			core	**136**	**78,184**		
			core U/G	**136**	**29,688**		

The project total in Table 1.2 includes the 2012 SGM Lincoln-Comet core holes (26 surface and 29 underground) drilled after the completion of MDA's 2011 resource estimate. The total 2012 drill footage represents approximately a 12 percent increase in drill footage from the 2011 drilling used in the resource estimate. MDA reviewed the 2012 drill data and believes the drilling substantially supports the 2011 estimate. Though the drilling and underground development did locally extend and expand the high-grade gold zones, this work did not change the resource in a material way.

Before underground exploration began in the Comet zone, many of the surface drill holes in the zone were drilled from the eastern side of the deposit, plunging steeply westward, in the mistaken belief that the principal mineralized veins in the Comet zone dipped steeply eastward. This resulted in holes drilled from the surface intersecting westward-dipping veins at a very low angle, making estimates of the true thickness of vein intercepts problematical. A further challenge in drilling at the Comet zone is the fact that the property is narrow in this area, preventing surface drill holes from being stepped back an adequate distance from the target.

All of the Keystone holes are drilled towards the west targeting the eastward-dipping mineralized veins.

Callahan, Callahan-Pancana, and Meridian had selectively sampled the drill core, using visible mineralization or alteration as a guide, with particular emphasis on quartz veins and altered wall rock. Sample intervals of 4 to 5ft were typical in long intervals of similarly altered wall rock, with shorter intervals in mineralized intervals selected based on visible geological differences in the core. In much of the core sampling, the smallest unit of measurement used was 0.5ft. Core recovery by these operators was generally good (90% or better), except in the upper weathered zones of drill holes. The few rotary RC holes were apparently continuously sampled, routinely on 5ft intervals, reducing to 2.5ft intervals in visibly altered or mineralized rock.

Callahan and Callahan-Pancana used four different laboratories for their analyses, with Shasta Analytical Geochemistry Laboratory ("Shasta") and Barringer Laboratories Inc. ("Barringer") doing most of the work. There is no information on details of analytical procedures. Callahan and Callahan-Pancana did extensive check assaying of mineralized intervals. Meridian used Barringer and ALS Chemex ("Chemex") as their principal labs. Chemex analyzed 30g samples using fire assay with an atomic absorption finish; samples with greater than 20g Au/tonne were re-analyzed on a duplicate pulp using fire assay with a gravimetric finish. No check assaying was conducted on the Meridian holes. The historic holes suffer from a lack of external quality assurance/quality control ("QA/QC") measures common today but not common at that time.

For SGM's 2006-2007 drilling, sample intervals for mineralized core ranged from 3.0 to 3.3ft where practical, with the minimum being 2.9ft. This was designed to ensure an adequate sample size was available for multiple 500g or 1,000g screened metallics fire assays. For veins or sulfide replacement intervals greater than 3ft in width, the sample interval may be up to 4.5ft long in order to maintain sample size and not introduce undue wall-rock dilution. For quartz intervals longer than 4.5ft, the interval was broken into two equal parts. Samples of core containing visible gold, significant arsenopyrite, fault-bounded ribboned or banded vein quartz, or strong sulfide replacement mineralization, were analyzed by screened metallics fire assay. All other pyritic mineralized rock, phyllonite, and altered rock were analyzed by one-assay ton fire assay. American Assay Laboratories ("American Assay") performed the analyses for SGM, including all replicate fire assaying utilizing an approved method of blind re-submission.

SGM's QA/QC program included use of blanks, three different reference standards created from dump material collected from the Lincoln Mine property, and a replicate assaying program. Analytical results for the standard samples were highly variable, likely due to the "nugget" character of the mineralization, rendering this material unsuitable for use as reference standards.

1.5 Resource Database and Data Verification

The Lincoln Mine project database used for the Lincoln-Comet and Keystone resource estimates contains collar information on 226 drill holes and 778 individual underground channel samples. The underground channel samples are included in the database as short, horizontal "drill holes." Where two or more channel samples form a continuous sequence of samples taken across a face, these samples are linked together with each sample representing a "downhole" interval. As a result, the database contains 435 channel-sample collars.

The project database contains gold values for 8,814 sample intervals of which 7,343 are within Lincoln-Comet resource model area and 1,471 are within the Keystone resource model area. Due to a lack of data verification procedures and uncertain precision of the sample intervals, none of the RC assay data is included within the current database.

The majority of underground samples and many of the mineralized drill samples were re-assayed, often multiple times. Due to the "nugget" character of the mineralization and apparent sub-sampling variability, the average value is considered to be more representative of the sample interval than the single initial assay value. The gold data within the database, therefore, are often calculated values determined by averaging the often multiple analyses completed on each individual sample.

The blank data indicate some minor contamination associated with high-grade coarse gold samples, though the level of contamination is low and the risk in using the assay data for resource estimation is considered minimal. Various verification procedures were conducted on the Lincoln Mine drill hole and underground sample data to be used in the current resource estimates. These procedures included an audit of all historic and SGM data, a review and analyses of much of the QA/QC data, and core recovery versus gold grade studies.

The database audit includes a detailed audit and reconstruction of all available underground chip sample data. Only a few significant errors in the database were noted and corrected. MDA considers the project database to be adequate for use in the development of a classified resource estimate and for further mine planning.

There is limited blank sample and no acceptable standard sample quality control analyses on the project assay data. A detailed gold-grade reproducibility study indicated high variability in gold grades within the vein material, most likely due to the presence of coarse gold or possibly to gold occurring in coarse clots. This high variability occurs at all sub-sample stages from pulps all the way up to, it has been proposed, a macro or mining-round scale within and along the mineralized veins. The estimation of a *locally* accurate resource will, therefore, be difficult to achieve due to this inherent high sample-grade variability. Moderate to high risk is imparted from using assay values that are potentially not representative of the localized volume of rock.

1.6 Metallurgical Testing

SGM provided approximately 23 metallurgical testing, analysis, processing, piloting and engineering reports and studies to Allihies Engineering Incorporated of Butte, Montana and Dr. Corby G Anderson, QP CEng FIMMM FIChemE, for a review. This task was undertaken in detail and resulted in an affirmation that previous work has been of a sufficient quality and quantity necessary to support a Preliminary Economic Assessment Technical Report in accordance with the requirements of NI 43-101. Summary results of metallurgical testing are presented in Table 1.3.

Table 1.3 Metallurgical Property Summary for the Lincoln-Comet Resource Material Testing

	Source: Hazen, 1989		Source: McClelland, 2009 (McPartland, 2009)	
Criteria	Rod Mill	Ball Mill		
Bond Work Index	11.4	12.4-12.9		
Grind Size			P80-100 mesh	
Head Grade (oz Au/ton)			0.24	Calculated from concentrate and tails analysis due to head-grade sampling issue
Gravity Recovery (% Au)			82.1	Centrifugal concentrator followed by hand panning (due to sample size)
Gravity Concentrate Grade (oz Au/ton)			72.87	
Flotation Recovery (% Au)			15.8	Combined cleaner concentrate and tails
Flotation Concentrate Grade (oz Au/ton)			3.97	
Total Recovery (% Au)			97.9	
Concentration Ratio			1:48	Calculated as 2.1% of feed weight recovered

Further, after a review of the proposed gold mill design by Paul E. Danio & Associates and a corroboration of this document with existing metallurgical studies to date, Allihies confirms that these proposed designs and economic estimates are now appropriate as a preliminary conceptual design and preliminary estimate based on the current data available. Allihies does not confirm or take responsibility for, or confirm, any past, current or future operations and any detailed designs. The mill designed by Danio was constructed during 2012.

1.7 Geologic Model and Mineral Resource Estimation

1.7.1 Lincoln-Comet

Gold mineralization within the Lincoln-Comet resource area is characterized by sheared quartz veins, containing free gold and 1 to 2% accessory sulfides, hosted within a graben of metavolcanic rocks. The gold-quartz veins branch and anastomose along the 3,000ft length of the Lincoln-Comet resource, with the strongest gold mineralization often localized within distinct dilation zones along the veins or at structural/vein intersections. Within the higher-grade portions of the Lincoln and Comet zones, the west-dipping veins often terminate against shallow, east-dipping fault/vein structures which serve as structural traps for mineralization. The gold has a strong nugget character, being highly erratic in grade both on a sample scale and along strike within the individual veins. Gold grades of >1oz Au/ton can quickly transition to <0.1oz Au/ton over just a few feet along strike, while duplicate underground sampling has shown consistent assay differences of over 100%.

The geologic model is based on 57 cross sections spaced 50ft apart along a N30°W axis. The cross sections are oriented perpendicular to the general strike of the deposit. All significant structures and associated veins are modeled, resulting in a total of 38 unique mineralized veins within the Lincoln-Comet resource area. Many of the veins have a limited strike and/or dip extent, but some veins, such as veins 42, 50, and 51, extend for much of the full length of the resource area. The veins are dominantly steeply west-dipping, though shallow east-dipping veins are modeled in the Comet zone (veins 20, 23, and 61) and in the Lincoln zone (vein 9). In general, vein widths range from 1 to 4ft, though vein thickness often increases to a maximum of 15 to 20ft at the top of the west-dipping veins where they intersect, and often terminate, at the east-dipping vein structures.

There are a total of 33 density measurements on various lithologies from within the Lincoln-Comet project area, though only two samples are from within the resource area and have known locations. A single density value (12.0 ft^3/ton) is used within the Lincoln-Comet resource model due to the scarcity of data and the difficulty in correctly estimating density within highly variable mineralized structures.

The geologic cross-sectional model was used as a base and guide for the gold mineral model. The underground workings were also plotted on cross-section to guide the gold model. Quantile plots of gold were made to help define the natural populations of metal grades to be modeled on the cross sections. The quantile plots, along with additional statistical analyses, indicated that the gold mineralization can be modeled using three mineral domains. The low-grade gold domain (domain 100) is characterized by a range of grades of ~0.01oz Au/ton to ~0.07oz Au/ton and generally represents mineralization associated with weak veining and/or shearing either in the wallrock outside the primary vein or within the structures at depth or along strike away from the center of the deposit. The mid-grade gold domain (domain 200) is characterized by a range of grades of ~0.07oz Au/ton to ~0.25oz Au/ton and generally represents gold mineralization associated with increased shearing and/or sporadic coarse gold deposition within or along the immediate boundaries of the mineralized veins. The high-grade gold domain (domain 300) is defined by grades generally exceeding ~0.25oz Au/ton that are associated with increased shearing and coarse gold deposition within the high-grade core of the mineralized veins.

Color-coded assays corresponding to population breaks indicated by the quantile plots, along with the geologic cross-sectional interpretation, were plotted on cross sections and were used in the creation of the gold mineral domains. Each vein is considered a unique entity for sample coding and estimation purposes, so unique mineral domains were created for each vein. Using the cross-sectional interpretations as a framework, level plans of the gold domains were created at a 10ft spacing. The 10ft-spaced level plans were 3-D rectified to fit the drill and underground sample data, and Surpac mining software was used to code domain percentages into the block model.

The cross-sectional gold mineral domains were used to code the drill assays and underground samples. Twenty samples were capped after completing a statistical analysis of the coded samples along with a spatial analysis of the individual domains. Capping grades, for those samples which were considered to be statistical and/or spatially anomalous, ranged from 0.25oz Au/ton in the low-grade domains up to 6.0oz Au/ton in one of the individual high-grade domains. The resulting assay database used in the estimate contains gold values up to 11.19oz Au/ton. Once the individual gold samples were capped, they were down-hole composited into 5ft composites honoring all mineral domain contacts. No minimum length restrictions were imposed on the composites, and length-weighted composites were used in the estimation.

The model used 10ft by 10ft by 1ft-wide blocks with the long dimensions oriented N30°W. The block dimensions were chosen to minimize dilution for underground mining of a deposit of this kind.

Following compositing and the previously described statistical analyses of those composites, correlograms were constructed in multiple directions for all domains together. The estimation criteria were, in part, defined by these correlograms and, in part, by attempting to honor understood geologic controls and distributions. All gold domains have the same estimation parameters, which include a 50ft first pass, a second 250ft pass, and a final pass that filled the respective domains. All searches were isotropic, though the individual vein domains spatially controlled the estimation, which resulted in very planar search ellipses oriented along the general strike and dip of the veins. Estimation within each mineral domain used only those composites coded to that respective domain. Inverse-distance estimation was chosen as the base case, while estimates were also made by nearest neighbor and Kriging. The latter two were used as checks on the given estimate.

Resource classification was determined using distance to the nearest sample, number of samples, geologic confidence, and mineral domain continuity. The samples used for the classification criteria stated are independent of the modeled domains. The low-grade estimated blocks were not included in the resource, nor were the low-grade composite data used in the classification criteria, due to the erratic and likely sub-economic nature of the mineralization.

There are only Indicated and Inferred resources within the Lincoln-Comet deposit. There are no Measured resources associated with the Lincoln-Comet deposit due to a) a scarcity of density measurements, b) significant mineral variability leading to uncertainty in grade estimation, and c) some spatial uncertainty in the geologic model. Indicated resources are spatially associated with underground development and/or tightly-spaced drill information.

The reported resource is given in Table 1.4 The resource is reported at a 0.12oz Au/ton cutoff gold grade that is reasonable for deposits of this nature and for the expected mining conditions and methods. The stated resource is undiluted and includes just the mid- and high-grade domain-coded blocks.

Table 1.4 Lincoln-Comet Reported Resource

Lincoln-Comet Reported Resource				
Classification	**Au Cutoff** (oz Au/t)	**Tons**	**Grade** (oz Au/t)	**oz Au**
Indicated	0.12	152,000	0.401	61,000
Inferred	0.12	506,000	0.254	128,000

Numerous checks were made on the Lincoln-Comet resource model including: 1) the geologic model, including mineral domains, drill-hole assays and geology, topography, sample coding, and block grades with classification were plotted and reviewed for reasonableness, 2) cross-section volumes to block-model volumes were checked, 3) a polygonal model was made with the original modeled section domains, and 4) nearest-neighbor and Kriging models were made for comparison. The resource estimate is deemed reasonable, honors the geology, and is supported by the geologic model.

No mineral reserves have been identified on the project and mineral resources that are not mineral reserves do not have demonstrated economic viability.

1.7.2 Keystone

Resource estimation using a cross-sectional polygonal model was chosen for Keystone due to the limited drill data and the expected Inferred-only classification. The stated resource is undiluted.

Mother Lode-style Gold mineralization within the Keystone deposit is localized within two north-northwest-trending structural zones: the West Contact zone on the west and the Medean zone on the east. The mineralization style is similar to the Lincoln-Comet mineralization in that the gold-bearing veins, usually 1 to 4ft in thickness, occur as fissure veins within the structural zones. A pervasive fault overprint is common, and gold mineralization appears to coincide with the late faulting. Mineralized quartz veins are often bounded by fault slip planes that generally define one or both vein walls, and sheared, ribboned vein-quartz exceeding 1ft in width is commonly associated with the higher-grade mineralization.

The “K5” structure/vein is the dominant vein in the West Contact zone. Smaller structures/veins (“K22 through “K26”) occur within the footwall and hanging wall of the K5 structure. Some, and possibly all, of these mineralized veins were exploited by the historic South Spring Hill mine.

The “K13” structure/vein is the footwall vein of the Medean zone and a number of smaller structures (“K16” through “K20”) occur within the hanging wall, up to 200ft to the east of the K13 vein. These mineralized veins appear to have been exploited by the Medean mine and northern extension of the Talisman mine.

Cross-sections looking N30W and spaced at 100ft intervals were created across the Keystone deposit area. Drill assay data along with the designated vein intervals were plotted on the cross-sections. After reviewing the 2008 Keystone resource model, and then evaluating the current cross-sections, MDA determined that the minor footwall and hanging wall veins would not be included in the current resource due to the limited drill intercepts and uncertainty in structure/vein continuity. Accordingly, the current resource model is based on just the K5 and K13 veins.

Using the same assay coding procedures as used at Lincoln-Comet, the Keystone drill assays were color-coded based on the population breaks seen in the general assay population: low-grade 0.01oz Au/ton to 0.07oz Au/ton, mid-grade (0.07oz Au/ton to 0.25oz Au/ton), and high-grade (>0.25oz Au/ton). Using the labeled K5 and K13 vein intervals as a guide, low- and mid-grade gold mineral domain cross-sectional polygons were created based on the drill assay populations. Due to the limited number of high-grade samples, a unique high-grade domain was not created and those high-grade samples are included within the mid-grade domain. The gold polygons were limited in their elevation extent, both up-dip and down-dip, by either existing drill data or by geologic constraints (structural intersections, etc.) as modeled by Payne in 2008.

The low-grade assay population represents the weakly mineralized wallrock, or low-grade portions of the structure/veins, that are likely sub-economic with a limited chance of eventual economic extraction. These samples values are all well below the resource cut-off grade (using the same 0.12oz Au/ton cut-off gold grade as at Lincoln-Comet) and therefore are not included within the undiluted polygonal resource estimate.

The current K5 mineralized vein, as interpreted in the cross-sectional model, has an approximate 1,000ft strike length and an 800ft down-dip extent. The vein thickness that contributes to the current polygonal resource ranges from 3ft to 8ft.

The K13 mineralized vein, as interpreted in the cross-sectional model, has an approximate 800ft strike length and a 600ft down-dip extent. The vein thickness that contributes to the current polygonal resource ranges from 3ft to 6ft.

A tonnage factor of 12cuft/ton is used for the Keystone mineralization. This is the same tonnage factor used in the Lincoln-Comet resource.

The Keystone resource is based on an undiluted polygonal cross-sectional model using only the mid-grade gold polygons. In order to localize the estimate, the cross-sectional mid-grade polygons were broken into sub-polygons localized around each drill hole. These local polygons, which extended about 100ft from the drill intercept, were assigned the grade of the coded drill intercept. Away from the drill data, the mid-grade polygons which had no coded drill assays within them were assigned the average grade of all drill intercepts of that specific vein.

MDA reviewed the assigned grade for each polygon and those polygons with grades below the 0.12oz Au/ton cut-off were removed from the resource tabulation. Due to the wide drill spacing, and lack of modern underground sampling data, the Keystone resource is restricted to an Inferred classification. Table 1.5 shows the tons, gold grade, and gold ounces within each vein along with the total Keystone resource.

Table 1.5 Keystone Inferred Resources

vein	Tons	grade oz Au/ton	oz Au
K5	301,000	0.261	79,000
K13	98,000	0.189	18,000
total	399,000	0.243	97,000

Additional drilling within the currently defined deposit could materially change the existing resource with the discovery of either localized, high-grade mineralization within the known veins, or new veins branching off the main structures.

Tightly spaced drilling could result in an upgrade in classification. As in the Lincoln-Comet, some underground development is advised to better characterize the local grade variability along the veins. An Indicated classification would also warrant the construction of a three dimensional block model and grade estimate to better characterize the local grade variability and vein location.

There is potential for expanding the Inferred resource by drilling both down-dip and along strike to the northwest and southeast.

An issue that affects the Keystone resource model and estimate is that the current resource is spatially related to historical underground development. There is the possibility that some of the current resource in both veins has been mined out.

The Keystone resource is undiluted and some dilution is expected depending on proposed mining methods. The affect on the resource is unknown but it is possible that some portions of the resource would no longer be considered economic under certain circumstances.

1.8 Lincoln Mine Resource Summary and Conclusions

The Lincoln Mine property lies within the most productive portion of the historic Mother Lode in the western foothills of the Sierra Nevada. Eight major past-producing Mother Lode mines within the project area together accounted for 3.4 million ounces of gold production prior to 1953, or about 25% of the entire Mother Lode lode gold production.

The current combined Lincoln Mine gold resource is contained in over 30 distinct veins, many of which branch off the main through-going vein structures. The most important characteristics that impact the resource estimates are the strongly anastomosing nature of the narrow, gold-bearing veins and the significant mineral variability within the veins. The mineral variability is on a sub-sample scale to a mining scale and results in uncertainty in grade estimation both in the resource model and also in mine planning and reconciliation. Away from the underground development, there is spatial uncertainty in the geologic model due to the more widely spaced drill data and the highly variable, branching nature of the vein system.

Additional drilling within the currently defined deposits, especially away from the underground development, could materially change the existing resource with the discovery of either localized, high-grade mineralization within known veins or new veins branching off the main structures. There is potential for expanding the Inferred resources by drilling both down-dip and along strike to the northwest and southeast. Increasing the Indicated resource, though, would likely entail further underground development and/or tightly-spaced drilling.

An issue that affects the current Lincoln-Comet resource model and estimate is that the current property boundary impinges on the resource along the northeast boundary of the Comet zone. Any changes to this boundary would have a material effect on the resource model and estimate.

An issue that affects the Keystone resource model and estimate is that the current resource is spatially related to historical underground development. There is the possibility that some of the current resource in both veins has been mined out.

No mineral reserves have been identified on the project and mineral resources that are not mineral reserves do not have demonstrated economic viability.

1.9 Preliminary Economic Assessment

SGM constructed a 210 ton per day processing mill on the Lincoln Mine property and completed about 3,300ft of underground development during the period of late 2012 through early 2014. Approximately 3,100 tons of low grade material has been stockpiled near the mill. About 1,000 tons of low-grade material was processed in the mill during 2013-2014, but the rod mill produced excessive fines and the rate of processing material through the grinding circuit was much lower than expected. Gold recovery was very low due to the excessive fines. Tons Per Hour, Inc completed a detailed cost estimate for mill optimization modifications which are incorporated in this PEA study.

A preliminary economic assessment was completed for the project in 2011 based on mining the Lincoln-Comet deposit by underground methods. MDA has updated this study with work completed between 2012 and 2014. The scope of work includes the analysis and selection of an appropriate mining method and production rate, along with appropriate capital and operating cost estimates. Cash flow projections were completed based on assumed recoveries, both in situ and during processing, payment for products sold at the approximate current 3-year trailing average, plus two-year forward estimated gold price. The gold price used in this preliminary economic assessment is $1,200.00 per ounce of gold.

This preliminary economic assessment is preliminary in nature, includes 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, and there is no certainty that the preliminary assessment will be realized.

Various mining methods were evaluated and cut-and-fill stoping utilizing mill tailings for the fill is the selected method for this evaluation and for planning. Cut-and-fill stoping with an average minimum mining width of three feet is the preferred method as it allows for a high degree of flexibility, excellent recovery, and low dilution. If the veins display areas of greater width or regularity, breast stoping could be applied locally, which would provide an economic benefit. A 150 ton per day production rate was chosen based on the mining method and the desire to maintain a minimum of a five-year operation. This production rate will require about eight stopes to be mined per day with about 13 being active. Preliminary economic estimates indicated the cutoff grade would be around 0.22oz Au/ton.

Mine access using rubber-tire trucks will be provided by one primary decline, the existing Stringbean Alley decline, and two secondary declines. Horizontal levels are planned on regular 100-foot vertical spacing for access to the stope areas. Each level, consisting of one main drift (more if geometry warrants) driven from the Stringbean Alley decline and its existing cross-cuts, will allow for access to multiple veins and stope panels, while providing near-horizontal transport of “ore grade” material from stopes to main haulage ore-passes. Each vein and its stopes will be accessed via crosscuts from the main level drift.

Production mining utilizing the overhand back stoping, cut-and-fill mining method will take place in nominal 100-foot by 100-foot by 3-foot panels. Each panel will be developed by at least one raise and a slusher drift ("scram drift") in the vein on each level.

Based on the review of the available metallurgical reports and conclusions contained therein, a mill flowsheet has been developed for a 210 tons per day (150 tons per day equivalent at seven days per week) gravity and flotation mill. The mine trucks will deliver “ore grade” material to drive-over truck dump bins of approximately 400 tons live capacity. Sequential crushing and screening will result in a mill feed of 0.5 inch material to the rod mill. Minus 10 mesh ground material will flow by gravity to a centrifugal concentrator for the production of a rougher gold concentrate and rougher gravity tailing. The rougher gravity tailing will be pumped and sized by a hydrocyclone, with 100 mesh material flowing by gravity to the flotation cells. The hydrocyclone underflow will report to rod mill for regrinding. After flotation, the final tailings will be pumped to a 50 to 60-foot-diameter thickener for the reclamation of process water and for the production of mine hydraulic sand backfill. Tailings not used as backfill will be dewatered and transported by truck to the Surface Fill Unit (“SFU”).

The rougher gravity and flotation concentrates produced will contain in excess of 90 percent of the precious metal contained in the whole ore. For the purpose of economic analysis, 96% recovery from processing is assumed with 70% of gold reporting to the gravity circuit concentrate. It is expected that the mill will produce approximately 10 pounds of gravity final product per day. This high-grade concentrate, which assays up to 1,000oz Au/ton in bulk sample metallurgical tests, is amenable to direct smelting and will be processed onsite in the planned metallurgical laboratory into doré bars for sale to a refinery. The flotation circuit will consist of roughing and one-stage cleaner flotation. The circuit is designed to recover fine gold that the gravity circuit misses and to eliminate arsenic in the final tailings to a level below that consistent with the California Solid Waste Requirements for Class B solid wastes. The flotation concentrate will be dried and stored on-site in sack-type containers until at least one truckload has been accumulated. The approximately 22-ton shipment will be loaded and transported for further processing by a second party.

The SFU will impound undersize and whole mill tailings in a location to the east of the mine site (Swift Parcel). Dewatered tailings will be transported via 26-ton transfer dump trucks from the mill to the SFU, where they will be dumped/stacked and contoured. The Waste Rock Pile ("WRP") will be located very near the portal and will fill the small valley in front of the portal. The WRP will feature geosynthetic clay and a double membrane liner system, with associated drainage controls and water diversions.

Staff scheduling for mining operations of this nature would consist of two shifts of 10 hours each for the mine and three shifts of eight hours each for the mill. The mine will operate seven days per week, 350 days per year. Four shift crews will be necessary to fill the rotation of seven days on, seven days off for the mine. Milling will require only three shift crews, working five days per week, with weekends off. Total manpower needs, including administrative personnel, are 108 employees.

Estimated capital costs, including nine months of pre-production development, totals $11,292,100, while estimated operating costs average approximately $15,478,900 per year over the five year mine life. Mining labor costs are estimated at $6,554,100 per year, about 43%, of the total operating costs. The unique geometry, grade, and narrow vein widths of the Lincoln-Comet deposit leave little alternative to a high-selectivity method and consequently higher mining cost per mined unit.

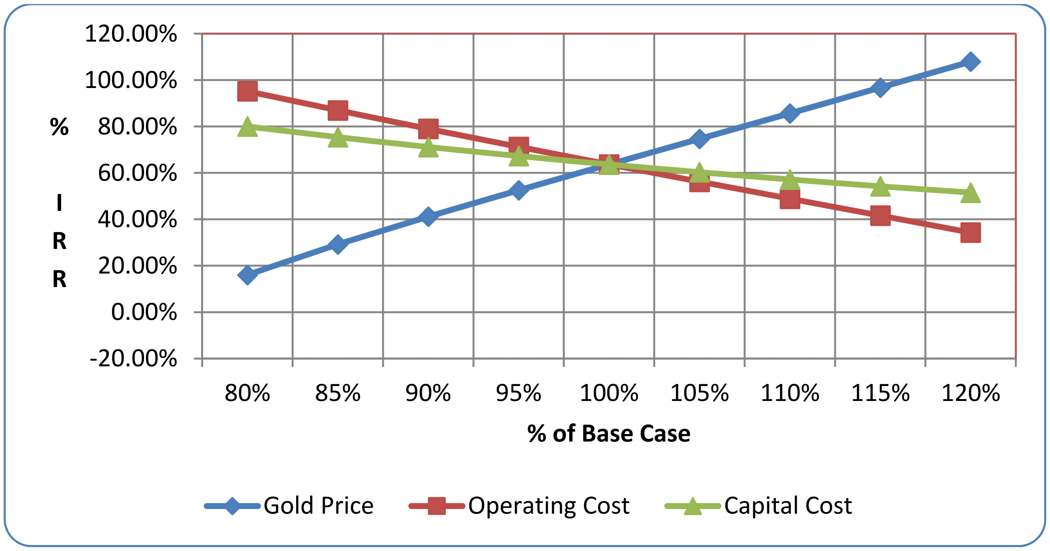
The company has in excess of $30 million in tax write-offs to offset any taxes so an after-tax cash-flow evaluation is not applicable. The pre-tax cash-flow evaluation indicates an internal rate of return ("IRR") of 63.7% while the net present value ("NPV") at 5% is $23,411,300. A sensitivity analysis completed for the cash-flow indicates the project is most sensitive to the price of gold and then operating costs. Table 1.6 summarizes the pre-tax cashflow from the project. The operating cost to produce an ounce of gold is $703.95.

Table 1.6 Pre-tax Project Cashflow

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	TOTAL
PRODUCTION								
Waste Tons (000'S)	39.6	13.7	12.2	12.2	12.2			89.8
Mine Production - Ore Tons (000'S)	8.4	52.5	52.5	52.5	52.5	26.4		244.8
Grade (ounce per ton)	0.46	0.46	0.46	0.46	0.46	0.46		0.46
Contained Oz Au (000'S)	3.8	24.0	24.0	24.0	24.0	12.1		111.9
Production - Gravity (000's oz Au)	2.7	16.8	16.8	16.8	16.8	8.4		78.3
Production - Flotation (000's oz Au)	1.0	6.2	6.2	6.2	6.2	3.1		29.1
Production - Flotation (000's oz Au Paid- 85%)	0.8	5.3	5.3	5.3	5.3	2.7		24.7
Total Sales (000's oz Au)	3.5	22.1	22.1	22.1	22.1	11.1		103.0
Gold Price - $/oz Au	1200	1200	1200	1200	1200	1200		
Gross Revenue ($000's)	$4,244.2	$26,491.7	$26,491.7	$26,491.7	$26,491.7	$13,260.3		$123,471.4
Royalty								
4.0% Gross Royalty (~Net of all royalties)	$169.6	$1,058.7	$1,058.7	$1,058.7	$1,058.7	$529.9		$4,934.2
Net Revenue	$4,074.6	$25,433.1	$25,433.1	$25,433.1	$25,433.1	$12,730.4		$118,537.2
OPERATING COSTS								
Mining ($000's)								
Development - Waste (000's)	$0.0	$902.7	$686.2	$686.2	$686.2	$0.0		$2,961.4
Development - Ore (000's)	$0.0	$255.4	$255.4	$255.4	$255.4	$127.7		$1,149.4
Ore Mining	$565.3	$3,304.3	$3,304.3	$3,304.3	$3,304.3	$1,643.8		$15,426.3
Mine Labor	$1,638.5	$6,554.2	$6,554.2	$6,554.2	$6,554.2	$3,280.7		$31,136.0
Total Mining ($000)	$2,203.8	$11,016.6	$10,800.2	$10,800.2	$10,800.2	$5,052.1		$50,673.2
Processing ($000's)	$481.9	$2,210.3	$2,210.3	$2,210.3	$2,210.3	$1,099.9		$10,423.1
General and Administrative ($000's)	$538.1	$2,415.9	$2,415.9	$2,415.9	$2,415.9	$1,202.3		$11,404.0
Totals ($000's)	$3,223.9	$15,642.8	$15,426.4	$15,426.4	$15,426.4	$7,354.3		$72,500.2
Net Profit ($000's)	$850.7	$9,790.2	$10,006.7	$10,006.7	$10,006.7	$5,376.1		$46,037.0
Capital Investment	$12,474.7							$12,474.7
Working Capital	$3,223.9		($3,223.9)					$0.0
Closure Costs							$3,998.4	$3,998.4
Cash Flow	($14,848.0)	$9,790.2	$13,230.6	$10,006.7	$10,006.7	$5,376.1	($3,998.4)	$29,563.8
Cumulative Cashflow	($14,848.0)	($5,057.7)	$8,172.8	$18,179.5	$28,186.2	$33,562.3	$29,563.8	
NPV (5%)							$23,411.3	
NPV (8%)							$20,368.7	
IRR							63.7%	

Figure 1.1 indicates that the internal rate of return is most sensitive to changes in the gold price, followed by operating cost and capital cost.

Figure 1.1 Internal Rate of Return Sensitivity



1.10 Recommendations and Conclusions

MDA believes that the biggest risk with the Lincoln Mine project is the ability to identify "ore grade" material within the quartz veins. Additionally, the normal risks associated with underground mining are present, such as dilution and mining cost. The main opportunity at the property is believed to be the potential to increase resources down-dip of the current mineralization. While the drilling has indicated a drop off in grade, there are a number of veins that have not been adequately drilled down-dip of the current resources. In addition, consideration should be given to deep exploration by gathering as much data as possible on historic production on the property and from neighboring deposits. Adding to risk is that about 80% of the current resources are in the inferred category.

The following work is recommended to improve the knowledge of the deposit:

- During the pre-production development period as planning for the project proceeds, a test mining period should be included for the purpose of completing a detailed evaluation of the stope panels required to achieve production. Test mining should be completed to develop detail design parameters for productivity and to develop a sampling program for mining. Test mining will also help assess the costs and dilution estimated to occur during mining. An extensive sampling program coinciding with further development work to determine what level of sampling is appropriate to identify the ore-grade mineralization. Cost has been accounted for in development costs.

- Further testing of tailings and concentrate material characteristics should be completed prior to final backfill engineering. The estimated cost for this testing is $40,000.

- Although the need for ground support during development and production mining phases is expected to be light, with increasing mining depth a future review of the ground support program will be needed. MDA recommends that a ground support plan be designed by a geotechnical engineer. The estimated cost of a ground support plan is $20,000.

2.0 INTRODUCTION AND TERMS OF REFERENCE

Mine Development Associates ("MDA") has prepared this technical report on the Lincoln Mine project, located in Amador County, California, at the request of Sutter Gold Mining Inc. ("SGM"), which is incorporated in the province of British Columbia and is listed on the TSX Venture Exchange. Sutter Gold Mining Inc. controls the Lincoln Mine project through its wholly owned subsidiary, Sutter Gold Mining Company. In this report, "SGM" will be used interchangeably for both Sutter Gold Mining Company and Sutter Gold Mining Inc., unless a distinction is necessary. This report was written in accordance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014.

2.1 Project Scope and Terms of Reference

The purpose of this report is to provide an updated technical summary of the Lincoln Mine project that includes mineral resource estimates of the Lincoln-Comet and Keystone portions of the property, in accordance with NI 43-101, as well as a Preliminary Economic Assessment ("PEA") of the project.

The Lincoln-Comet and Keystone resources are Mother Lode-style gold deposit hosted within near-vertical, 1 to 4ft-wide mesothermal quartz veins. The Lincoln-Comet resource occurs over a 3,000ft strike length and has an 800ft downdip extent. SGM proposes to exploit the resource using underground mining methods that include cut-and-fill stoping and access through multiple declines. Gold recovery is primarily through gravity separation, though a flotation circuit will recover the remaining fine gold and also remove the arsenic from the mine tailings.

The Keystone resource area lies 2,000ft north of the Lincoln-Comet resource. The current resource occurs within two distinct veins that have a maximum 1,200ft strike length and an 800ft down-dip extent.

The mineral resources were estimated and classified under the supervision of Paul Tietz, Senior Geologist for MDA, and Steven Ristorcelli, Principal Geologist for MDA, who are qualified persons under NI 43-101. The PEA was compiled under the supervision of Neil Prenn, Principal Engineer for MDA and a qualified person under NI 43-101. No Mineral Reserves are estimated for the project. There is no affiliation between Mr. Tietz, Mr. Ristorcelli, or Mr. Prenn and SGM except that of an independent consultant/client relationship. The mineral resources reported herein for the Lincoln Mine project were estimated to the standards and requirements stipulated in NI 43-101. Other resource estimates presented in Section 6.3 are reported for purposes of completeness only and do not necessarily meet the reporting requirements of NI 43-101.

The scope of this study included a review of pertinent technical reports and data provided to MDA by SGM relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, drilling programs, and metallurgy. MDA has relied on the data and information provided by SGM for the completion of this report, including the supporting data for the estimation of the mineral resources. In compiling the background information for this report, MDA relied on the 2011 technical report and PEA prepared by MDA (Tietz et al., 2011), a 2008 report prepared by Mark Payne, the 2004 technical report prepared by MDA (Ronning and Prenn, 2004), the 2007 pre-feasibility study prepared by Behre Dolbear & Company (USA), Inc. ("Behre Dolbear"), and on other references as cited in Section 20.0. Behre Dolbear (2007) stated that their report should be considered not in accordance with NI 43-101 reporting requirements "*since it makes grade and tonnage projections in future years of the project's 10-year plan that requires verification by further exploration of the ore deposit*."

The authors' mandate was to evaluate the effects of development work done by SGM since 2011 on the 2011 Lincoln-Comet PEA and resource estimate and bring the Keystone mineralized zones into current status. The mandate also required on-site inspections and the preparation of this independent technical report containing the authors' observations, conclusions, and recommendations. Mr. Prenn visited the site November 10, 2003. Mr. Tietz conducted a site visit on April 19, 2009, and a second site visit was conducted by Mr. Tietz and Mr. Ristorcelli on June 11, 2009. Mr. Ristorcelli made a second site visit on June 18, 2009. Subsequently, Mr. Tietz and Mr. Prenn visited the site on May 12 and 13, 2015 and June 20, 2015, respectively. The site visits included tours of the underground workings, mine infrastructure, and reviews of the project geology and database to be used in the resource estimate. MDA has made such independent investigations as deemed necessary in the professional judgment of the authors to be able to reasonably present the conclusions discussed herein.

The drill and underground assay data used in the Lincoln-Comet and Keystone resource estimate has an effective date of September 2, 2009. The collar location database has an effective date of February 15, 2010. The initial Lincoln-Comet resource model and estimate, based on the February 15, 2010 data, were completed in May 2010, with a revised model and estimate completed December 14, 2010. The December model revision was required due to MDA's receipt of a revised land map indicating a minor change in the SGM-controlled property position. The Lincoln-Comet resource reported in 2011 reflected the revised model and estimate.

SGM completed additional underground mapping and sampling in 2009 and 2010 and a surface and underground drill program associated with the renewed Lincoln-Comet underground development in 2012 and 2013. MDA has reviewed these data and determined that inclusion of these data will not materially change the Lincoln-Comet resource estimate. Therefore the Lincoln-Comet resource estimate reported in 2011 is considered current.

The effective date of this technical report, which includes the updated Lincoln-Comet PEA and the Keystone resource estimate, is July 2, 2015.

This preliminary economic assessment is preliminary in nature, includes 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, and there is no certainty that the preliminary assessment will be realized.

No mineral reserves have been identified on the project and mineral resources that are not mineral reserves do not have demonstrated economic viability.

2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

In this report, measurements are generally reported in Imperial units. Where information was originally reported in metric units, MDA has made the conversions as shown below.

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

Linear Measure

1 centimeter	= 0.3937 inch	
1 meter	= 3.2808 feet	= 1.0936 yard
1 kilometer	= 0.6214 mile	

Area Measure

1 hectare	= 2.471 acres	= 0.0039 square mile

Capacity Measure (liquid)

1 liter	= 0.2642 US gallons

Weight

1 tonne	= 1.1023 short tons	= 2,205 pounds
1 kilogram	= 2.205 pounds	

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

Frequently used acronyms and abbreviations

AA	atomic absorption spectrometry
Ag	silver
asl	above sea level
Au	gold
BLM	United States Department of the Interior, Bureau of Land Management
CIM	Canadian Institute of Mining, Metallurgical, and Petroleum
core	diamond core-drilling method
°C	degrees centigrade
°F	degrees Fahrenheit
FA-AA	fire assay with an atomic absorption finish

ft	foot or feet
in	inch or inches
kg	kilograms
lb(s)	pound/pounds
m	meter or meters
M	mesh
MDA	Mine Development Associates, the authors of this technical report
mi	mile or miles
mm	millimeter or millimeters
oz	troy ounce (12oz to 1 pound)
oz Au/ton	ounces of gold per short ton
QA/QC	quality assurance and quality control
RC	reverse-circulation drilling method
ROM	Run of Mine, referring to rock that has been blasted but not crushed or otherwise beneficiated
t	short ton
tpd	tons per day

2.3 Definitions of Terms

The following information on mine levels and spatial reference systems in subsections 2.3.1 and 2.3.2 is taken from Ronning and Prenn (2004).

2.3.1 Mine Levels

In those parts of this report that discuss historical mining operations and past production, frequent reference is made to mine levels. Unless otherwise stated, the designation of mine levels is local to each specific mine. Levels are usually measured starting at the surface collar of each mine's main shaft and counting downwards, so that, for example, the 700ft level of the South Spring Hill mine would be 700ft below the collar of the South Spring Hill shaft. The 800ft level would be 100ft deeper than the 700ft level.

It is important not to confuse historical mine levels with elevation above sea level. For example, because the collar of the Talisman shaft is at a higher elevation above sea level than the collar of the South Spring Hill shaft, the Talisman 900ft level is closer in elevation to the South Spring Hill 700ft level than to the South Spring Hill 900ft level.

With the several separately owned mines that once occupied the SGM land holdings, there is a multiplicity of mine-level systems in the existing records.

2.3.2 Spatial Reference Systems

Depending on their source and purpose, geographic or spatial references in data relating to the Lincoln Mine project may refer to one of three systems.

- **Truncated State Plane** - Most surface and many underground maps show a California State Plane ("CSP") grid in feet. The first two digits of the Easting and the first digit of the Northing have been truncated. In other words, only the last five digits of the Northing and Easting are shown. For example, real coordinates of 2,340,000 East, 271,000 North become 40,000 East, 71,000 North when truncated.

- **Lincoln Shaft -** Drill-hole cross sections showing the holes drilled from the surface are spaced along a longitudinal line (the "reference line") that trends 330 degrees from a State Plane origin at 42,520E, 69397N, as measured from a 1in = 100ft mylar map. This origin is at or near the original site of the Lincoln shaft. Cross sections are perpendicular to the longitudinal line; oriented at 240° - 060°. Cross sections are numbered from the origin, so for example a 1000 N cross section would be 1,000ft northwest, at 330 degrees from the origin.

- **Decline** - The Stringbean Alley decline portal, as surveyed by SGM, is at State Plane 40,367.58E, 72,890.88N, elevation 1,182.84ft asl. Decline crosscuts are sequentially numbered by counting from the first crosscut from the portal and labeled as to which side of the decline the crosscut extends (east or west), e.g. SBA4W or simply 4W.

In this report, where possible, all spatial references are given using the first system, the truncated California State Plane grid references. Where it is necessary to use the other systems, their use is noted.

3.0 RELIANCE ON OTHER EXPERTS

MDA is not an expert in legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, and property agreements in the United States. MDA did not conduct any investigations of the environmental or social-economic issues associated with the Lincoln Mine project and is not an expert with respect to these issues.

The authors have relied on SGM to provide full information concerning the land area, current legal title, material terms of all agreements, and existence of applicable royalty obligations that pertain to the Lincoln Mine property. Sections 4.1, 4.2, and 4.3 are based on information provided by SGM, and the authors offer no professional opinions regarding the provided information.

MDA has relied upon Stephen T. Lofholm, Senior Consultant and Associate with Golder Associates Inc., who is an expert on environmental issues and who has provided technical support for various environmental studies to Sutter Gold Mining Company periodically since 2005, for Section 4.4 Environmental Permits and Potential Liabilities, and 20.0 Environmental Studies, Permitting, and Social or Community Impact.

MDA has relied upon Dr. Corby G. Anderson, CEng FIChemE, with Allihies Engineering Incorporated of Butte, Montana, for Section 1.6 Metallurgical Testing and Section 13 Mineral Processing and Metallurgical Testing. Mr. Anderson is a qualified person under NI 43-101.

4.0 PROPERTY DESCRIPTION AND LOCATION

The authors are not experts in land, legal, environmental, and permitting matters. Sections 4.1, 4.2, and 4.3 are based entirely on information provided to MDA by SGM and on other references as cited. Section 4.4 is based entirely on information provided by David Cochrane of SGM and approved by Stephen Lofholm of Golder Associates Inc. MDA presents this information to fulfill reporting requirements of NI 43-101 and expresses no opinion regarding the legal or environmental status of SGM's Lincoln Mine Project property and mineral holdings, or any of the agreements and encumbrances related to the property.

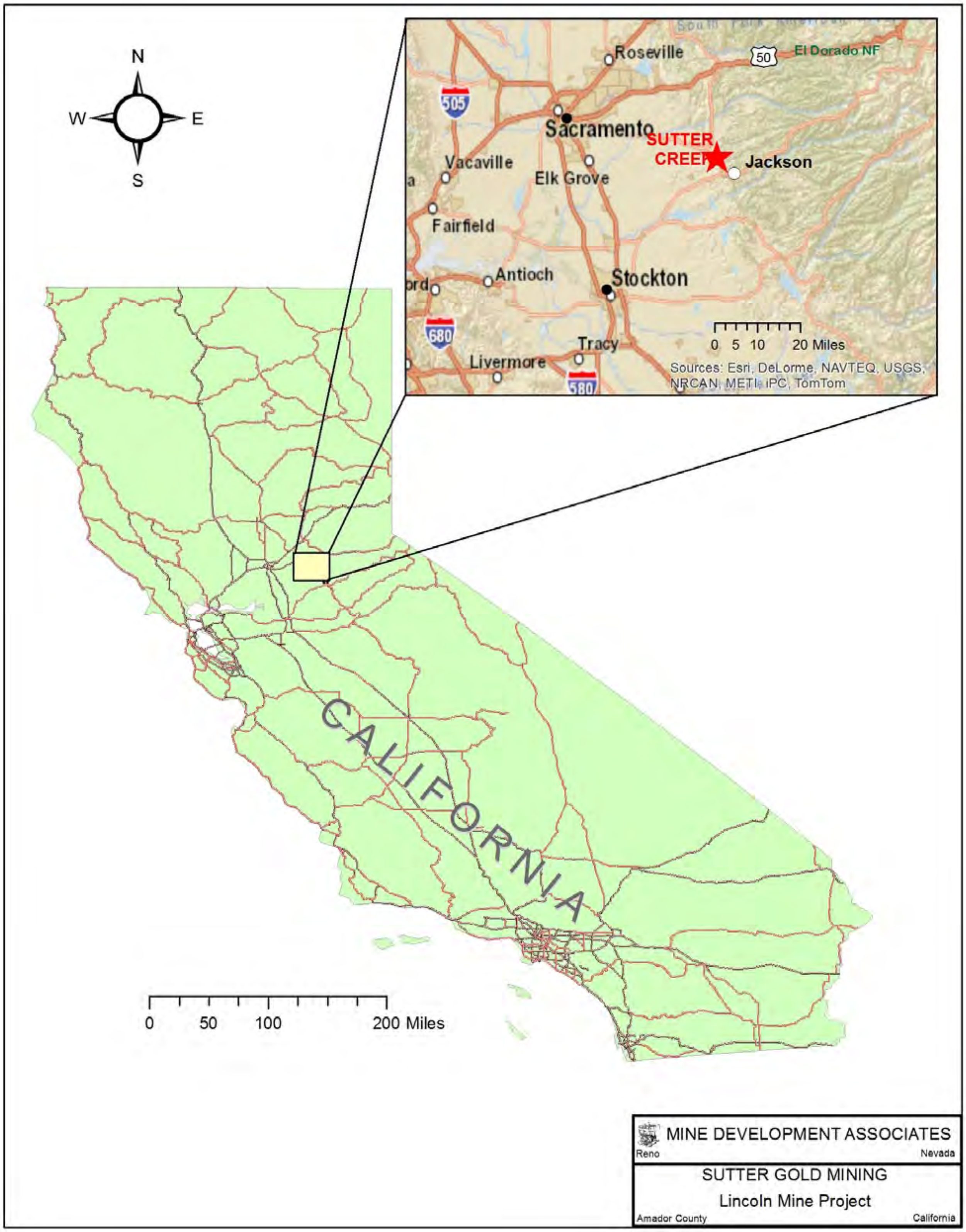
4.1 Location

The Lincoln Mine Project is located in central California, about 45mi east-southeast of Sacramento in western Amador County (Figure 4.1). The project lies in the foothills of the western slope of the Sierra Nevada mountain range in the central part of the historic Mother Lode gold belt. The SGM's Lincoln Mine Project property and mineral holdings trend about 3.6 mi northwest from the southern edge of the town of Sutter Creek, on the southeast, to beyond the northern edge of the town of Amador City on the northwest (Figure 4.2). The project's mine site is about 0.5mi east of State Highway 49 near the north end of the town of Sutter Creek.

The property is situated within the Amador City 7 ½ minute quadrangle map. The central part of the property is located at about 38° 25'N latitude and 120° 49' W longitude. Using NAD 27 California State Plane 27, Zone 2, coordinates in feet, the northwest end of the property is at about 2,336,600E, 276,800N, and the southeast end at about 2,345,660E, 267,100N (Ronning and Prenn, 2004).

The focus of this report and its resource estimate is the Lincoln-Comet zone, which is a 1 mi long area located within the larger Lincoln Mine Project, and located a little north of the central part of the property, north of the Lincoln and south of the Spring Hill mines.

Figure 4.1 Location of the Lincoln Mine Project



4.2 Land Area

The following information on the land area of the Lincoln Mine Project has been provided by SGM, with additional information from other references as cited. A copy of the 2007 title update on 9 of the mining claims held by Sutter Gold, prepared by attorneys Harris & Thompson (Thompson, 2007), appears as Appendix 2.0 in the Behre Dolbear (2007) report. The 9 claims reviewed by Thompson (2007) were: Golden Eagle claim, Comet quartz mine, North Star quartz mine, Wabash quartz mine, Ronald Littlefield fee parcel, East Keystone lode mining claim, South Spring Hill quartz mine, Medean mine, Herbertville quartz mine, and the mineral lease with Keystone Mines, Inc., affecting only the minerals to East Keystone, South Spring Hill, Medean, and Herbertville. It should be noted that these 9 claims do not include all of those that cover the Lincoln-Comet mineralized zone as listed below. Figure 4.2 shows the Lincoln Mine Project property including surface and mineral rights; Figure 4.3 is an enlargement showing the Lincoln-Comet resource outline and the claims that overlie it.

In total, SGM owns or leases 711.08 acres of mineral rights and 173.21 acres of surface rights. The property currently consists of 47 individual parcels, including the original Lincoln Mine Project lands and mineral holdings (29 parcels) and 18 additional parcels added in 2004 and 2009. In addition to these parcels, which SGM either owns or leases, SGM has easement agreements that cover their required private access road for the mine and mill, and their access road for the surface fill unit, shown in blue on Figure 4.2 and Figure 4.3. SGM also has easements for construction and operation of a slurry pipeline to the surface fill unit, although they do not plan to construct the pipeline at this time.

SGM's original Lincoln Mine Project property consists of two non-contiguous portions. The larger, which contains the portion relevant in accessing the mineralized zones, including the Lincoln-Comet and Keystone, is potentially available for mining and is a northwest-trending belt made up of original patented mining claims of the district, residential lots, and agricultural holdings. It covers portions of Section 36, T7N, R10E; Section 31, T7N, R11E; and Sections 5, 6, 7, and 8, T6N, R11E. In these two portions of the property, SGM owns or leases 417.09 acres of mineral rights and 173.21 acres of surface rights in 29 parcels that are shown in red on Figure 4.2 and Figure 4.3. These 29 parcels of the original Lincoln-Comet property are listed on Table 4.1. According to SGM, all property boundaries are accepted as defined by the property deeds and the Amador County Assessor. In addition, SGM leases a parcel of about 35 acres for tailings disposal in Section 32, T7N, R11E and Section 5, T6N, R11E. This is not contiguous with the larger portion of the property; SGM controls only surface rights in this area (shown as "surface fill unit" on Figure 4.2).

SGM also leases an additional 293.26 acres of mineral rights in 18 parcels (The Bunker Hill -Original Amador property and the Central Eureka property) that are contiguous to the original Lincoln Mine Project mineral and land holdings, and extend SGM's holdings both northwest and southeast of the original holdings. SGM obtained these additional mineral rights in 2009 and 2004, respectively, to further consolidate the district and for future exploration and development. These are newer acquisitions, which have not been included in current permits, studies, resource estimates, or pre-feasibility studies for the Lincoln Mine project. They are not covered under the existing Conditional Use Permit and have not been subject to environmental review. These properties are listed in Table 4.2 and are shown with blue outlines on Figure 4.2. All property boundaries are accepted as defined by the property deeds and the Amador County Assessor.

Table 4.1 SGM Original Properties Owned and Leased for the Lincoln Mine Project
(Updated by SGM, 2015)

Claim/Description	Parcel #	Acres	Surface	Mineral
Comet	40-010-012-000	6.62	Lease	Lease
Emerson	18-010-007-502	1.80		Own
Emerson	18-010-008-502	46.68		Own
Golden Eagle/Triumph	40-010-013-000	12.26	Lease	Lease
Herbertville	15-210-043-501	16.11	Own	Lease*
Keystone, S. Spring Hill, Medean and Herbertville	15-210-017-000	148.39		Lease
Keystone Gold, Spring Hill, Geneva & East Keystone	08-260-024-502	50.18		Lease
Spring Hill, Geneva, East Keystone & South Spring Hill	08-260-030-502	25.38		Lease
Keystone Gold	08-310-017-502	0.20		Lease
Spring Hill, Geneva, East Keystone & S. Spring Hill	08-260-027-501/502	20.11	Own	Lease**
South Keystone	15-210-010-000	1.63	Lease	
Lincoln	18-010-001-502	8.13		Own
Lincoln	18-010-002-502	3.61		Own
Lincoln	18-010-003-502	1.78		Own
Lincoln	40-010-018-502	17.50	Lease***	Own
Wildman/Mahoney	18-010-006-502	20.43		Own
Medean	15-210-042-501	9.09	Own	Lease*
Mill Road	18-010-004-502	4.14		Own
Mine House	40-020-007-501	8.00	Own	Lease*
Niagara	08-260-038-502	17.90		Lease
North Star	40-010-008-000	8.35	Own	Own
Old Office Location	40-020-002-000	0.86		Own
Ron Littlefield Parcel	15-210-023-000	16.99	Own	Own
South Spring Hill	15-210-044-501	9.08	Own	Lease*
Stewart	40-010-019-502	1.82		Own
Sutter Creek Grammar School	18-133-009-000	3.02		Own
South Herbertville (Shop/Staff Services Building)	40-010-003-000	5.05	Own	Own ****
Surface Fill Unit (Tailings Disposal)	40-030-087-501	34.42	Lease	
Wabash	40-010-007-000	8.00	Own	Own

* Mineral rights included in parcel # 15-210-017-000
** Mineral rights included in parcel 08-260-024-502
*** Surface rights included in lease for parcel 40-210-018-501
**** 1.65 acres of minerals are owned; 3.4 acres leased as parcel 15-210-017-000

Table 4.2 SGM Properties Adjacent to Lincoln Mine Project
(Updated by SGM, 2015)

Claim/Description	Parcel #	Acres	Mineral	Royalty[1]	Lease Payment
Bunker Hill-Original Amador Property					
Bunker Hill	08-230-018-522	27.62	Lease (half)	4.0%	$5,000.00*
Bunker Hill Mill	08-230-020-502	20.96	Lease	4.0%	↓
Bunker Hill, Mayflower	08-230-023-502	58.62	Lease	4.0%	↓
East Amador, Great Eastern	08-250-020-502	13.92	Lease	4.0%	↓
East Amador, Great Eastern	08-250-021-502	16.71	Lease	4.0%	↓
Great Eastern	08-322-013-502	4.50	Lease	4.0%	↓
Hotel Alley	08-310-022-502	2.75	Lease	4.0%	↓
Niagra	08-260-048-502	4.04	Lease	4.0%	↓
Original Amador	08-250-047-502	7.42	Lease	4.0%	↓
School Street	08-287-008-502	5.70	Lease	4.0%	↓
Central Eureka Property					
Alpha	18-010-014-502	30.74	Lease	4.0%	$4,800.00*
Alpha	18-270-011-502	3.38	Lease	4.0%	↓
Amador	18-270-010-502	20.86	Lease	4.0%	↓
Amador	18-270-012-000	2.35	Lease	4.0%	↓
Amador	18-270-013-000	1.65	Lease	4.0%	↓
Maxwell	18-010-005-502	1.00	Lease	4.0%	↓
Railroad	18-010-011-502	56.52	Lease	4.0%	↓
Summit	40-030-048-502	15.25	Lease	4.0%	↓

1. In addition to the royalties shown on this table, there is a 5% Net Profits Royalty on all SGM properties that is payable to U.S. Energy Corp. See text for description.
*Lease for properties is in one combined payment.

Figure 4.2 Property Map of the Lincoln Mine Project

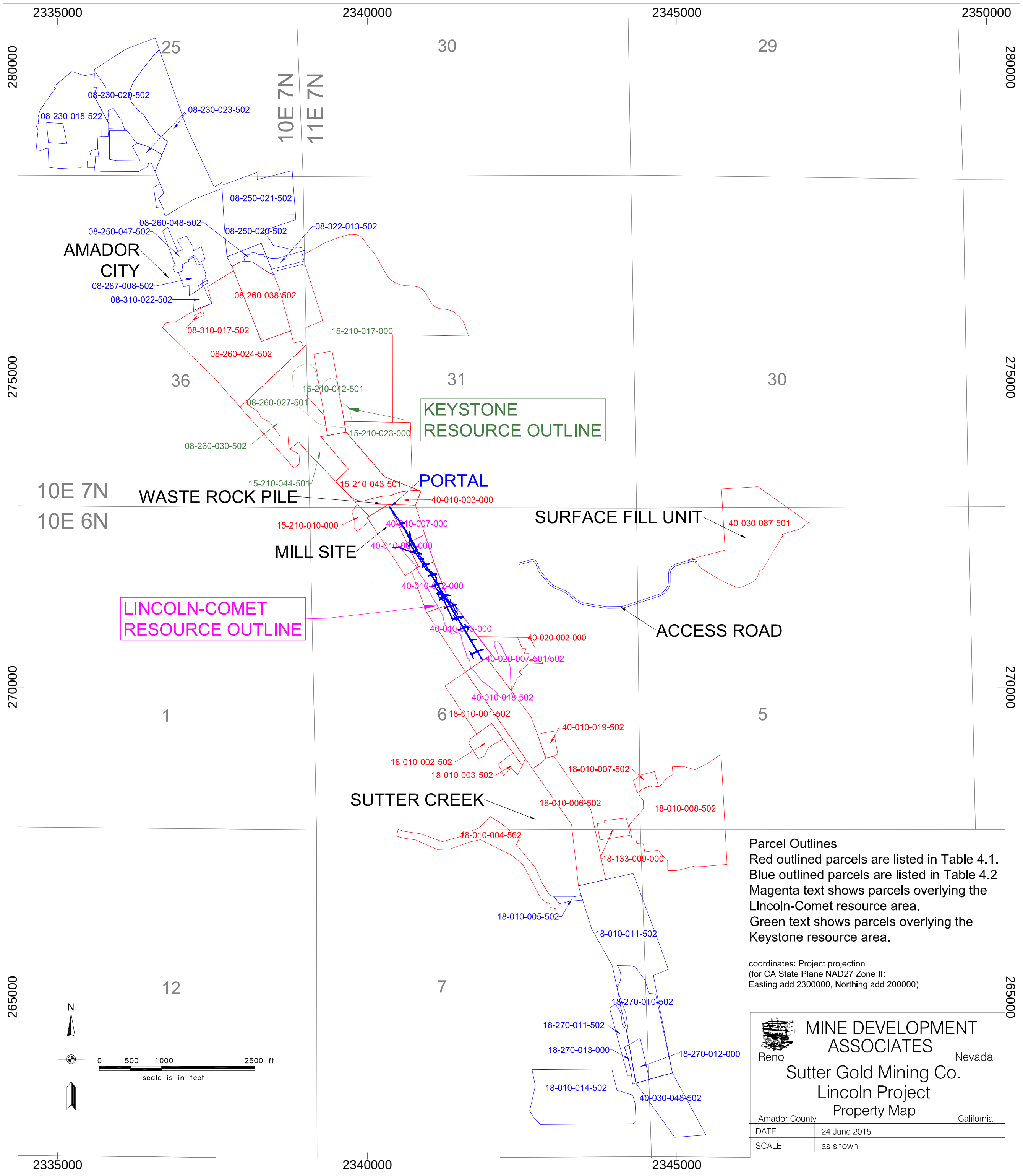
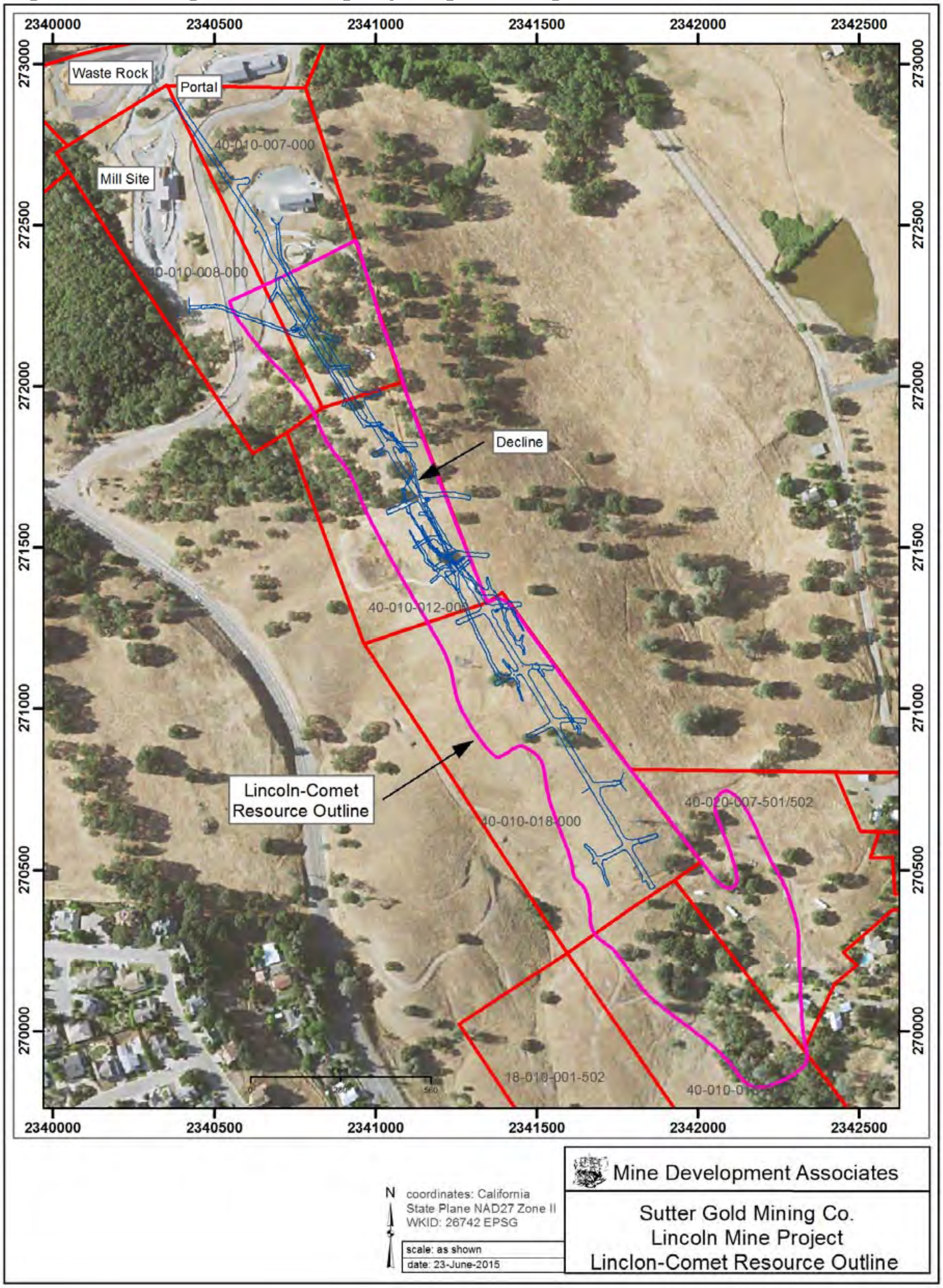


Figure 4.3 Enlargement of Property Map Showing Lincoln-Comet Resource Outline



Of those parcels listed in Table 4.1 and shown on Figure 4.2 and Figure 4.3, the following overlie the Lincoln-Comet and Keystone resources:

Comet (parcel 40-010-12-000)
Golden Eagle/Triumph (parcel 40-010-013-000)
Lincoln (parcel 40-010-018-501 and 502)
Mine House (parcel 040-020-007-502)
North Star (parcel 40-010-008-000)
Wabash (parcel 40-010-007-000)
Spring Hill, Geneva, East Keystone, and S. Spring Hill (parcel 08-260-027-501/502)
Spring Hill, Geneva, East Keystone, and S. Spring Hill (parcel 08-260-030-502)
Keystone, S. Spring Hill, Medean and Herbertville (parcel 15-210-017-000)
Ron Littlefield Parcel (parcel 15-210-023-000)
Medean (parcel 15-210-042-501)
South Spring Hill (parcel 15-210-044-501)

4.3 Agreements and Encumbrances

SGM controls the properties that make up the Lincoln Mine Project through outright ownership as a result of purchase and through lease agreements summarized in Table 4.1 and Table 4.2. Lease obligations and royalties for the properties in the original Lincoln Mine Project property (Table 4.1) that contains the portion relevant in accessing the Lincoln-Comet and Keystone mineralized zones are listed in Table 4.3. The twelve parcels overlying the Lincoln-Comet and Keystone resource area are shown with an asterisk on Table 4.3. Table 4.4 and Table 4.5 list details of the royalty obligations for the parcels that overlie the Lincoln-Comet and Keystone resource areas, respectively.

Table 4.3 shows the royalties and lease payments for those properties which SGM also leases adjacent to the original Lincoln Mine Project land and mineral holdings for future exploration and development. SGM acquired mineral rights to the Bunker Hill-Amador (Cecchettini) parcels by lease in 2009. SGM acquired mineral rights to the Central Eureka (Garibaldi) parcels by lease on December 23, 2004.

In addition to the royalties shown in Table 4.2, Table 4.3, Table 4.4, and Table 4.5, and as indicated above or below each table, there is a 5% Net Profits Royalty on all SGM properties that is payable to U.S. Energy Corp. This royalty is calculated on gross proceeds less all expenses, other royalties, depreciation, and taxes and is capped at a total of $4.6 million, with a 1% royalty thereafter.

For the Comet, Golden Eagle, Lincoln (parcel 40-010-018-502), and Mine House parcels listed on Table 4.4, there is an additional 0.5% royalty capped at $1 million that is payable to a consultant.

SGM has indicated to MDA that it believes that the total royalty on the material projected to be mined by this study would be about equivalent to a 4% net royalty.

Table 4.3 SGM Royalties and Lease Obligations for Properties Comprising the Original Lincoln-Comet Project
(Updated by SGM, 2015)

Claim/Description	Parcel #	Owner	Royalty[1]	Taxes Paid[2]	Lease Payment[3]
*Comet	40-010-012-000	TLC/MHC	4.0%	$398.78	$3,600.00
Emerson	18-010-007-502	SGMC	2.5%	$70.10	$0.00
Emerson	18-010-008-502	SGMC	2.5%	$100.56	$0.00
*Golden Eagle/Triumph	40-010-013-000	Crotty	4.0%	$574.98	$4,800.00
Herbertville	15-210-043-501	SGMC	5.0%	$235.12	$0.00
Keystone Gold, Spring Hill, Geneva & East Keystone	08-260-024-502	Keystone	5.0%	$106.86	$7,000.00
*Spring Hill, Geneva, East Keystone & S. Spring Hill	08-260-030-502	Keystone	5.0%	$18.98	? [4]
*Keystone, S. Spring Hill, Medean & Herbertville	15-210-017-000	Keystone	5.0%	$133.46	? [4]
Keystone Gold	08-310-017-502	Keystone	5.0%	$7.98	? [4]
*Spring Hill, Geneva, East Keystone & S. Spring Hill	08-260-027-502	SGMC	5.0%	$275.56	$0.00
South Keystone	15-210-010-000	KoldJeski	0.0%	$0.00	$10.00
Lincoln	18-010-001-502	SGMC	2.5%	$17.56	$0.00
Lincoln	18-010-002-502	SGMC	2.5%	$7.86	$0.00
Lincoln	18-010-003-502	SGMC	2.5%	$4.02	$0.00
*Lincoln[5]	40-010-018-502	SGMC	2.5%	$38.60	$0.00
Wildman/Mahoney	18-010-006-502	SGMC	2.5%	$44.16	$0.00
*Medean	15-210-042-501	SGMC	5.0%	$119.58	$0.00
Mill Road	18-010-004-502	SGMC	2.5%	$8.94	$0.00
*Mine House	40-020-007-502	SGMC	4.0%	$3,277.22	$0.00
Niagara	08-260-038-502	Keystone	5.0%	$13.50	? [4]
*North Star	40-010-008-000	SGMC	4.0%	$10,728.12	$0.00
Old Office Location	40-020-002-000	SGMC			$0.00
*Ron Littlefield Parcel	15-210-023-000	SGMC	4.0%	$2,974.48	$0.00
* South Spring Hill	15-210-044-501	SGMC	5.0%	$119.42	$0.00
Stewart	40-010-019-502	SGMC	2.5%	$4.05	$0.00
Sutter Cr.Grammar School	18-133-009-000	SGMC	2.5%		$0.00
South Herbertville	40-010-003-000	SGMC	0.0%	$60,165.78	$0.00
Surface Fill Unit	40-030-087-501	Swift	0.0%	$194.38	$2,399.38
*Wabash	40-010-007-000	SGMC	4.0%	$193.38	$0.00

* properties overlie the Lincoln-Comet and Keystone resource areas
[1]Royalty details vary by agreement.
[2]Paid annually, subject to change
[3]Paid annually
[4]Keystone Gold properties have one combined payment
[5] 4% royalty from surface to 100 ft only with annual lease ($2,400) and taxes ($760.56)

In addition to the royalties shown on this table, there is a 5% Net Profits Royalty on all SGM properties that is payable to U.S. Energy Corp. as described in text above. Also, in addition for the Comet, Golden Eagle, Lincoln (40-010-018-502) and Mine House parcels, there is a 0.5% NSR royalty payable to a consultant as described in text above.

Table 4.4 Royalty Obligations for the Parcels Containing the Lincoln-Comet Resource

APN#	Name	Holder	Royalty
40-010-008-000	North Star	Perrigo	4% of proceeds, less transportation, processing, etc.
40-010-007-000	Wabash	Perrigo	4% of proceeds, less transportation, processing, etc.
40-010-012-000	Comet	TLC/MHC Ranch	4% of proceeds from ore, concentrates, doré or other forms of saleable product produced during mining operations
40-010-013-000	Golden Eagle/Triumph	Salcido	4% of proceeds from ore, concentrates, doré or other forms of saleable product produced during mining operations
40-010-018-501 & 502	Lincoln	TLC/MHC Ranch	4% of proceeds from ore, concentrates, doré or other forms of saleable product produced during mining operations. Only covers the surface down to 100ft.
		Chester Corp.	2.5% net smelter returns royalty
40-020-007-502	Mine House	Lundlee, Lubiens & Hallum	4% gross proceeds royalty

In addition to the royalties shown on this table, there is a 5% Net Profits Royalty on all SGM properties that is payable to U.S. Energy Corp. as described in text above, Also, in addition for the Comet, Golden Eagle, Lincoln (40-010-018-502) and Mine House parcels, there is a 0.5% NSR royalty payable to a consultant as described in text above.

Table 4.5 Royalty Obligations for the Parcels Containing the Keystone Resource

APN#	Name	Holder	Royalty
15-210-17-000	Keystone, S. Spring Hill, Medean &Herbertville	Keystone	5% Net Returns defined as: proceeds, less transportation, processing, etc., until payback, then increases to 6% Net Returns.
08-260-027-502	Spring Hill, Geneva, East Keystone, S. Spring Hill		
08-260-030-502	Spring Hill, Geneva, East Keystone, S. Spring Hill		
15-210-042-501	Medean		
15-210-044-501	South Spring Hill		
15-210-023-000	Ron Littlefield Parcel	Keystone	4% Net Returns defined as: proceeds, less transportation, processing, etc.

In addition to the royalties shown on this table, there is a 5% Net Profits Royalty on all SGM properties that is payable to U.S. Energy Corp. as described in text above, Also, in addition for the Comet, Golden Eagle, Lincoln (40-010-018-502) and Mine House parcels, there is a 0.5% NSR royalty payable to a consultant as described in text above.

4.3.1 Tailings Storage Property

SGM leases surface rights to the tailings disposal property ("Surface Fill Unit" in Table 4.1 and Table 4.3, and shown on Figure 4.2), as described by Ronning and Prenn (2004):

> *"In December 2002 Sutter Gold Mining Company signed a lease with two trusts allowing SGMC to use a parcel of land for tailings storage. The parcel is described as:*

Portion of the NE ¼ of the NW ¼ of Section 5, Township 6 North, Range 11 East, M.D.B.&M, and portion of the SE ¼ of the SW ¼ of Section 32, lying northerly and westerly of the existing dirt road, within the boundaries of the lands of Swift (the lessor), being no less than 30 acres of the Swift Ranch.'

At the time of the December 2002 lease agreement, the boundary of the leased area had not been surveyed. The agreement described plans to have it surveyed, but MDA has not determined whether that survey has yet taken place.

In order to maintain the lease on the tailings storage site, SGMC must pay $1,000 per month beginning December 1st 2003 and continuing while the lease is in effect but not in use for the purpose of storing tailings. Once the land is in use for storing tailings, the monthly fee becomes $2,000 per month. Beginning each December 1st, the amount payable is to increase according to the Consumer Price Index for the San Francisco area. The term of the lease is for ten years starting December 1st 2002, and is renewable for two additional ten year terms. SGMC is responsible for taxes and, when finished with the site, must leave it in a condition that is in compliance with all applicable regulations. SGMC is responsible for any post-closing monitoring that may be required."

In May of 2012 SGM provided written notice to the Lessors of SGM's intention to renew the lease for a another 10-year term. At the time SGM was engaged in purchase negotiations with the Lessor, but these negotiations did not result in a sale. In the summer of 2012, SGM constructed the access road to the Surface Fill Unit along an easement and removed trees from the Surface Fill Unit area in preparation for construction of Cell 1 of Phase 1 of the Surface Fill Unit. As of this writing SGM has not constructed Cell 1 of Phase 1 of the Surface Fill Unit, but they have increased the monthly payment as if tailings disposal had begun owing to construction of and use of the access road.

4.4 Environmental Permits and Potential Liabilities

The following discussion of environmental permits and potential liabilities relates to SGM's Lincoln Mine Project including the Lincoln-Comet portion of the project; more specifically, the 591 acres of land owned by or under control of SGM and its affiliates through leases or agreements that include the Lincoln-Comet and Keystone resource (Table 4.1). However, the following discussion does not include newer lease or land acquisitions described in Table 4.2 (approximately 294 acres) obtained for future exploration and not included in the permits, environmental review, or previous pre-feasibility studies.

MDA (Ronning and Prenn, 2004) reported that although the Lincoln-Comet project was an advanced exploration project, permitting for a mining operation had proceeded intermittently since the late 1980s. To a large extent, the permitting process is dictated by local (Amador County) and California permitting requirements and processes, with some involvement from federal agencies (e.g., the U.S. Army Corps of Engineers ("USACE") and the Department of Labor Mining Safety and Health Administration "MSHA")) that have not delegated permitting authority to the state and local government agencies.

This and the following three subsections draw upon information provided by SGM, including reports prepared by third parties working under contract to SGM or its affiliates. The information summarized in this and the following three subsections is based upon work by Golder Associates, Inc. ("Golder") and

Behre Dolbear and Company (USA), Inc. ("Behre Dolbear"). Both of these firms have appropriate expertise in the following areas:

- Environmental permitting and assessing potential liabilities;
- Environmental review and impact analysis;
- Environmental permit compliance; and,
- Reclamation, corrective action, closure and post-closure requirements and financial assurances.

More specifically, this and the following three subsections draw extensively from the following reports:

- Pre-Feasibility Study of the Sutter Creek Gold Mine prepared by Behre Dolbear and dated December 2007;
- Environmental and Regulatory Evaluation Sutter Gold Mining, Inc. Lincoln Project prepared by Golder and dated August 2007 (hereinafter Golder, 2007a); and
- Phase I Environmental Site Assessment Sutter Gold Mining, Inc. Lincoln Mine Project prepared by Golder and dated October 2007 (hereinafter Golder 2007b).

MDA is not an expert regarding environmental issues and presents this information with no opinion. Stephen Lofholm of Golder, to whom SGM has provided access to all appropriate documents and records, takes responsibility for Section 4.4.

4.4.1 Required Permits and Permitting Status

In 2007, SGM retained Golder to complete an Environmental Review and Regulatory Evaluation of the Lincoln-Comet project, and the resulting report (Golder 2007a) summarized the regulatory framework for the project as follows. The Lincoln-Comet project is subject to federal, state, and local environmental regulations and permitting requirements. As an approved state with respect to most federal programs (e.g., the Clean Water Act and the Clean Air Act), the federal government has delegated authority to the State of California for administration and enforcement of many federal requirements. California, in turn, has developed many of its own regulations that are as strict as, or more stringent than, their federal counterparts.

California administers and enforces environmental regulations and requirements through a number of state and regional agencies and, in some cases, local agencies to whom the state has delegated authority. Many environmental regulations in California fall under the jurisdiction of the California Environmental Protection Agency ("Cal EPA") which, in turn, delegates authority to subordinate state agencies. For example, the State Water Resources Control Board ("SWRCB"), under authority granted by Cal EPA, is responsible for protecting waters of the state, including groundwater and surface water. The SWRCB, in turn, delegates a portion of their authority to nine Regional Water Quality Control Boards ("RWQCB"s). Similarly U.S. EPA has delegated air quality responsibilities to the California Air Resources Board ("ARB") at the state level, and a series of regional Air Quality Management Districts ("AQMD"s) and Air Pollution Control Districts ("APCD"s) at the local level. Local AQMDs and APCDs issue permits, regulate emissions, and enforce regulations for stationary sources of air pollution. In addition, many local governments (cities and counties) develop their own local rules and permit requirements that affect operations like the Lincoln Mine project.

SGM's Lincoln Mine project, therefore, is subject to compliance with federal, state, and local laws, regulations, rules, and permits administered and issued primarily by state and local agencies. In some instances, however, the federal agencies become involved and maintain responsibility for specific aspects of a project. For example, the USACE has jurisdiction if a project impacts federal wetlands. With a few exceptions, however, most of SGM's Lincoln Mine project environmental permits and requirements will be administered by state and local agencies. Table 4.6 summarizes the federal, state, and local agencies that have permitting authority and responsibility over the Lincoln project.

Table 4.6 Agencies with Regulatory Authority Relevant to the Lincoln Mine Project
(Updated by SGM, 2015)

Federal Agencies
Army Corps of Engineers
Department of Labor, Mine Safety & Health Administration
Department of the Treasury (ATF)
Environmental Protection Agency
Fish and Wildlife Service
State Agencies
State Water Resources Control Board
Regional Water Quality Control Board
Office of Mine Reclamation
Department of Toxic Substance Control
Department of Fish and Game
Occupational Safety and Health Administration
Office of Emergency Services
California Highway Patrol
Office of Historic Preservation
Local (Amador County) Agencies
Air Pollution Control District
Amador County Planning Department (Lead Agency)
Building Department
Environmental Health Department
Fire Protection District
Public Works Department
Sheriff's Department

Golder (2007a) evaluated the permitting requirements for the Lincoln Mine project and, in so doing, identified two main categories of permits and approvals: major permits and approvals, and operating permits and approvals. In general, major permits and approvals are those that are very broad in nature or are intended to protect a unique resource (e.g., water, land or air), and typically require extensive applications and often complex supporting permit documents or reports and plans. In addition, major permits and approvals typically require some form of discretionary action on the part of the issuing or responsible agency. Such approvals or actions are often time consuming, requiring several months to years to complete. In addition, such actions often engage the public through one or more public meetings or hearings, where the public may comment or otherwise voice their opinions with regard to the project, the permit, or specific permit conditions. Typically a project proponent, in this case SGM, applies for and obtains major permits and approvals relatively early in the course of a project. The major permits and approvals often guide or chart the course to obtain subsequent permits and approvals, in that responsible agencies engaged during some of the initial permitting processes comment on the project and establish a framework for later major permits, as well as operating permits and approvals.

In contrast, operating permits and approvals typically focus on a specific aspect, activity, or element of the project and in some cases a specific piece of equipment. Operating permits and approvals require less extensive applications and supporting documentation, and in some cases may only require filing a notification with the appropriate agency. Operating permits and approvals typically require only days, weeks, or months to obtain. Typically, a project proponent like SGM will obtain many of their operating permits and approvals as needed, based on certain milestones in a project's development history, after acquiring many, if not all, of the major permits and approvals.

Golder (2007a) and Behre Dolbear (2007) concluded that SGM had obtained the major operating permits and approvals for the Lincoln Mine project. Table 4.7 summarizes the major permits and approvals already obtained by SGM for the Lincoln Mine project, which are also discussed below. These include the 1998 environmental review completed by Amador County pursuant to the California Environmental Quality Act ("CEQA") and the Conditional Use Permit ("CUP") adopted by the County. Previously in 1993, Amador County had completed the original environmental review for the Lincoln Mine project and adopted the 1993 CUP. The 1998 CUP and environmental review incorporated several improvements and changes to the Lincoln Mine project requested by SGM. Major permits and approvals already obtained also include two sets of Waste Discharge Requirements ("WDR"s) issued by the RWQCB. Issued in 1999, WDR Order No. 99-035 allows the discharge of treated mine water. WDR Order No. R5-2007-0006, obtained in 2007, regulates the discharge of waste rock and mill tailings. In 2005, the RWQCB issued the original WDRs regulating discharge of waste rock and mill tailings for the Lincoln Mine project. The 2007 WDRs are a revision of 2005 WDRs adopted by the RWQCB to update the project status, including acquisition of an additional parcel of land by SGM in 2006. The other major permit and approval already obtained by SGM is the Mine Reclamation Plan issued in 1999, which regulates reclamation of surface disturbances associated with underground mining activities at the Lincoln Mine project.

Table 4.7 Major Permits Already Obtained for the Lincoln Mine Project
(Updated by SGM, 2015)

Permit/Approval	ID #	Responsible Agency	Year Issued
Environmental Review	89052205 and 92042063	Amador County Planning Department	1998
Conditional Use Permit	UP-97; 7-4	Amador County Planning Department	1998
Waste Discharge Requirements	Order No. 99-035	Central Valley Regional Water Quality Control Board	1999
Waste Discharge Requirements	Order No. R5-2007-006	Central Valley Regional Water Quality Control Board	2007
Surface Mining Permit and Reclamation Plan and Bond	Mine ID # 91-03-006	Amador County Planning Department and State Office of Mine Reclamation	1999

Golder (2007a) and Behre Dolbear (2007) also discussed the operating permits and approvals required for the Lincoln Mine project, including those already obtained by SGM and required for the current operations, and those required for the 1,000 tpd mining and milling operations allowed in the 1998 CUP. In 2008, the law firm of Harris & Thompson (Reno, Nevada) also reviewed the current environmental permitting status as reported by Golder Associates (2007a), with the results supporting Golder's report (Thompson, 2008). Sutter began ramping up their permitting effort in 2010. In 2011 SGM obtained partial financing to construct the Lincoln Mine Project and began the design work necessary for project construction and to obtain the remaining operating permits and approvals. Table 4.8 summarizes the federal operating permits and approvals obtained for the Lincoln Mine project, and Table 4.9 summarizes the state and local operating permits and approvals obtained for the project. Table 4.10 summarizes additional permits that may be required for the project depending on future plans, but not yet obtained.

Table 4.8 Federal Operating Permits and Approvals Obtained for the Lincoln Mine Project
(Updated by SGM, 2015)

Permit/Approval	ID #	Responsible Agency	Issue or Revision Date
NPDES Permit (Industrial Stormwater Permit)	WDID 5S03I024115	State Water Resources Control Board	2013
Alternative Mine Rescue Plan	Mine ID #0405038	MSHA	2014
Legal Identity Report	Mine ID #0405038	MSHA	2015
MSHA Training Program	Mine ID #0405038	MSHA	2013
MSHA Annual Refresher Training	Mine ID #0405038	MSHA	2015
Escape and Ventilation Plan	Mine ID #0405038	MSHA	2015
Ventilation Plan	Mine ID #0405038	MSHA	2015
Surface Fire Fighting Plan	Mine ID #0405038	MSHA	2015
Radio Station Authorization	21749908	Federal Communications Commision	2012
Section 404 Clean Water Act Permit to Fill Wetlands	SPK-2008-01204	US Army Corps of Engineers	2012
Informal Endangered Species Consultation	81420-2011-I-0756-1	US Fish and Wildlife Service	2011

Table 4.9 State and Local Operating Permits and Approvals Obtained for the Lincoln Mine Project

(Updated by SGM, 2015)

Permit/Approval	ID #	Responsible Agency	Issue or Revision Date
NPDES Permit (Industrial Stormwater Permit)	WDID 5S03I024115	State Water Resources Control Board	2013
California DOT Encroachment Permit	1097-6MC-0749	California Department of Transportation	1998
Streambed Alteration Agreement	1600-2011-0086-R2	California Department of Fish and Wildlife	2012
Water Quality Certification	WDID#5B03CR00062	Central Valley Regional Water Quality control Board	2012
Permit to Operate Air Pressure Tank	A006865-13, A006866-13, L006825-13, L006826-13	CAL OSHA Pressure Vessel Unit	2013
Permit to Operate Diesel Engines Underground	D016-005-97M	CAL OSHA Mining and Tunneling	2015
Underground Classification	C009-005-01M Amended (Formerly 57-03-90)	CAL OSHA Mining and Tunneling	2011
CAL EPA ID Number	CAL000189686	CAL EPA Depart of Toxic Substances Control	1998
On-Site Sewage Dsiposal System Permit	11841	Amador County Environmental Health Department	2011
Hazardous Materials Business Plan	001168-15	Amador County Environmental Health Department	2015
2344 AGT Self Certified SPCC	0001267-15	Amador County Environmental Health Department	2015
Building Permits	33978, 33979, 34292, 34520, 34521, 34570, 34569, 34612, 34715	Amador County Building Department	2012
Grading Permits	GEO4188, GO4189, GO3414, GEO4180	Amador County Public Works Agency	2012
Authority to Construct	11-244-1, 11-244-2	Amador Air Pollution Control District	2015
County Encroachment Permits	11061, 12040	Amador County Transportation and Public Works	2012

Table 4.10 Operating Permits and Approvals Not Yet Obtained for the Lincoln Mine Project
(Updated by SGM, 2015)

Permit to Operate	Amador County Air Pollution Control District	Operation of stationary sources of air pollution following construction and testing
Section 106 Consultation Memorandum of Understanding	Office(s) of Historic Preservation	Mitigation of impacts to select cultural resources associated with Section 404 Permit
Explosives Permit	U.S. Department of Treasury Bureau of Alcohol, Tobacco and Firearms	Blasting
Blaster's License	Occupational Safety and Health Administration	Blasting
Building Permits	Amador County Planning Department	Construction or modification of structures
Grading Permits	Amador County Public Works Department	Future earthwork and drainge improvements
Class V Injection Well	US EPA	Underground Disposal of Tailings
Proof Load Test & Certification	Cal/OSHA	Overhead Crane Operation

As the project continues to develop, some existing permits may require updating or modification. SGM will identify and obtain these as needed.

4.4.2 Compliance with Permits and Approvals

Golder completed a Preliminary Compliance Review of SGM’s Lincoln Mine project as part of its regulatory and environmental review. The Compliance Review (Golder, 2007a) was based on Golder’s visual observations at the time of Golder’s reconnaissance, review of available records from Golder’s files, SGM’s files, and records reviewed at various local and state agencies. The results of Golder’s review (Golder, 2007a) indicate that SGM is in substantial compliance with existing major permits and project approvals, as well as existing operating permits and approvals for the Lincoln Mine project. In addition, this conclusion is confirmed, in part, by an independent review of SGM’s compliance with their CUP conducted by Resource Design Technology, Inc. (“Resource Design”) for the ACPD. According to Golder (2007a), in July, 2007, Resource Design concluded that SGM had met all of the requirements for all applicable conditions of approval required at the project’s current stage of operation at the time of Resource Design’s review.

During the course of the review, Golder did identify some relatively minor inconsistencies in some of the operating plans and permits. When Golder identified these issues, SGM took appropriate actions to promptly rectify the inconsistencies. Golder also identified a few instances where SGM submitted late or incomplete reports to the RWQCB. When these were pointed out to SGM either by the RWQCB or Golder, Golder determined that SGM was very responsive and took appropriate actions to correct the noted deficiencies in a timely manner. To Golder’s knowledge, the RWQCB has been satisfied with SGM’s responsiveness and has not imposed any fines or penalties (Behre Dolbear, 2007). The law firm of Harris and Thompson completed an independent review of Golder’s findings and came to similar conclusions (Thompson, 2008).

Periodically, various governmental and regulatory agencies inspect the Lincoln Mine project site for compliance with various environmental and permit requirements. These inspections serve as an additional and independent source of permit and regulatory compliance. During the 2014 calendar year, the Amador County Planning Department conducted their annual inspection as the lead Agency for the Surface Mining and Reclamation Act. In 2015 the Planning Department submitted their report to the state documenting substantial compliance with the approved Reclamation Plan for the project and confirming their approval of SGM's 2014 Reclamation Plan Financial Assurance Cost Estimate. The Amador Fire Protection District also inspected the project site in 2014, recommending establishment of a dry fire hydrant at the Sand Barn. SGM is in the process of completing this installation in the first half of 2015.

4.4.3 Environmental Review and Impact Analysis

As the lead agency for purposes of CEQA, the ACPD completed the environmental review and impact analysis for the Lincoln Mine project in two phases. ACPD completed the review and analysis of the original project in 1993. The Final Environmental Impact Report was then certified by the Amador County Board of Supervisors ("ACBOS") who subsequently issued a CUP for the Lincoln-Comet project, which was the project name at that time. In 1998, ACPD completed their environmental review and analysis of the project, and the ACBOS then certified the Final Supplemental Environmental Impact Report and subsequently issued a CUP for the revised project (incorporating changes to the project proposed by SGM), including appropriate mitigation measures.

Each environmental review by the County included an assessment of baseline environmental conditions, analysis of impacts that would result from the project, and formulation of mitigation measures needed to reduce significant impacts to levels less than significant. As a result of each review, the County found that even with mitigation, some impacts could not be reduced to a level less than significant. In approving the project each time, the County considered socio-economic factors and adopted a statement of overriding considerations finding that the benefits of the project outweighed the impacts that could not be reduced to less than significant levels.

According to Behre Dolbear (2007) and based on Golder's findings (2007a) identified above, in adopting their statement of overriding considerations, both in 1993 and again in 1998 for the Lincoln Mine project approval, the County cited several benefits that would result from the project and concluded that the benefits outweighed the impacts that could not be mitigated to less than significant levels. These benefits or socio-economic factors included:

- Wealth to Amador County
- Tax Revenues
- Jobs
- Recovery of a Valuable Mineral Resource
- Increased Supply of Domestically Produced Gold
- The Opportunity to Demonstrate an Environmentally Improved Mining Operation.

For each of the identified environmental impacts, the County identified one or more agencies that are responsible for monitoring the mitigation measures. In addition, the County provided for retention of an independent, professionally qualified mitigation monitor to be hired by the County at the expense of SGM to assist with monitoring the progress of mitigation measures (Behre Dolbear, 2007).

4.4.4 Known and Potential Environmental Liabilities

As part of a larger environmental due diligence effort in 2007, SGM also retained Golder to complete a Phase I Environmental Site Assessment of the properties associated with the Lincoln Mine project (Golder 2007b). Golder (2007b) summarized the purpose, methods, and results of their Phase I Environmental Site Assessment as follows. The purpose of the Phase I work was to identify, to the extent feasible, characterize recognized environmental conditions. Such conditions may be associated with, or be the source of, potential environmental liabilities. Golder's methods were consistent with those prescribed in the ASTM Practice E 1527-05 entitled, "Standard Practice for Site Assessments: Phase I Environmental Site Assessment Process", the U.S. EPA Rule entitled, "Standards and Practices for All Appropriate Inquiries; Final Rule" (AAI Rule, 40 CFR Part 312), and Golder's professional judgment. Golder's assessment revealed no evidence of recognized environmental conditions and identified no historical environmental conditions and no *de minimis* conditions in association with the properties included in the 574 acres of land included in their assessment (Golder, 2007b). SGM has one known, but funded environmental liability associated with reclamation of disturbed surface areas as required by SMARA that is discussed in Section 14.2.2. SGM has identified additional financial assurance requirements pursuant to Closure, Post-Closure and Corrective Action financial assurance requirements that will be triggered as the project is put into production. SGM has reviewed these requirements and maintains cost estimates which are discussed in Section 21.12.3.

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

5.1 Access

The Lincoln Mine project lies about 45mi east-southeast of Sacramento, California. The northwestern end of the property lies on the southern edge of the town of Amador City, and the southeastern end of the property lies in the southern edge of the town of Sutter Creek. The two towns are connected by State Highway 49, which lies parallel to and just west of the western boundary of the SGM property. There is also an extensive network of county roads in the area.

The project's mine site can be reached from Highway 49 by way of two paved county roads and a paved, private driveway.

5.2 Climate

This portion of the Sierra foothills has hot, dry summers and mild, rainy winters. According to National Weather Service data recorded in Sutter Creek, temperatures average a high of 90°F in summer and 55°F in winter (Behre Dolbear, 2007). The area receives about 320 days of sunshine annually. Annual precipitation in 2006 was 28.5in., mostly occurring October through March. Annual rainfall averages 22in (Stinnett *et al.*, 1993). The climate permits exploration and mining to be conducted on the property year round.

5.3 Local Resources and Infrastructure

The following description of local infrastructure is largely taken from Ronning and Prenn (2004) with additional information provided by SGM. See Section18.0 for information on the current project-specific infrastructure.

As of 2013, the population of Sutter Creek was 2,452. Amador City had a population of a little under 200. Current land use in the area is a mixture of agricultural and residential.

Lode mining was the original stimulus for development and was once the principal industry of the region. In recent years, government, tourism, and residential development have become predominant. Many of the county's 35,000 residents commute to jobs in Sacramento or coastal California. Nevertheless, workers with knowledge of mining still reside in the area, and there is at least one underground mining contractor in the vicinity. Utility services appear to be typical for a populated area in a developed part of the United States.

The 1993 Pincock Allen & Holt ("PAH") (Stinnett *et al.*, 1993) pre-feasibility study mentioned well water or city water as a source of water, but did not include any details about water supply. All of the old mine shafts are believed to be flooded. SGM sees evidence for a perched groundwater table in the weathered and more highly fractured zones extending to approximate depths of 40 and 90ft, respectively (SGM, electronic communication, February 2011; Lofholm and Cochrane, 2005). Below those depths, the Brower Creek metavolcanic rocks are relatively tight, and evidence supports a conceptual model wherein groundwater occurs in isolated fractures that are not interconnected to any degree. Hence, there is no real groundwater table or flow in the deeper bedrock, although potential gradients likely exist between fracture sets. The ground water accumulates in the mine workings owing to penetrations (portal, vent borehole, exploration borehole, fractures) of the overlying weathered zone. This is true for SGM's modern workings as well as historic workings nearby. Meteoric inflow from weathered near-surface bedrock and alluvium collects in the very low-permeability metavolcanic units only where excavation (historic mining) allows.

SGM controls surface rights on some of the patented mineral claims that comprise the property. The surface rights on the patented claims are probably sufficient for the surface facilities needed to service an underground mine.

Domestic water and wastewater services are provided to the property by the Amador Water System, and electrical service is provided by Pacific Gas & Electric (Behre Dolbear, 2007).

As described in Section 4.3.1, SGM leases a separate site for tailings storage. According to Ronning and Prenn (2004), when finished with this site, SGM must leave it in a condition that is in compliance with all applicable regulations and will be responsible for any post-closing monitoring that may be required.

5.4 Physiography

The Lincoln Mine project lies in the foothills of the western slope of the Sierra Nevada. Elevations range between 1,000 and 1,500ft asl. The topography consists of rolling hills that are covered with grass and scattered oak trees.

SGM reports that very little surface water is present on site. A small, seasonal creek runs adjacent to Stringbean Alley; all other drainages are located outside the area of planned operations.

6.0 HISTORY

6.1 Pre-1983 History and Production

The Lincoln Mine project lies in the central portion of California's historic Mother Lode. Although the full length of the western foothills of the Sierra Nevada is sometimes called "Mother Lode Country," technically the Mother Lode is a 120mi-long, one-mile-wide system of gold-quartz veins and mineralized schist and greenstone extending from the town of Mariposa north and northwest to northern El Dorado County (Clark, 1970). The most productive portion of the Mother Lode was the 10mi segment between Plymouth and Jackson in Amador County (Clark, 1970), which includes the Lincoln Mine project. The Lincoln Mine project covers about 3.6 mi of strike length of the Mother Lode vein systems.

The California gold rush began with discovery of placer gold at Sutter's Mill on the American River in 1848. The rich surface placers were largely exhausted by 1855 (Clark, 1970). Placer deposits in the vicinity of the Lincoln Mine project were relatively small (Ronning and Prenn, 2004). Buried Tertiary river channels also containing placer gold were mined underground and with hydraulic mining throughout the region, but hydraulic mining essentially stopped in 1884 by court decree that prohibited dumping of debris into rivers.

Mining of quartz veins on the Mother Lode began in 1849 in Mariposa County. Within Amador County, the first discovery of lode gold was made in 1851 on the site of what would become the Burke shaft of the Keystone mine, in the northern portion of what is now the Lincoln Mine property (Payne, 2008). Most of the important lode deposits in the Plymouth-Jackson belt were discovered during the 1850s. Within the Lincoln Mine project area, the South Spring Hill and Lincoln mines were first developed in 1851; the Keystone in 1853; and the Central Eureka in 1855 (Clark, 1970). The Keystone and South Spring Hill mines were major operators by 1875, with the Central Eureka and Lincoln Consolidated (Lincoln, Wildman, and Mahoney) becoming important in the 1880s and 1890s (Clark, 1970). From the 1890s until 1942, the Plymouth-Jackson belt was one of the more important gold-mining districts in the United States, producing an estimated $180 million (Clark, 1970) or approximately 7.8 million ounces (Ronning and Prenn, 2004) of gold. Lode mining was a major industry in this area for 90 years.

Due to various land, environmental and regulatory concerns, a number of mines were shut down in the early 1900s. Within the Lincoln Mine project area, the South Spring Hill mine closed in 1902, and Lincoln Consolidated closed in 1912 (Clark, 1970). But the Old Eureka and Central Eureka mines merged in 1924, ultimately reaching over 4,000ft in depth (Clark, 1970). All of the mines were shut down by government order soon after the start of World War II. Although the Central Eureka reopened in 1945, increased costs forced it to shut down again in 1958 (Wagner, 1970); in its final years, it was being mined at 150tpd, at a grade of 0.40oz Au/ton and a depth of 4,000ft (Payne, 2008). It was the last active major gold mine on the Mother Lode (Clark, 1970).

Table 6.1 lists the historical production of the eight major past-producing mines in what is now the Lincoln Mine project. These 3.4 million ounces of production represent about 25% of the entire Mother Lode lode gold production of 13.6 million ounces and about 44% of the production from the Jackson-Plymouth segment of the Mother Lode.

Table 6.1 Summary of Historic Gold Production from Major Mines of the Lincoln Mine Project Area
(From SGM)

Mine (North to South)	Reported Gold Production ($)	Calculated Gold Productic	Years of Production	Notes
Bunker Hill	$5,142,382	250,000	1853-1934	1
Original Amador	$3,500,000	169,500	1852-1937	2
Keystone Consolidated				
Keystone	$18,778,000	889,300	1851-1920; 1935-1942	3
South Spring Hill	$1,953,749	94,600	1878-1883; 1883-1894	4
Medean	$156,093	7,500	1894-1899	4
Talisman	$402,000	20,000	1854-1876; 1879-close	4
Wabash		8,000		5
Lincoln Consolidated				
Lincoln	$2,200,000	106,500	1851-close	6
Wildman & Mahoney	$3,270,269	158,200	1851-1887; 1887-1894; 1894-1901	7
Central Eureka (including Old Eureka)	$36,000,000	1,672,000	1852-1952	8
TOTAL	**$71,402,493**	**3,375,600**		

Notes:
1. Reported Production is from Logan (1934) and calculated production is based on $20.65/oz average gold price
2. Reported production from Carlson & Clark (1954) and Clark (1970) and calculated production assumes $20.65/oz
3. Reported production from Keystone Mine Vertical Cross Section (1929) and Carlson & Clark (1954); calculated production assumes $20.66/oz through 1920 and $34.42/oz beginning in 1935 (averages)
4. Reported production from Keystone Mine Vertical Cross Section (1929); calculated production assumes $20.66/oz average
5. Payne (2008)
6. Reported production from Bowen & Crippen (1948, Carlson and Clark (1954) and Clark (1970); calculated assumes averge $20.65/oz
7. Reported Production is from Logan (1934) and calculated production is based on $20.65/oz average gold price
8. Reported form Carlson & Clark (1954), Wagner (1970) and Clark (1970); calculated production assumes 90% at 20.67/oz and 10% at 34.42/oz

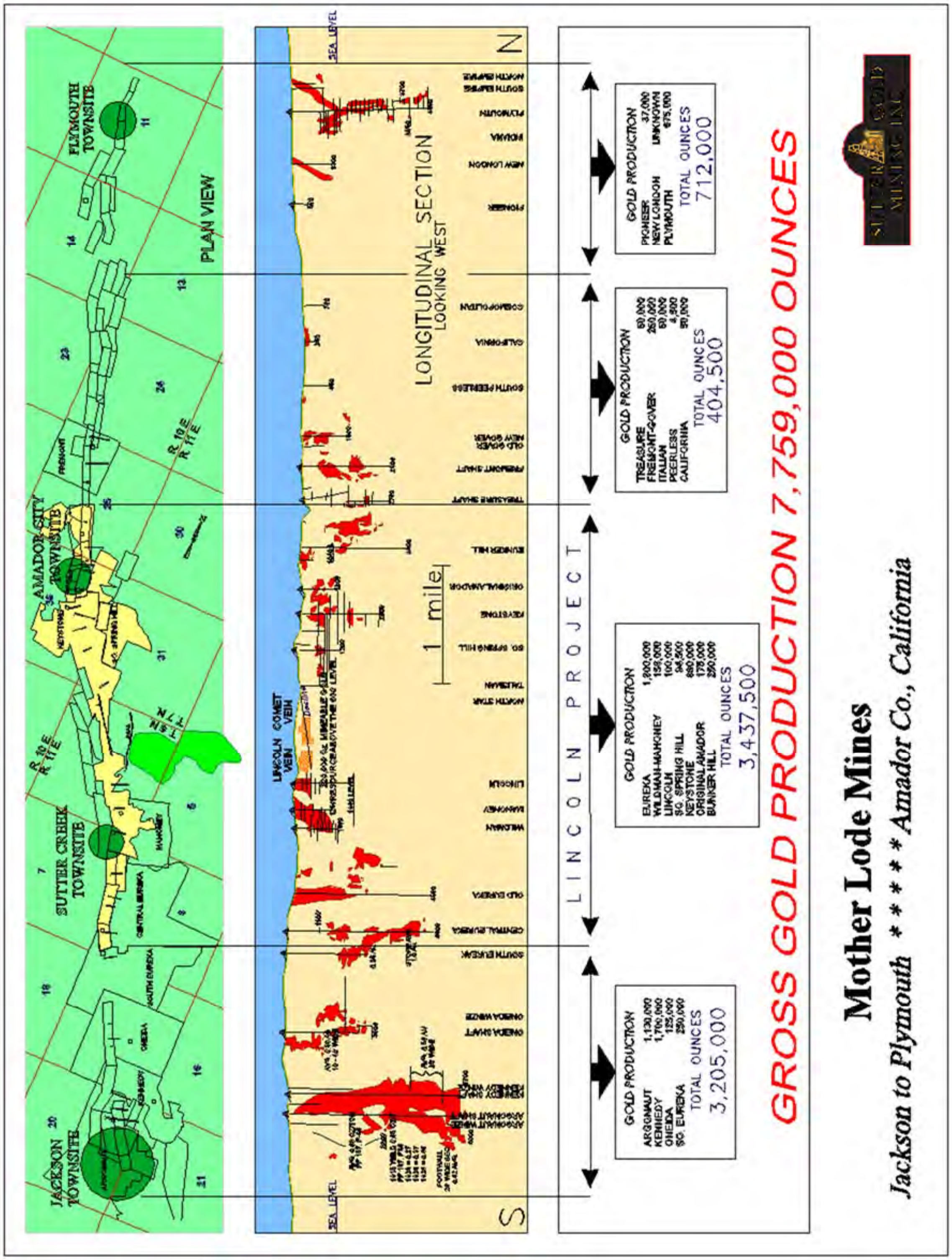
PAH (Armbrust, 1994) reported that a review of reports available to them had indicated none of the mines had closed due to lack of ore, but rather because of complex property ownership, litigation, insufficient land for tailings disposal, lack of capital, or regulations of World War II. Payne (2008) reached a similar conclusion citing references not reviewed by MDA:

> "*None of the mines in this part of the Mother Lode Belt were reported to be depleted of mineral resources when closed. The Central Eureka (1952), Wildman & Mahoney (1901), South Spring Hill (1902), and Keystone (1942) mines all had large resources reported in-place at their times of closure (Carlson & Clark, 1954). The Central Eureka mine allowed the deep workings of the Central Shaft to flood in 1930, abandoning three developed bodies at the 4850 Level, including a wedge-shaped sulfide replacement zone with a maximum width of 60 feet grading 0.36 oz/ton gold (Logan, 1934). In 1952, a large zone averaging 0.40 oz/ton remained open to depth below the 4150 Level in the Old Eureka Shaft workings when the mine closed. Farish (1901) reported two million tons of low grade mineralization at an undisclosed grade remained in place above the 1400 Level of the Wildman & Mahoney workings, while a cross cut on the 1400 Level exposed a 100 foot width grading 0.16 oz/ton in a zone that is open to depth. Several million tons of developed and undeveloped resources in the Keystone 5 Vein were reported to be in place when the Keystone Mine closed in 1920 (Meiklejohn, 1935). Keystone operations were resumed from 1936-1942 with the extraction of approximately 200,000 tons of*

ore from the K5 Vein above the 1000 Level. Large developed resources remain in place below the 1000 Level."

Little mining and exploration activity seem to have occurred during historic mining days in the Lincoln-Comet portion of the Lincoln Mine property, probably due to the absence of surface vein outcroppings on this portion of the property (Behre Dolbear, 2007). Substantial mining activity has taken place along the strike of the veins both north and south of the Lincoln-Comet, as described above. According to SGM staff, those mines were narrow vein operations, with ore grade shoots typically being vertical and extending to depth. Figure 6.1 shows a long section and plan view of the present Lincoln Mine project holdings in relation to past mining activities in Amador County.

Figure 6.1 Historic Mine Production in the Central Mother Lode
(Provided by SGM, 2011)



6.2 Post-1983 History and Exploration

The following description of exploration activity on the Lincoln Mine project since 1983 is taken from Ronning and Prenn (2004), Payne and Grunwald (2006), Payne (2008), Behre Dolbear (2007), and other references as cited.

Exploration since 1983 on the Lincoln Mine property has been localized around the Spring Hill, Talisman and Medean shafts north of the Stringbean Alley decline, and in the Lincoln and Comet zones. Exploration activities by SGM are described in Section 7.5.

Callahan Mining Corp. ("Callahan") was the first company to engage in modern exploration for gold on what is now the Lincoln Mine project. Following a favorable geologic evaluation of a 780ft stretch of ground immediately north of the inactive Lincoln mine, Callahan acquired the Lincoln and Golden Eagle properties in 1983 and the Lundlee and Comet properties in 1984 and 1985, respectively (Thomas and Chapman, 1987). The Lundlee property is now called the Mine House property and is so listed in Table 4.1. In 1983, Callahan drilled two reverse circulation ("RC") drill holes to test a strong arsenic soil anomaly north of the historic Lincoln Consolidated mines. In the same year, Callahan initiated exploration along the eastern contact vein of the Keystone mine and drilled seven RC and two core holes there from 1983 to 1987. In 1984-1985, Callahan drilled 13 core holes on the Lincoln property to follow up on strong initial results. Callahan's exploration also included detailed geologic mapping and high-density rock and soil sampling.

According to Thomas and Chapman (1987), Callahan and Pancana Minerals, Inc. ("Pancana") formed a joint venture in 1986 and continued definition drilling on the Lincoln property, drilling 15 core holes in 1986 (Payne, 2008). In addition to drilling on the Lincoln property, the joint venture excavated several short backhoe trenches and took channel samples in the vicinity of drill hole 11 (Thomas and Chapman, 1987); although shallow anomalous gold values were found in channel samples, drill hole 11 was drilled underneath one of the trenches, but did not find comparable mineralization. Callahan/Pancana's initial drilling defined a resource in the Lincoln zone that represented the first significant new gold discovery that was not related to past-producing mines made in the Mother Lode gold belt since the 1940s. The Lincoln-Comet vein system did not crop out and was a blind discovery within a one-mile-long, previously unproductive area bracketed by large historic gold producers. In addition, Callahan's drilling defined a gold resource along the Medean vein at the Keystone mine.

Meridian Gold Company ("Meridian") bought the Lincoln-Comet property in 1987 and the Keystone property in 1988 from Callahan-Pancana (Bright, 1990?). From 1987 to 1990, Meridian drilled 59 core holes from the surface that continued to define the Lincoln resource and discovered the Comet zone and a deeper zone in the Keystone 5 vein beneath the Comet. An additional 11 core holes delineated Indicated and Inferred resources along the eastern contact vein of the Keystone system. In 1989-1990, Meridian began underground development with the Stringbean Alley decline to explore the Comet mineralized zone. The decline was 2,885ft long, 12ft high, and 15ft wide with a minus 12 percent grade and included 2,400ft of cross-cuts. According to Ronning and Prenn (2004), it is reported that Meridian originally intended to drive the decline through the Comet zone into the area of the Lincoln mineralization identified by Meridian's drilling, but that Meridian terminated the underground exploration before reaching the Lincoln zone. Meridian conducted extensive chip sampling in its underground workings with at least 810 and perhaps as many as 836 samples collected. During this period, Meridian drilled 74 core holes from underground stations in the cross-cuts to further delineate resources in the Comet zone. In 1990, Meridian drove four development raises and 900ft of sublevel drift in the Lincoln 40, 41, 42, 43, and 50 veins of the Comet zone. They also collected an 8,119 ton bulk sample from several small test stopes that was milled at Meridian's nearby Royal Mountain King mine and mill.

Through its purchase of Meridian in 1990, FMC Gold Company ("FMC") acquired the Lincoln and Keystone properties. Later that year Seine River Resources Inc. ("Seine River"), in a joint venture ("the venture") with U.S. Energy Corp ("U.S. Energy"), purchased the project from FMC (Bright, 1990?; Stinnett *et al.*, 1993). In December 1990, Crested Corp. purchased one-ninth of U.S. Energy's interest in the venture. The venture continued the mine permitting process and conducted additional exploration and test work underground. In 1993, PAH completed a pre-feasibility study (Stinnett *et al.*, 1993), followed by a gold resource assessment in 1994 (Armbrust, 1994). In May 1994, U.S. Energy and Crested Corp. acquired 100 percent interest in the venture and incorporated Sutter Gold Mining Company to operate the project (Armbrust, 1994; Payne, 2008).

In 1998, the project obtained all necessary permits for mining and milling at a rate of up to 1,000tpd (Payne, 2008). According to SGM staff, in 1999 a mine reclamation plan was approved by the EPA and county agencies. Permitting activities continued, and SGM consolidated its property position to reduce advance royalty payments. In 2004, SGM leased the Central Eureka mine property (Railroad, Summit, and Amador parcel 18-270-013 on Table 4.2), which added a significant extension to known mineralized zones at the south end of the property.

In December 2004, Sutter Gold Mining Inc. completed a reverse take-over by acquiring Sutter Gold Mining Company. In August 2008, RMB Resources Ltd., a wholly-owned unit of the Rand Merchant Bank division of FirstRand Bank, (as Trustee for the Telluride Investment Trust) of Sydney, Australia, acquired a 49.9% interest in Sutter Gold Mining Inc. through the purchase of common shares from U.S. Energy and is now the major shareholder of Sutter Gold Mining Inc. (Alexander, 2009; SEDAR, 2008).

In 2009, SGM leased the Original Amador mine and Bunker Hill mine properties, adding a significant extension to known mineralized zones at the north end of their property.

6.3 Historical Mineral Resource and Mineral Reserve Estimates

The following information on historical mineral resource and reserve estimates for the Lincoln Mine project is presented for completeness only. Some of the estimates described in this section were prepared prior to establishment of NI 43-101 reporting requirements. Classifications are as described in the original references and do not necessarily meet the current NI 43-101 definitions. This information is presented solely for historical reference and purposes of disclosure and should not be relied upon. These historical mineral resource estimates are superseded by the current mineral resource estimates described in Section 14.0.

6.3.1 1985 and 1987 Callahan Mining Corporation's "Drill Indicated and Inferred Reserves"

Thomas and Chapman (1987) reported that from 1983 to 1985, Callahan developed "drill indicated and inferred reserves" totaling 110,000t grading 0.233+oz Au/ton. They further reported that as of 1987, Callahan/Pancana's "reserves" totaled 817,600 "geologic tons" grading 0.201oz Au/ton (uncut weighted average), including 543,362 "drill indicated and inferred tons" grading 0.214oz Au/ton (uncut weighted average). The "drill indicated plus inferred reserves" represented 980ft of strike and 550ft of dip; the "geologic reserves" covered 1,195ft of strike. The 1987 "reserves" were calculated by the standard polygonal method on six cross sections on 200ft intervals. A tonnage factor of 12 cubic ft/ton was used.

6.3.2 Meridian Gold Company's "Geologic Resource" Estimate

Meridian developed a "geologic resource" estimate of 421,470 tons averaging 0.249oz Au/ton for the Lincoln block in 1988 (Clarke, 1988). Of this, the "Main Zone" contained a "drill indicated resource" of 228,612 tons at a grade of 0.249oz Au/ton for 56,862oz and a "possible resource" of 129,705 tons averaging 0.201oz Au/ton for a total of 26,021oz of gold; total "drill indicated" and "possible" resources were 358,317 tons at a grade of 0.231oz Au/ton for a total of 82,883oz of gold. This estimate was based on data from 73 drill holes, of which the last 43 were Meridian core holes. The cutoff grade was 0.08oz Au/ton. No assay values were cut, and a tonnage factor of 12 cubic ft/ton was used. The minimum true width was 3ft.

According to Ronning and Prenn (2004), between September 1989 and August 1990, Meridian developed a polygonal resource estimate for the Lincoln and Comet mineralized zones, which they updated several times during that period. MDA has not seen any reports describing these estimates in detail, but Ronning and Prenn (2004) did review a series of spreadsheets prepared in 1989 and 1990 as well as drawings illustrating resource blocks from which they made inferences on the resource estimate. Ronning and Press (2004) reported only the most recent of the estimates, completed in August 1990, that used a cutoff grade of 0.15oz Au/ton and a minimum mineralized width of 4ft. MDA did not audit the 1990 Meridian estimate. Meridian identified "drill indicated" and "possible" "geologic resource" estimates as shown in Table 6.2.

Table 6.2 1990 "Geologic Resource" Estimate for the Lincoln and Comet Zones by Meridian Gold
(Cutoff grade used was 0.15oz Au/ton. From Ronning and Prenn, 2004)

Category	Tons	Gold Grade, Oz Au/ton	Ounces of Gold
"drill indicated"	267,348	0.412	110,164
"possible"	306,486	0.363	111,293

Notes: information from a spreadsheet titled "Lincoln Project Resource Estimate, Based on Longitudinal Projection, Total Geologic Resource" and dated August 28, 1990. Cutoff grade used was 0.15oz Au/ton, tonnage factor was 12.5 cubic ft/ton. Veins 1, 2, 3, 4, 5, 6, 7, and M or 40 veins were included. The estimates shown in this table include no allowance for dilution. The original spreadsheet also contains estimates for 10% and 20% dilution, assuming a dilutant of zero grade.

For the current report, MDA reviewed an undated pre-feasibility report by Bright that was written as Seine River was purchasing the property, which would date the report in 1990. In that report, Bright stated that as of August 1990, the "present resource" included 294,083t of "Drill Indicated" or "Probable Reserves" at a grade of 0.375oz Au/ton and "Possible Reserves" of 337,134t at a grade of 0.330oz Au/ton, using a cutoff grade of 0.15oz Au/ton (Table 6.3). It is not evident from the report whether Bright himself made this estimate or whether it came from Meridian/FMC, who held the property at the time. Bright says that the estimate is based on polygons on vertical longitudinal projections of composited vein drill and drift intercepts. A tonnage factor of 12.5 cubic ft/ton and 10% dilution were used for the estimates.

Table 6.3 1990 "Geologic Resource Inventory" for the "M Vein" in the Lincoln Area
(Cutoff grade used was 0.15oz Au/ton. Bright, 1990?)

Category	Tons	Gold Grade oz Au/ton	Ounces of Gold
Drill Indicated	294,083	0.375	110,164
Possible	337,134	0.330	111,235
Total Drill Indicated & Possible	631,217	0.351	221,458

6.3.3 1992 Mineral Resource Estimate for the Lincoln-Comet Area for U.S. Energy Corp.

In January 1992, Russell and Hazlitt reported a mineral resource estimate for the Lincoln and Comet zones and prepared a preliminary scoping study. The estimate is based on polygons on vertical longitudinal projections of composited vein drill and drift intercepts. The "Drill Indicated" classification is based on a 50ft radius of influence, and the "Possible" classification is based on a 100ft radius minus the Drill Indicated class. Narrow intercepts were diluted to a minimum mining width of 4ft. They used a tonnage factor of 12.5 cubic ft/ton and a cutoff grade of 0.150oz Au/ton. The resource estimate was not in accordance with NI 43-101, according to Payne (2008). The results of the resource estimate are shown in Table 6.4, with the terminology as used by Russell and Hazlitt (1992). MDA notes that the data on Table 6.4 are the same as those on Table 6.2.

Table 6.4 1992 "Geologic Resource Inventory" for the Lincoln-Comet Area
(Cutoff grade used was 0.150oz Au/ton. Russell and Hazlitt, 1992)

Resource Category	Average Grade (oz Au/ton)	Tonnage (short tons)	Total Ounces of Gold
Drill Indicated	0.412	267,348	110,164
Possible	0.363	306,486	111,293
Total Drill Indicated & Possible	*0.386*	*573,834*	*221,458*

6.3.4 1993 Pincock, Allen & Holt Resource Estimate

In the course of preparing a pre-feasibility study of the Lincoln Mine project in 1993, PAH completed a resource estimate of the Lincoln and Comet zones using drill-core intercepts and underground drift chip/channel samples taken by prior operators (Stinnett *et al.*, 1993). The resource estimate was made by the polygonal method. Table 6.5 shows their estimate using a 0.10oz Au/ton cutoff grade and their terminology (Stinnett *et al.*, 1993). PAH defined "proven" material as the material within 25ft of sample information, and "probable" as the material between 25 and 50ft of sample information. "Possible resources" were defined as material between 50 and 100ft of sample information. Assays were capped at 4oz Au/ton. The reader is referred to the original report (Stinnett *et al.*, 1993) for estimates at additional cutoff grades.

Table 6.5 1993 Geologic Resource Estimate of the Lincoln-Comet Area by Pincock, Allen & Holt
(Cutoff grade used was 0.10oz Au/ton. From Stinnett *et al.*, 1993)

Classification	Tons	Grade oz Au/ton	Ounces Gold
Proven	172,823	0.352	60,834
Probable	314,668	0.304	95,651
Total Proven and Probable	***487,491***	***0.321***	***156,485***
Possible	511,242	0.305	156,119
Total Proven, Probable, and Possible	998,733	0.313	312,603

In reviewing PAH's resource estimate, Ronning and Prenn (2004) noted that "*The Lincoln area represents about 1,000 ft of strike length (drilled) and contains about 48% of the Measured and Indicated resource tons, but only 36% of the contained ounces. The Comet area represents drilling along approximately 2,500 ft of strike length and contains about 41% of the Measured and Indicated resource tons, but about 57% of the contained ounces. The Keystone (vein 80) represents about 900 ft of strike length and contains about 11% of the Measured and Indicated resource, and about 7% of the contained ounces*."

6.3.5 1994 Pincock, Allen & Holt Resource Estimate

In 1994, PAH reviewed available data on the resource of the Lincoln Mine project, including an evaluation of their 1993 resource estimate (Stinnett *et al.*, 1993) and review of areas outside of those covered by their 1993 report, but within the Lincoln Mine property (Armbrust, 1994). Table 6.6 is a summary taken from their report that shows what they term "proven and probable mineable reserves," which are taken from PAH's 1993 report (Stinnett *et al.*, 1993), but at a cutoff grade of 0.25oz Au/ton. Table 6.6 also includes what they call "inferred resources" ("possible" in the 1993 report) of 192,600t grading 0.557oz Au/ton, which is also taken from the 1993 report, but at the 0.25oz Au/ton cutoff grade. These figures are for the Lincoln and Comet zones. Table 6.6 also includes resource estimates, termed "Additional Inferred Resources," for past-producing mines on the property (Keystone, South Spring Hill, and Medean) that are based on measurements for blocks at those mines taken from old reports. Because the methods used to estimate these tons and grades are not specified in available reports, Armbrust (1994) classified these as "inferred resources."

In reviewing this table that was included in their subsequent report, Ronning and Prenn (2004) noted that the potential one to two million ounces of gold that may exist below the old mine workings as mentioned in the table's notes "…*is only a potential quantity of mineralization. It is conceptual in nature, as there has been insufficient exploration to discover and define a mineral resource in the locations described in this paragraph. It is uncertain if further exploration will result in such a discovery in these specific areas.*"

Table 6.6 Summary of 1994 Pincock, Allen & Holt Reserves and Resources
(Cutoff grade used was 0.25oz Au/ton. Modified from Armbrust, 1994)

Category	Tons	Grade oz Au/ton	Contained Gold (ounces)
"Mineable Reserves"*	194,740	0.571	111,197
"Inferred Resources"*	192,600	0.557	107,278
"Inferred Resources"**	280,781	0.192	54,118
Subtotal	*668,121*	*0.408*	*272,593*
"Additional Inferred Resources"***	2,860,000	0.191	546,260
Notes: * Cutoff grade of 0.25 oz Au/ton ** Additional Inferred Resources at a cutoff grade of 0.15 oz Au/ton ***No cutoff grade. Based on actual measurements for blocks at the Keystone, South Spring Hill and Medean mines. Additional resources containing 1 to 2 million ounces of gold may exist below the old mine workings.			

Note: The so-called "Mineable Reserves" and "Inferred Resources" in this table are taken from PAH's 1993 resource estimate and are based on drilling and sampling conducted since 1983 (Stinnett *et al.*, 1993). The "Additional Inferred Resources" are based on historical records from the mines described in the notes within the table. What was considered a mineable reserve in 1994 can no longer be considered such, as the economic assumptions that would have applied then are now out of date.

6.3.6 2006 Lincoln Mine Project Mineral Resource Estimates

In March 2006, an updated, undiluted mineral resource estimate for the entire Lincoln Mine project was prepared in-house by Payne and Grunwald (2006). This mineral resource estimate conformed to the reporting requirements of NI 43-101, according to the authors. As was done in the PAH (Armbrust, 1994) estimate described above, the 2006 estimate, in addition to the Resource estimates based on drill data from Lincoln-Comet, also included Inferred Resources based upon historic mine documents for the Keystone, Lincoln Consolidated, Wildman & Mahoney, and Central Eureka mines; the latter three at the southern end of the Lincoln Mine property.

According to Behre Dolbear (2007), the PAH 1993 and 1994 estimates and the Payne and Grunwald (2006) estimate were all based on the same data collected from 1983 to 1992 from drill core, chip/channel samples, drill logs, and gold assays from drill core and chip/channel samples.

According to Payne and Grunwald (2006), the resources derived from modern exploration drilling were estimated by manual methods from vertical longitudinal projections constructed for each individual gold-quartz vein structure. A 3ft undiluted minimum thickness was applied to the individual resource blocks. Individual gold fire assays greater than 2.300oz Au/ton were cut to 2.300oz Au/ton prior to being composited. A cutoff grade of 0.140oz Au/ton was applied to each individual resource block. The resource estimates were undiluted.

The Inferred Resources estimated from historical data were based on 17 items including correspondence, consultants' reports, company annual and monthly reports, and government reports that date from 1876 to 1939 (Payne and Grunwald, 2006). However, the various historic workings are inaccessible, and there is a lack of specific information regarding methods used to estimate volumes and grades (Payne and Grunwald, 2006). Table 6.7 shows the Payne-Grunwald estimate.

Table 6.7 2006 Payne-Grunwald Mineral Resource Estimate for the Lincoln Mine Project
(Cutoff grade used was 0.140oz Au/ton. Modified from Payne and Grunwald, 2006)

	Tons	Grade (oz Au/ton)	Ounces of Au	Horizontal Width (ft)
Indicated Mineral Resources				
Lincoln zone	189,300	0.352	66,685	4.5
Comet zone	244,500	0.374	91,480	4.3
Lincoln-Comet area Subtotal	433,800	0.365	158,165	4.4
Keystone area	59,100	0.260	15,379	3.9
Total Indicated Resources	***492,900***	***0.352***	***173,544***	***4.3***
Inferred Mineral Resources A (estimated from modern exploration drilling and underground development work in the 1980s)				
Lincoln zone	62,600	0.330	20,656	3.7
Comet zone	59,100	0.361	21,360	3.4
Lincoln-Comet area Subtotal	121,700	0.345	42,016	3.6
Keystone area	51,700	0.232	12,001	4.5
Inferred Mineral Resources B (estimated from pre-1980s historical data)				
Keystone area	1,913,000	0.19	357,950	20.1
Central Eureka area	217,000	0.18	38,300	26.5
Lincoln-Wildman-Mahoney	394,000	0.18	71,100	66.8
Total Inferred B Resources	2,524,000	0.19	467,350	36.1
Inferred Mineral Resources A and B	***2,697,400***	***0.193***	***521,417***	***34.0***

Note: Payne and Grunwald's (2006) Table 17-4 with project totals does not have the same Inferred Mineral Resources B values as shown on their Table 17-3. MDA has used the data from their Tables 17-1, 17-2, and 17-3 and the totals shown on their Table 1, which are consistent.

6.3.7 2007 Lincoln Mine Project Mineral Resource Estimate

Following drilling in 2006-2007 to test "Inferred Resource" areas in the Comet zone and "historic Inferred Resources" in the Keystone mine area, an updated property-wide in-house mineral resource estimate was prepared early in 2008 (Table 6.8) (Payne, 2008). This estimate was based on data from core drilling by prior operators, chip/channel sampling, and 2006 core drilling by SGM. Payne's mineral resource estimates for the Lincoln Mine project included "Indicated Mineral Resources" and "Inferred Mineral Resources A" for the Lincoln-Comet and Keystone areas based on modern exploration drilling and "Inferred Mineral Resources B" estimated from pre-1980s historical data for the Keystone, Central Eureka, and Lincoln-Wildman-Mahoney areas. The resource estimate included geologic and assay data from the 19,502ft of 2006-2007 surface and underground drilling. Payne estimated the mineral resources manually using traditional longitudinal sections and polygonal composited assay-geology domains. The estimate is undiluted and uncut. A cutoff of 0.14oz Au/ton was applied to individual blocks. Payne reported also using CIM Definitions for Indicated and Inferred Mineral Resources. His report gives additional details on the assumptions and methods used.

Table 6.8 2007 Mineral Resources Estimate for the Lincoln Mine Project
(Cutoff grade used was 0.14oz Au/ton. From Payne, 2008)

	Tons	Uncut Grade (oz Au/ton)	Ounces of Au	Horizontal Width (ft)
Indicated Mineral Resources				
Lincoln-Comet area	511,700	0.37	188,481	4.4
Keystone area	161,900	0.21	34,563	8.5
Total Indicated Resources	***673,600***	***0.33***	***223,044***	***5.4***
Inferred Mineral Resources A (estimated from modern exploration drilling)				
Lincoln-Comet area	194,100	0.28	53,986	4.3
Keystone area	559,800	0.20	110,778	26.6
Inferred Mineral Resources B (estimated from pre-1980s historical data)				
Keystone area	1,013,000	0.18	183,950	27.1
Central Eureka area	217,000	0.18	39,100	26.5
Lincoln-Wildman-Mahoney	394,000	0.18	71,100	66.8
Total Inferred Resources	***2,377,900***	***0.19***	***458,914***	***31.6***

It should be noted that the Behre Dolbear (2007) report, which used Payne's resource estimate as of August 2007, reported slightly different numbers for the Inferred Resource at Lincoln-Comet – 192,200 tons rather than 194,100 tons and 53,587oz of contained gold rather than 53,986oz of contained gold as shown in Table 6.8. MDA cannot account for this difference, but Payne may have slightly revised his estimate prior to the 2008 report's completion.

In comparing the 2007 Payne estimate with the 2006 Payne and Grunwald estimate, Behre Dolbear (2007) made the following observations:

> *"A comparison of the 2006 Payne-Grunwald resource estimates with the 2007 Payne updated estimates shows that the tons have increased by 18 percent, the grade has decreased by 12 percent, and the contained ounces have increased by only 2.5 percent with the addition of the 9,070 feet of 2006 core drilling. The average horizontal width (4.4 feet) of the Indicated resource blocks has not changed. Therefore, the increased tonnage comes from an increase in the total area within Indicated blocks.*
>
> *…Behre Dolbear recommends that a statistical comparison of the 2006 assays with the historical assays be performed to better evaluate the extent of, and perhaps mitigate, its concern with the grades of the historical assays."*

Behre Dolbear (2007) also compared the two PAH estimates of 1993 and 1994 with the two Payne estimates (Payne and Grunwald, 2006; Payne, 2008) and identified the following differences:

- Payne's 2007 estimate, reported in 2008, includes SGM drilling that was not included in the other three estimates.
- Payne's estimates were based on a detailed geologic model of the veins.
- The PAH estimates were based on a model of east-dipping veins, but Payne and Grunwald determined that most of the veins dip west. Many of the pre-2006 holes were angle holes drilled at a 240° azimuth. This resulted in the holes intercepting the veins at shallow angles, producing mineralized intervals that were much longer than the true width. PAH used a vein thickness of 9.7ft for the 1994 estimate, compared to a vein thickness of 4.4ft for the two Payne estimates.

7.0 GEOLOGIC SETTING AND MINERALIZATION

7.1 Regional Geology

The Sierra Nevada, a 400mi-long mountain range that trends north-northwest along the eastern border of California, separates the Basin and Range Province on the east in Nevada and Utah from California's Great Valley to the west. The Sierra Nevada range is dominated by a huge granodioritic batholith that intruded older metamorphic rocks during the Jurassic-Cretaceous.

The metamorphic rocks occur largely along the western foothills (commonly called the Foothills Metamorphic Belt) and in the northern end of the Sierra in a northwesterly trending belt. Clark (1970) described the following major metamorphic rock units that were intruded by the Sierra Nevada batholith:

Calaveras Formation – slates, phyllites, schists, quartzites, hornfels, and limestones of Carboniferous to Permian age
Amador Group – metasedimentary and metavolcanic rocks of Middle and Upper Jurassic age
Mariposa Formation – slate of Upper Jurassic age
Kernville Series – schists, phyllites, and quartzites of Jurassic or older age found in the southern Sierra Nevada, and
Undifferentiated pre-Cretaceous greenstones and amphibolites.

In addition to the above units, there are numerous intrusions of basic and ultrabasic rocks, many of which are serpentinized. The serpentine bodies are often parallel to or occur within the belts of gold mineralization and may have been structurally important in the localization of some gold deposits (Clark, 1970).

More recent interpretations suggest that the above units were formed in the Pacific Basin and subsequently accreted to the western margin of North America from Paleozoic to Jurassic times (Ronning and Prenn, 2004; Irwin, 2004). Folding, faulting, and shearing occurred during the late Jurassic and early Cretaceous as did formation of gold-quartz veins. The Melones fault zone, which separates Paleozoic rocks on the east from Jurassic rocks on the west, is a regional structure that is located about 0.5mi east of and parallel to the Mother Lode. According to Payne (2008), very little historic gold production came from the Melones fault zone directly.

The Mother Lode is a 120mi-long system of linked or *en echelon* gold-quartz veins and mineralized schist that extends north-northwest from Mormon Bar in Mariposa County on the south to northern El Dorado County on the north (Koschmann and Bergendahl, 1968; Clark, 1970). The Mother Lode is 1 to 4mi wide and is hosted by Jurassic rocks. Extensive systems of gold-bearing veins are also found in two parallel belts lying east and west of the Mother Lode, called the East Gold Belt and West Gold Belt. These belts are shorter and less continuous than the Mother Lode and may be separated from it by 5 to 15mi of unmineralized country rock. Although genetically and mineralogically similar to the Mother Lode, the East Belt and West Belt production has been reported separately (Koschmann and Bergendahl, 1968). Gold from the quartz veins and mineralized country rocks was eroded and re-deposited to form both the Tertiary and Quaternary placer deposits that initially fueled the California gold rush.

7.2 Local Geology

The Lincoln Mine project is located in Amador County within the central part of the Mother Lode gold belt. The 10mi-long section lying between the towns of Jackson on the south and Plymouth on the north and including the Lincoln Mine project area was the most productive portion of the Mother Lode (Clark, 1970). Figure 7.1 shows the generalized stratigraphy of the Mother Lode region between Plymouth and Jackson.

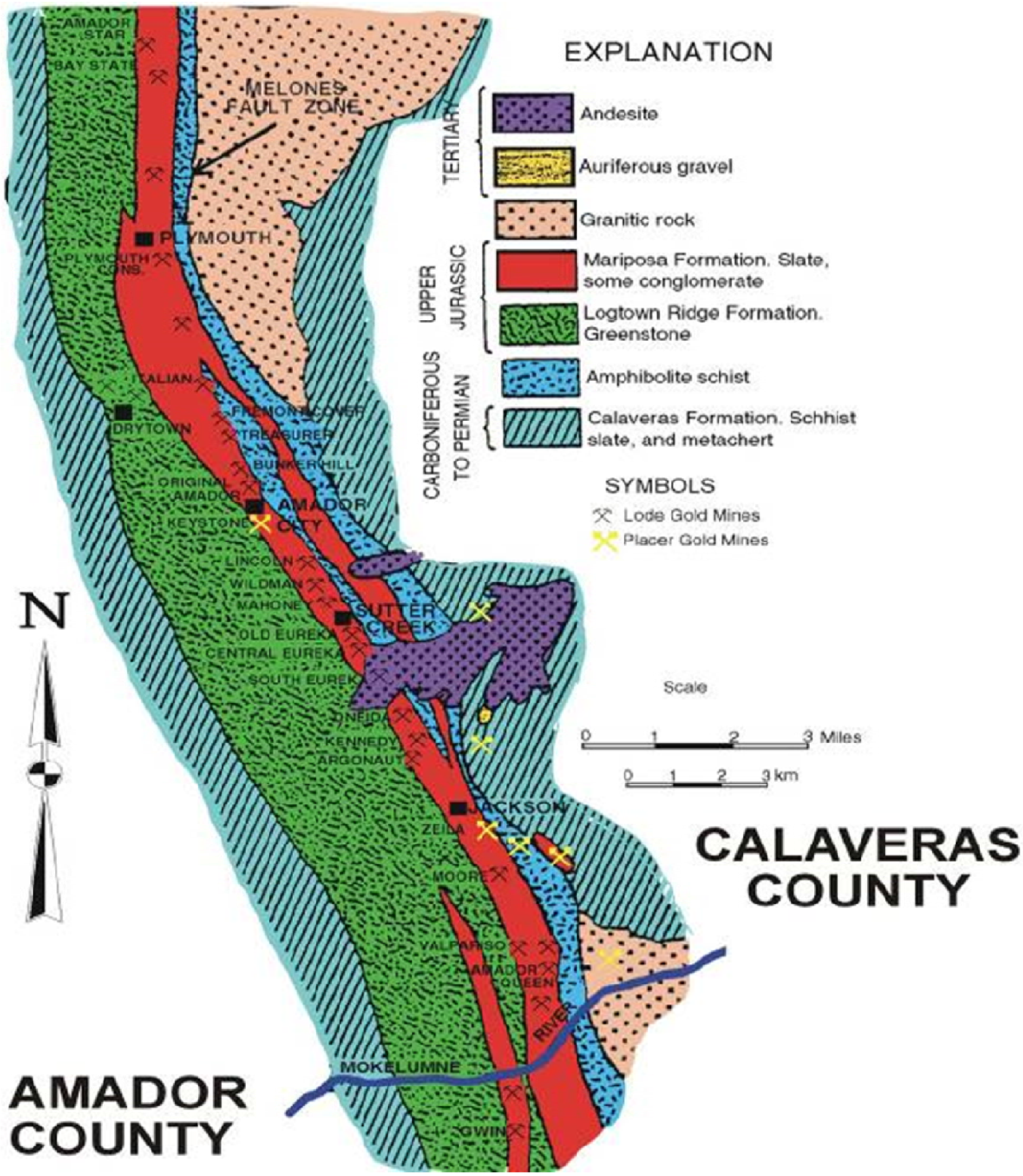
As described by Clark (1970), the gold deposits lie in a 1mi-wide, north- to northwest-trending belt of gray to black slate of the Mariposa Formation that also contains some interbedded coarse and occasionally sheared conglomerate, minor sandy and gritty layers, and localized metavolcanic and metasedimentary units. To the west of the Mariposa Formation is the massive sequence of altered mafic volcanic "greenstone" of the Logtown Ridge Formation. To the east are metasedimentary rocks of the Calaveras Formation, chiefly graphitic schist, metachert, and amphibolite schist.

Ronning and Prenn (2004) cite Zimmerman (1983), who described three types of mafic sills and plugs that have intruded the Jurassic section in the Lincoln Mine region:

1. Coarse-grained plagioclase-augite porphyry sills crop out northeast of the Lincoln-Comet resource area and near the Eureka mines, immediately to the south of the Lincoln-Comet.
2. Thin augite porphyry bodies crop out northeast of the Talisman shaft and south of the Niagara Mine adit, which is north of the Lincoln-Comet resource area. The best exposures are north of the Kennedy mine in Jackson, south of the Lincoln Mine project property. There, these rocks form sill-like bodies 10 to 33ft thick, moderately discordant to the host rocks.
3. There is a poorly defined intrusive plug of massive greenstone north of the Oneida shaft, which is south of the Lincoln Mine project property.

The dominant structure along the Amador County portion of the Mother Lode is the Gold fault zone, which is a braided corridor of high strain that is a branch from the regional Melones fault zone. It forms a footwall cymoid loop extending along an arcuate path and joining the Melones fault at both ends at Jackson and Plymouth. The Gold fault zone hosts all of the large productive mines in the Jackson-Plymouth belt (Payne, 2008). According to Payne (2008), "*Fluid channels for gold mineralization developed as the result of a mid-Cretaceous flattening event, which effectively developed district- and mine-scale boudinage features between rock units of contrasting competency. At the district-scale, and in detail, the gold mineralization is localized in and adjacent to faults developed along the contacts and within boudinaged meta-volcanic units.*"

Figure 7.1 Generalized Geologic Map of the Plymouth-Jackson Region
(From Campodonic, 2000)



7.3 Property Geology

The Lincoln Mine property lies within the Jackson-Plymouth portion of the Mother Lode, making up a 3.6 mi-long segment between the historic mining towns of Amador City and Sutter Creek in Amador County. The following description of the geology is taken from Irwin (2004), Behre Dolbear (2007), and Payne (2008) with additional information as cited.

The Lincoln Mine project is underlain by rocks of the Late Jurassic Mariposa Formation that lie west of the Melones fault zone. The Mariposa Formation consists of a basal slate and greywacke unit on the western margin of the property, overlain to the east by a thin metavolcanic unit, and finally overlain further to the east by an upper metavolcaniclastic and metasedimentary sequence. The rock sequence strikes N20-40°W and dips steeply east to vertical. The contacts between the three units are interpreted to be fault contacts.

The western slate unit consists of a monotonous sequence of laminated black graphitic slates and thin, fine-grained, dark gray metagreywacke beds. There are also thin, uncommon beds of metabasaltic flows and tuffs. The relatively narrow, competent metavolcanic unit that overlies the western slate unit consists of basaltic to andesitic metavolcanic flows and tuffs with subordinate inter-flow metasedimentary rocks. The metavolcanic unit, which has been correlated with the Brower Creek Member of the Mariposa Formation, is the host rock for most of the current Lincoln-Comet resource and portions of the nearby Keystone mineralization. Within historic reports, the term "greenstone" is often used for the Brower metavolcanic units. Greenstone refers to an undifferentiated sequence of weakly metamorphosed mafic volcanics and volcanic sediments. The Brower Creek tuffaceous units tend to accommodate strain and provide favorable pathways for alteration and gold mineralization. The eastern contact of the metavolcanic unit is a vertical to steeply west-dipping faulted contact with local bench-like, moderately east- and west-dipping sections.

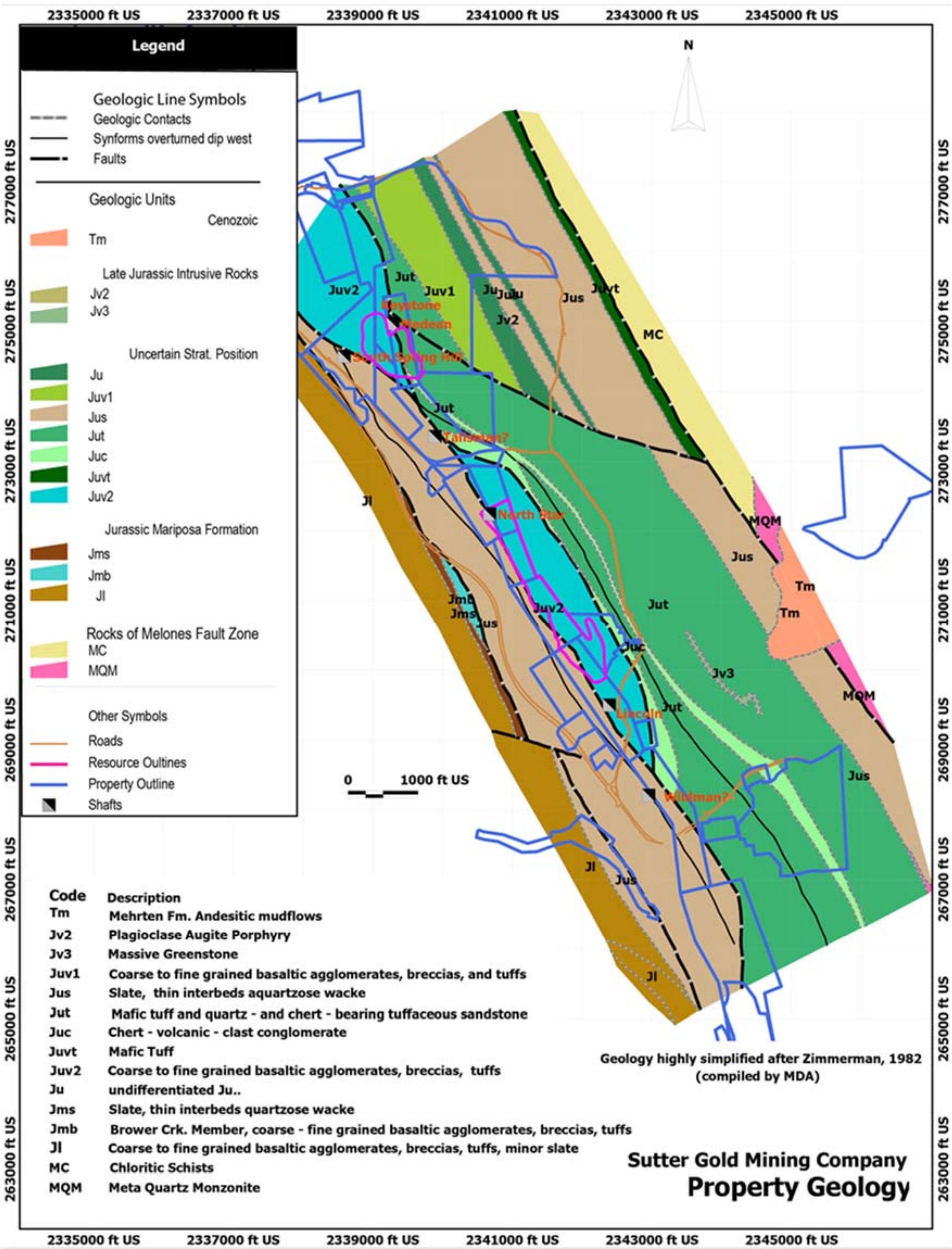
The eastern side of the property is underlain by the upper metavolcaniclastic and epiclastic unit, which hosts the southeastern part of the Lincoln-Comet resource and the eastern part of the Keystone resource. The unit is a thick, featureless sequence of thin- to medium-bedded reworked mafic tuffs, fragmental tuffs, graphitic tuffaceous slates and greywackes, and local mafic to intermediate flows. Soft-sediment deformation features, commonly seen in drill core, have been hard to differentiate from highly-strained phyllonitized zones associated with the gold-quartz veins.

The dominant structural feature on the property is the Gold fault zone. The fault zone strikes N20-40°W and dips moderately to steeply eastward. The fault zone is up to 1,500ft wide. Two branches – the East vein and the West Contact ("West") vein – define the boundaries of the Gold fault zone and are the primary hosts for the quartz veins and historic gold deposits within the Lincoln Mine project area. Subordinate structural features include resistant lensoid masses of metavolcanic rocks, deformational features affecting the lensoid structural blocks, and extensional fault arrays that are either developed between lensoid structural blocks within the Gold fault zone, such as the Lincoln structural block which hosts the Lincoln-Comet resource, or between the Gold and Melones fault zones. The sheared rocks that were formed by the high strain within the fault zones are called phyllonites. Nearly all significant gold mineralization on the property is related to deformation or dislocation along contacts between metasedimentary and metavolcanic rocks where the contacts are faulted. According to Payne (2008), "*At the property-scale, three structural trap types associated with lensoid meta-volcanic blocks localize high-grade shoots at the Sutter Gold Project. They include (1) strain shadows around lensoid boudin* *blocks, (2) east-dipping brittle faults which have segmented the boudin blocks, and (3) favorable tuffaceous stratigraphy within the lensoid boudin blocks, which accommodate strain and host mineralization."*

The Comet and Lincoln mineralized veins occur within fault arrays in the Lincoln structural block within the Gold fault zone. The two zones are localized primarily just east of the West vein within the hard Brower Creek metavolcanic rocks, and along the West vein fault contact of the Brower Creek member and the basal slate unit (Behre Dolbear, 2007). The Comet and Lincoln zones trend N30°W and generally dip steeply west at an average of 70°. There are also minor east-dipping fault/vein structures which extend through the Lincoln block and appear to localize mineralization within the west-dipping veins.

Figure 7.2 shows the geology of the Lincoln Mine property.

Figure 7.2 Geology of the Lincoln Mine property
(Modified from Ronning and Prenn, 2004)



7.4 Mineralization

This section describes mineralization on those portions of the Lincoln Mine property that have been the focus of exploration since 1983. The following information is taken from Payne (2008), Behre Dolbear (2007), Payne and Grunwald (2006), Ronning and Prenn (2004), Irwin (2004), and other references as cited.

Mineralization on the Lincoln Mine project is typical of that of the Mother Lode in general, i.e. gold-quartz-ankerite veins with free gold and 1-2% accessory sulfides. The veins cut the Jurassic Mariposa Formation and are controlled by shear zones. Typically the strongest gold mineralization is found in distinct dilation zones along the veins. Mother Lode deposits are characterized by their extensive horizontal and vertical continuity. Mineralized shoots within the main veins of the district may extend down plunge many times their strike length; some of these deposits have been mined to depths of 5,900ft from the surface. Within the Lincoln-Comet area, horizontal continuities of geology and grade are greater than vertical continuities; horizontal continuity is anomalous in the Lincoln-Comet deposit compared to most of the mined deposits along strike. However, Behre Dolbear (2007) notes that there is no information available for the Lincoln-Comet vein zones below about 800ft from the surface, and deeper exploration could show different continuity of the mineralized shoots.

Nearly all significant gold mineralization on the property is related to pronounced deformations or dislocations along metasediment/metavolcanic contacts. The favorable bends in the contacts are smaller-scale manifestations resulting from large-scale deformation and boudinage affecting the metavolcanic units within the Gold fault zone. The dilation zones have strike lengths ranging from a few tens of feet to as much as 400ft. Individual metavolcanic blocks show a pinching-swelling character in response to compressional flattening and subsequent brittle relaxation, which has resulted in a recurring brittle boudinage fracturing pattern within the blocks. Much of the 3.4 million ounces of historic gold production from the Lincoln Mine property is attributable to recurring brittle boudinage fracture patterns localized within, alongside, and exterior to the competent metavolcanic blocks.

The gold-quartz veins branch and anastomose along the 3.6mi length of the Lincoln Mine property, both macroscopically and microscopically. The gold mineralization was emplaced along through going, geologically continuous faults or corridors of high strain. A pervasive fault overprint is ubiquitous in mineralized areas. With few exceptions, ribboned vein quartz that exceeds 1ft in width and that is cut by a strong fault overprint occurs within mineralized shoots. Within mineralized shoots, the typical gold-quartz vein in the Lincoln Mine area is bounded by strong, overprinting fault slip planes that generally define one or both walls. Petrologic studies indicate that the quartz present within a mineralized shoot commonly exhibits cataclastic texture.

The primary gold-bearing quartz-ankerite veins of the Comet and Lincoln zones trend N30°W and generally dip steeply west at an average of 70 degrees. The west-dipping veins often terminate against shallow-dipping, east-dipping fault/vein structures which serve as structural traps for mineralization. The steeply west-dipping Comet and Lincoln veins are anomalous in that most of the other quartz veins in the West and East vein systems in the project area dip east. The other exceptions to the generally east dip of the veins in this area are the moderately southwestward-dipping Keystone 14 and 15 veins hosted along the Whiskey fault, and the westward-dipping Belmont vein located along the western contact of the Emerson Block. Ronning and Prenn (2004) noted another distinction in the Comet and Lincoln zones in that "*there is some evidence that the mineralized shoots in the Comet and Lincoln zones in fact* *have gentle to sub-horizontal plunges, different from those of the main veins (Campodonic, 2000…).*" Payne (2009, written communication) reports that "*the observed gentle southeastward plunge of the Lincoln-Comet shoots reflects the shallowly plunging line of intersection of the west-dipping Lincoln-Comet veins and (1) east-dipping fault/veins, and (2) more importantly the near-vertical volcanic stratigraphy. The rock competency contrast between volcanic flows and bedded tuffs controls the path, local bending and branching of the Lincoln-Comet vein system. The east-dipping fault veins are structural traps and in the case of the 20 Vein, terminate the tops of some of the shoots*."

The Lincoln Mine veins consist of variably tectonized quartz and ankerite with 1 to 2% fine-grained sulfides, dominantly pyrite, with minor quantities of arsenopyrite. Arsenic concentrations in the Lincoln Mine project's mineralization are on the order of 1,000 to 10,000ppm. The silver to gold ratio is low, and the purity of the gold is greater than 800 fine. In the Lincoln-Comet deposits, about 20% of the gold occurs as coarse grains up to 1/8in. in size. The coarse nature of the gold results in a significant "nugget" effect. Gold mineralization can be highly erratic in grade, both on a sample scale and along strike within the individual veins. Gold grades of >1oz Au/ton can quickly transition to <0.1oz Au/ton over just a few feet along strike.

While the quartz vein mineralization is the dominant mineral style in the Lincoln Mine zones, gold mineralization in the project area also occurs in hydrothermally altered greenstone of the Brower Creek Member. This mineralization is historically known as "gray ore" in the Mother Lode belt and as "sulfide replacement mineralization" on the project. The sulfide replacement mineralization consists of strongly altered metavolcanic rocks cut by varying amounts of thin quartz veining and makes up approximately 5% by volume of the Lincoln-Comet mineralization. The alteration consists of complete, or nearly complete, replacement of the metavolcanic rocks by ankerite, albite, and sericite, plus 2 to 3% fine-grained sulfides. The sulfide replacement mineralization generally has higher total sulfide content, but lower average gold grades than the gold veins. The sulfide replacement mineralization on the Lincoln Mine property universally occurs where vein structures bend, propagating vein splits. The auriferous replacement deposits occur within the intervening wedge of rock between the two veins and can constitute large-tonnage, bulk-minable deposits with maximum widths of 20ft at Lincoln-Comet and 45ft at Keystone.

Modern surface core drilling since 1983 has delineated resources in an extensive gold-quartz vein system in the Lincoln and Comet zones. Modern surface core drilling in the Keystone mine area has delineated resources along two primary gold-quartz veins (the K5 and K13 veins as designated by SGM) and partially tested a number of smaller vein systems developed along both contacts of the Lincoln zone in the areas of the historic South Spring Hill, Talisman, and Medean shafts. Core drilling has been a cost-effective method to test the vein systems, and underground development with closely spaced drill holes has further defined and demonstrated continuity of the mineralized zones in the Lincoln-Comet area. The underground development work was terminated when FMC acquired Meridian in 1990. Chip sampling of underground exposures within the development workings has added further definition and demonstrated good continuity of the gold-quartz vein shoots in the Lincoln and Comet zones.

According to Russell and Hazlitt (1992), there is an association between arsenopyrite and gold in the Lincoln Mine area, and arsenic and gold are used as indicators in both drill sampling and surface soil geochemical sampling.. They reported that typically any rock that contains 1,000 ppm arsenic will also contain at least 0.02oz Au/ton. Zahony (2010) describes the relationship between gold and arsenopyrite as follows: "*Gold correlates with arsenopyrite, but not perfectly. Gold correlates with the amount of* *total estimated sulfides in quartz veins, that is, both pyrite and arsenopyrite, but not perfectly. There seem to be exceptions to each elemental or mineralogical correlation of vein minerals with gold.*

Payne (2008) identified an inventory of 14 exploration targets within the Lincoln Mine property, including five within the Lincoln-Comet area and the remainder to the north and south. The reader is referred to Payne's (2008) report for details on the individual targets. Ronning and Prenn (2004) also had considerable discussion on exploration targets within the Lincoln Mine property that is not repeated here.

7.5 Mineralized Zones with Recent Exploration

Four mineralized zones on the Lincoln Mine property have been evaluated by modern exploration conducted by SGM and its predecessor operators since 1983. These descriptions are taken from Stinnett *et al.* (1993), Ronning and Prenn (2004), and Irwin (2004), with updates resulting from the current resource model.

The four zones described below host the mineral resources as described in this technical report. The Lincoln zone lies approximately 1000ft to the south of and at a generally higher elevation than the Comet zone while the two Keystone zones (Medean and South Spring Hill) lie about 2,000ft to the north of the Comet with the South Spring Hill zone generally at a lower elevation than the Comet.

The use of the term "Keystone" for the area explored by SGM and its predecessors, north of Stringbean Alley and the portal of the Stringbean Alley decline in the 1980s and 1990s, is somewhat inaccurate. Most of the modern exploration was neither on the Keystone property, nor on veins that would historically have been exploited by the Keystone operation. However, the term Keystone has been used since 1983 to refer to the drilling and other exploration done on the northern part of the property (e. g., drill holes that are prefixed with "K"). In this report, the term "Keystone" is maintained for continuity with the recent past, but Medean and South Spring Hill are also used, as it more accurately reflects the historical name of the vein system that was explored.

The Comet and Lincoln zones consist of as many as 38 gold-bearing quartz-ankerite veins, although most of the mineralization is hosted within five major veins; these are numbered the "6", "40", "42", "43", and "50" veins. The "40", "42", and "50" veins are through-going veins that occur within the full length of the resource area, while the "6" and "43" veins are localized within the Lincoln and Comet zones, respectively. The gold-quartz-ankerite veins are generally 1 to 4ft in width, with locally substantial widths of up to 20ft of strong gold mineralization. Weak to moderate gold mineralization does extend between the two zones along the more through-going veins.

The Medean and South Spring Hill zones represent mineralization along the East and West vein systems, respectively. The Medean resource is hosted with the "K13" vein while the South Spring Hill resource is hosted within the "K5" vein.

The veins that comprise the Lincoln and Comet zones are situated in the panel of Brower Creek metavolcanic rocks between the equivalent of the West vein and the East vein, both considered to be extensions of the main producing veins of the district. The Lincoln and Comet veins are primarily in the hanging wall east of the West vein and may be extensional veins.

The veins of the Comet and Lincoln zones may contain mineralized shoots whose morphologies are different than the shoots of the main district veins. Campodonic (2000) suggested, based on his geostatistical work, that mineralized shoots in the Comet and Lincoln zones may have gentle or even sub-horizontal plunges, different than the steeply-plunging shoots in the main veins.

7.5.1 Lincoln Zone

The Lincoln zone is located directly north of, and in the hanging wall of, the old Lincoln mine workings. The area was drilled by Callahan and Pancana during the mid-1980s and later by Meridian. New cross sections prepared by Mark Payne and Bill Mitchell in 2009, and which form the basis for MDA's current resource estimate, indicate that the high-grade portion of the Lincoln zone occurs within a series of four distinct, closely spaced veins (the 2, 3, 6, and 42 veins) at their upper terminations against a shallow, east-dipping structure (vein 9). Directly below the east-dipping structure, the four veins coalesce and form a mineralized zone that is up to 40ft wide and averages greater than 0.1oz Au/ton. Downdip away from the east-dipping vein, mineralization occurs primarily within the "6" and "42" veins.

7.5.2 Comet Zone

The Comet zone is located approximately 1,000ft north of the Lincoln zone. Meridian and SGM conducted all the drilling in the Comet zone and explored it via the Stringbean Alley decline. Mineralization appears to be controlled by steeply west-dipping structures which have connecting west-dipping splits and which often terminate at their top against shallow, east-dipping structures. Resources are identified on multiple veins, though the bulk of the known mineralization in the Comet zone is in the "40", "42", "43", and "50" veins. The 40, 42 and 43 veins have been drifted on by sublevels, above the decline, at approximately the 1,030ft and 1,050ft elevations. Cross-cuts off the main decline cut the "50", "51," and "23" veins.

7.5.3 Medean Zone

The Medean zone is located along the north and east side of the property about 2,000ft north of the Comet zone. Prior to 1983, the term "Medean zone" was used for the eastern branch of the Gold fault zone. Since the late 1980s, the Medean zone was first called the "80" vein, but has come to refer to a broad, east-dipping zone bounded on the footwall side by the strong "K13" vein and on the hanging wall side by the much weaker "K16" vein, with branching veins within the zone linking the "K13" and "K16" veins (Payne, 2010, written communication). The Medean zone is along the contact between the upper metavolcaniclastic and epiclastic unit, and the underlying Brower Creek Member. The zone was explored and/or exploited from the Medean shaft and from the 600, 700, and 900 levels of the Talisman shaft.

7.5.4 South Spring Hill Zone

The South Spring Hill zone is located along the north and west side of the property about 2,000ft north of the Comet zone. SGM explored this zone in 2006-2007 with the primary vein structure designated as the "K5" vein with weaker hanging wall "K23" and footwall "K25" and "K26" veins. The zone is along the slate-greenstone contact. The zone was explored and/or exploited from the South Spring Hill shaft down to the 1,200 level though almost all production was from above the 900 level.

Considered a southern extension of the South Spring Hill zone or West vein system, the Stringbean Alley zone is located about 500ft north of the Comet zone, approximately under the Stringbean Alley decline shop area. Mineralized intercepts in 3 drill holes are below about 600ft from the surface, between about 200 and 600ft above sea level. These intercepts were assigned to the "70" vein by workers of the 1980s and 1990s.

8.0 DEPOSIT TYPES

Clark (1970) provided the following overview of the mineralization in the Jackson-Plymouth belt of the Mother Lode:

> "*The ore bodies occur in massive and sheared quartz veins often with abundant fault gouge. The veins are mainly in slate of the Mariposa Formation. The veins sometimes are tens of feet thick; in places the Keystone vein is as much as 200 feet thick. Usually there are many stringers. The ore bodies contain disseminated fine free gold, pyrite, and minor amounts of other sulfides. The sulfides usually average one to two percent of the ore. In addition, greenstone bodies with disseminated auriferous pyrite known as "gray ore" sometimes are adjacent to the quartz veins at depth. The milling ore usually is low to moderate in grade (1/7 to 1/3 ounce of gold per ton), but a number of the veins have been mined to inclined depths of 4000 to 6000 feet. The ore shoots usually had stope lengths of 200 to 500 feet, but pitch lengths were much greater, and often nearly vertical. A number of high-grade pockets were found.*"

The gold-quartz mineralization at the Lincoln Mine project is of orogenic (mesothermal) type. Such deposits are hosted by metamorphosed submarine volcanic-sedimentary rocks and the intrusive rocks that cut them. The deposits are spatially associated with deep crustal faults. Structurally controlled gold mineralization occurs as vein quartz filling dilatant zones and as sulfidized replacements in altered wall rocks. Orogenic gold deposits are locally high grade and commonly contain coarse particulate gold, which can make grade estimation within mineralized shoots difficult. Closely spaced drilling and underground development are generally required to demonstrate the existence of resources with economic potential.

9.0 EXPLORATION

The following information is largely taken from Payne (2008), with additional information from SGM and other sources as cited.

SGM was incorporated in 1994 to operate the Lincoln Mine project. Much of SGM's initial work on the property involved permitting, which continues along with site design and property consolidation.

SGM conducted a limited underground sampling program in 2003 and 2005, which totaled 58 rock chip and muck samples. An additional 51 underground samples were collected in 2006 and 2007.

In July 2006, SGM initiated a two-phase drilling program to test "Inferred Resource areas" in the Comet and "historic Inferred Resources" in the Keystone area (Payne, 2008). This 41-hole drill program was completed in 2007. Payne (2008) described the results of this drilling as follows (MDA notes that the current drill-hole database records 9,127ft of underground drilling and 8,068.5ft of Keystone surface drilling, rather than the 9,067ft and 8,082ft described below):

> *"The 2006 underground drilling in the gap area between the Lincoln and Comet Resources totaled 9,067 feet* [33 holes] *and identified modest additions to the Indicated Resource inventory and new Inferred Resources in several new structures. The 2006 surface drilling of the historic Keystone Inferred Resource totaled 8,082 feet* [8 holes] *and was designed to upgrade mineralization into the Indicated Resource category. Significant drill hole deviations created a wider hole spacing than was anticipated, and resulted in the estimation of mixed drill-Indicated and Inferred Resources. It was significant to note that seven of the eight intercepts in the Keystone 5 Vein encountered 3 to 31 feet of mineralization grading between 0.08 and 0.28 oz/ton gold, with an arithmetic average of 11.2 feet @ 0.19 oz/ton."*

In 2007, work continued on the structural geology and exploration model for the project by SGM's consultant, Mark Payne (Payne, 2008). In July 2007, a six-hole surface core drilling program was proposed to step southward from the historical Inferred Resource in the Keystone area and to identify new resources to the south, toward the Lincoln-Comet area (Payne, 2008). Only two holes totaling 2,350ft were completed (KDH-0029 and KDA-0030, although portions of KDA-0030 have not been logged or assayed), but both intersected mineralized intervals.

Historical records such as assay ledgers and monthly development reports from the Keystone mine were also reviewed to reconstruct a plan map of sample assays for the Keystone 5 vein (Payne, 2008).

SGM collected a bulk sample of the Lincoln-Comet mineralized zone in 2008-2009 that was submitted for gravity/flotation testing by McClelland Laboratories Inc. ("McClelland"). Rougher tailings generated from the sample at McClelland were sent to Golder Paste Technology and utilized for paste backfill testing (Golder Paste Technology Ltd., 2009).

SGM conducted an underground sampling program in 2009 that included face sampling, long-hole percussion drilling into the face, channel sampling on the backs and the faces, muck sampling from the LHD buckets, and bulk sampling of each round. The current database includes 71 channel samples; none of the other sample types are included and were not used in MDA's resource estimate.

SGM completed a detailed survey of the underground workings in 2009 (Sutter Gold Mining Company, 2010b) in response to questions and discrepancies raised by Ronning and Prenn (2004) and SGM concerning the historic survey coordinates. Of specific concern to Ronning and Prenn (2004) was the location of the underground workings in relation to the property boundaries. According to the SGM report, the 2009 survey corrected all previously identified errors and issues associated with the spatial location of the workings and the survey was used to update the underground base maps.

The new survey data is referenced to the same NAD 27 California State Plane 27 Zone 2 coordinates as are the surface land maps. The updated property map showing the workings, vein geology and property boundaries was used by MDA to constrain the resource model.

SGM conducted additional underground sampling in 2010. These data are not included in the data used for the current resource estimate.

SGM drilled 26 core holes for a total of 10,240ft in the Lincoln-Comet resource area in 2012. The purpose of the drilling was to test internal gaps in the Lincoln-Comet resource and to explore nearby regions for extensions of gold mineralization. SGM's evaluation of the results indicates a potential 10-15% increase in the Lincoln-Comet resource.

SGM completed about 3,300ft of underground development during the period of late 2012 through early 2014. Channel, chip, and muck sampling were conducted along with a limited underground drilling program.

MDA reviewed all 26 of the surface holes completed in 2012, along with the majority of underground drilling and sampling, and concludes that this drilling and sampling substantially supports the 2011 estimate. Though the drilling and underground development did locally extend and expand the high-grade gold zones, this work did not change the resource in a material way. For this reason, the Lincoln-Comet resource estimate described in Section 14.2 is still current.

Except as noted above, all investigations have been carried out by SGM. Details on drilling, including drill contractors, are provided in Section 10.0.

10.0 DRILLING

This section of the report only describes drilling completed since Callahan's acquisition of the project in 1983. MDA has no knowledge of any drilling completed prior to 1983. For the purposes of this report, MDA uses the term historical to describe work completed on the Lincoln Mine project prior to the formation of SGM in 1994.

Exploration and definition drilling from 1983-2013 of 9 RC and 272 core holes (from surface and underground) has totaled approximately 112,754ft (see Table 10.1). This total includes the 2 surface RC holes, 87 surface core holes, and 107 underground core holes used in the 2011 Lincoln-Comet resource estimate and the seven RC and 23 surface core holes used in the Keystone resource estimate.

Table 10.1 Drilling on the Lincoln Mine Project

Company	Date	Area	Type	# of Holes	Depth (ft)	Total # of Holes	Total Depth (ft)
Callahan	1983-1984	Keystone	RC	7	4,416	24	13,147
	1983	Keystone	core	2	1,467		
	1983	Lincoln-Comet	RC	2	466		
	1984-1985	Lincoln-Comet	core	13	6,798		
Callahan-Pancana	1986	Lincoln-Comet	core	15	9,742	15	9,742
Meridian	1987-1990	Lincoln-Comet	core	59	30,334	144	57,788
	1990	Lincoln-Comet	core U/G	74	18,273		
	1988-1989	Keystone	core	11	9,182		
Sutter Gold	2006	Lincoln-Comet	core U/G	33	9,127	43	19,546
	2006-2007	Keystone	core	10	10,419		
Project Drilling (used in resource estimates)	**1983-2007**	**all**	**RC**	**9**	**4,882**	**226**	**100,222**
			core	**110**	67,940		
			core U/G	**107**	27,400		
Sutter Gold	2012	Lincoln-Comet	core	26	10,244	55	12,532
			core U/G	29	2,288		
Total Project Drilling	**1983-2012**	**all**	**RC**	**9**	**4,882**	**281**	**112,754**
			core	**136**	**78,184**		
			core U/G	**136**	**29,688**		

The project total in Table 10.1 includes the 2012 SGM Lincoln-Comet core holes (26 surface and 29 underground) drilled after the completion of MDA's 2011 resource estimate. The total 2012 drill footage represents approximately a 12 percent increase in drill footage from the 2011 drilling used in the resource estimate. MDA reviewed the 2012 drill data and believes the drilling substantially supports the 2011 estimate. Though the drilling and underground development did locally extend and expand the high-grade gold zones, this work did not change the resource in a material way.

Payne (2008) reported that all drill core from exploration and infill drilling of the Lincoln-Comet resource prior to 2006 is no longer available due to the failure of project operators to properly store and maintain the core. Most of the core from the 13 historic holes drilled at Keystone prior to work by SGM is available in the core-storage facility (Payne, 2008). Payne (2008) did not report on the status of the cuttings from the 9 RC holes. The mineralized intervals of core from the 2012 drill program are available in the core storage facility.

10.1 Drilling Contractors

MDA has no information on the drill contractors or types of rigs used by Callahan and the Callahan-Pancana joint venture in their drilling on the property. Payne (2008) reported that all holes were drilled from the surface and that the core was NQ in diameter.

Meridian used SDS Drilling ("SDS") as their contractor. SDS used a skid-mounted rig for the underground drilling and a truck-mounted Longyear rig for the surface holes. All of Meridian's surface holes were drilled with HQ core; their underground holes were drilled with BW44 or NQ core (Payne, 2008).

Kirkness Drilling ("Kirkness") was the drill contractor for SGM's 2006 underground drilling at Lincoln-Comet. They used a skid-mounted Hagby 75HP rig for this work, drilling NQ2 core. For SGM's surface drilling at Keystone, from July 2006 through April 2007, two drill contractors were used. Both drilled HQ core. Kirkness drilled with a track-mounted Longyear LM75. Sierra Madre Exploration Services ("Sierra Madre") drilled with a track-mounted Casagrande C5 rig. SGM used Sierra Madre for their surface drilling at Keystone in September-November 2007, and Sierra Madre again used the track-mounted Casagrande C5, drilling HQ core.

Ruen Drilling ("Ruen") was the drill contractor for SGM's 2012 surface drilling at Lincoln-Comet. Ruen drilled using a track-mounted Longyear LF50 rig and the holes were drilled with HQ core. The 2012 underground drilling comprised relatively short holes (<100ft average depth) drilled by SGM using an electric "termite" drill.

10.2 Collar and Down-hole Surveys

Callahan, the Callahan-Pancana joint venture, and Meridian all conducted down-hole surveying of their core holes using an Eastman camera and surveying every 100ft. Payne (2008) reported that all core hole collars from SGM's 2006-2007 drilling were surveyed, and down-hole surveys were conducted with a Reflex EZ Shot tool at 100ft intervals in all drill holes. A ball-mark-type ACE tool was used to provide core orientation information in some of SGM's 2006-2007 diamond drill holes.

The 2012 surface drill collars were professionally surveyed by Terra Firma Surveyors. Down-hole surveys were taken in each drill hole at 100ft intervals, in some cases every 50ft, from drill collar to bottom of hole using an EZ-shot survey instrument.

10.3 Core Recovery

Core recovery data were recorded on the paper drill logs for 169 of the 194 core holes drilled within the current Lincoln-Comet resource area (only summary logs with no core recovery data are available for the initial 25 core holes) and for all of the Keystone core holes. MDA has been provided the core recovery data in digital form for the Lincoln-Comet drilling, though for many holes the data are "summarized" by combining drill-run intervals of similar recoveries; i.e., if ten 5ft drill runs all have

100 percent core recovery, the digital data indicates a 50ft interval with 100 percent recovery. In a few holes, the digital core recovery intervals are greater than 100ft in thickness. MDA reviewed the drill logs and core photos for a number of these large core recovery intervals and found that the digital data accurately reflect the drill log data. However, the core photos indicate that there can be isolated, thin (less than 5ft) intervals which have lower recoveries than the combined interval average, indicating some imprecision in the core recovery data. MDA has not conducted a thorough review of all of the core recovery data to determine how widespread this lack of precision is, but the initial review indicates that this is not a significant issue and will not materially affect the resource estimate.

The core recovery data are dominated by measurements of >90 percent recovery with isolated zones of lower recovery. The average core recovery for all readings is approximately 97 percent. Approximately 60 percent of all core recovery measurements have values of 100 percent recovery. The prevalence of exact 100 percent core recovery values is indicative of the massive, weakly fractured nature of the country rock, but also suggests possibly less rigorous measurement techniques. The average core recovery for the mineralized intervals used in the Lincoln-Comet resource estimate is approximately 90 percent. An analysis of core recovery versus gold grade was conducted by MDA and is reported in Section 12.3.4.

MDA evaluated the core recovery data for the Keystone holes by checking the drill log data against the core photos. As with the Lincoln-Comet recovery data, the Keystone core recovery data is primarily >90 percent recovery, with isolated zones of lower recovery. Average recovery has not been calculated, though it would likely be >95 percent.

10.4 Underground Sample Resource Database

The database used in the Lincoln-Comet resource estimate contains collar information on 778 individual underground channel samples. Within the database, the underground channel samples are considered as short, horizontal “drill holes.” Where two or more channel samples form a continuous sequence of samples taken across a face, these samples are linked together with each sample representing a “downhole” interval. As a result, the database contains 435 channel-sample collars.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

11.1 Modern Operators Prior to Sutter Gold Mining Inc.

All of the original drill hole logs are available and are essentially complete. There are no remaining drill cores, chip samples, coarse rejects, or pulp samples from the historical drilling and chip sampling programs within the Lincoln-Comet resource area.

11.1.1 Pre-1994 Logging and Sample Selection

Ronning and Prenn (2004) provided the following observations based on review of drill logs and assay certificates. This information has been checked and confirmed by the authors.

The drill core was selectively sampled, using visible mineralization or alteration as a guide. Quartz veins, in particular, were selected for sampling, as were intervals of altered wall rock. In long intervals of similarly altered wall rock, 4 or 5ft sample intervals were typical, although longer intervals were occasionally used. In sections of core deemed most likely to be mineralized, shorter intervals were selected based on visible geological differences in the core. The relationship between sample width and true thickness of vein mineralization is variable as its dependent on the drill angle and orientation of the intercepted vein. In some instances, the sample and vein widths are very similar. In much, but not all of the core logging and sampling, the smallest unit of measurement used was 0.5ft. This would result in, for example, a 3.3ft vein intercept being included in a 3.5ft sample.

In most instances, apparent care was taken to collect samples in wall rocks adjacent to visible or expected mineralization, so that high-grade samples would be “bracketed” by low-grade samples. However, some instances exist where this was not done. For example, in MDDH-154, a sample collected from 60.5 to 62.5 feet contained 1.538 oz Au/ton. The interval from 54 feet to 60.5 feet was not sampled. It would have been prudent to do so.

The few rotary RC drill holes appear to have been continuously sampled. Five-foot sample intervals were routinely used, reducing to 2.5ft intervals in visibly altered or mineralized rock. The equipment and methods used for sampling are not recorded.

Meridian conducted underground sampling from 1989 through 1991. Copies of the original sample card field notes for 781 individual samples indicate that the samples consisted primarily of rock chip “channel” samples, though some muck grab samples and random face samples were also collected. In addition to these recorded samples, assay certificates indicate that a significant number of additional muck samples were also collected and analyzed. There is no further information on these latter muck samples, and MDA has no documentation on the methods of sampling employed in collecting the samples.

The current database includes 707 Meridian channel samples. It is not clear whether the channel samples are rock-chip grab samples collected along the sample length, or continuous channel samples, in which care was taken to collect the same volume of rock along each portion of the sample length. Channel sample lengths ranged from 0.3ft to 12ft. Sample lengths appear to have been selected based on geological criteria, principally vein boundaries. Where there is a continuous sequence of samples, there will be a sample within the vein and then one or more samples within the hanging wall and/or footwall. Evidence of the historic sampling can be observed within some areas of the current underground workings. Sample identification numbers and channel orientation lines are still in evidence in areas of no subsequent development. None of the face samples or muck grab samples are included in the database due to uncertainties as to their sample location and methods of sampling.

The current information indicates that the core drilling and underground sampling programs conducted by Callahan and Meridian were done in a professional manner and in accordance with accepted industry standards of the time. MDA believes that the risk to the resource estimate from the core drilling and chip/channel sampling procedures is low.

11.1.2 Pre-1994 Sample Preparation, Analyses and QA/QC Procedures

MDA has no information on sample preparation used by Callahan, Callahan-Pancana, or Meridian in their work on the Lincoln Mine project. The following information on analyses is taken from copies of laboratory certificates that are on file with the original drill logs for each hole. Additional details are provided by Ronning and Prenn (2004), who based their information on that supplied by Ray Irwin (2003).

For Callahan's and Callahan-Pancana's first 30 drill holes, four different laboratories were used: Rocky Mountain Laboratories ("Rocky Mountain"), Shasta Analytical Geochemistry Laboratory ("Shasta"), Barringer Laboratories Inc. ("Barringer"), and ALS Chemex ("Chemex"), with Shasta and Barringer doing most of the work. There is no information on details of analytical procedures. For the work completed by Chemex, gold samples analyzed by atomic absorption ("AA") that contained over 20g Au/ton would have been re-analyzed by fire assay with a gravimetric finish using a 30g charge. Barringer conducted screen metallics fire assays on sample composites from holes LDDH 14 and LDDH 15 – the only screen metallics fire assays on Callahan's drilling.

Callahan and Callahan-Pancana did extensive check assaying of mineralized intervals. This check assaying usually consisted of the principal lab assaying a second and sometimes a third split of the same pulp. Occasionally, a new pulp would have been prepared from the remaining coarse reject. According to Irwin (2003), all of these analyses were apparently performed by fire assay with an atomic absorption finish ("FA-AA") or gravimetric finish on a 30g charge. The checks and screen fire assay results suggested that coarse gold was present in some samples (Irwin, 2003).

For Meridian's Lincoln-Comet holes MDDH 31 through 72, and Keystone holes KDH 10 through 15, Barringer was the principal lab. No check assaying was conducted on these holes. For holes MDDH 74 through MDDH 113, and KDH15 through 20 Chemex was the principal lab. Chemex used the following analytical procedures (Irwin, 2003):

- Crush the entire sample to minus 10 mesh.
- Take a 250g split with a rifle splitter.
- Pulverize the split with an early version of a ring and puck pulverizer.
- A 30g sub-sample was analyzed with FA-AA.
- Samples found to contain greater than 20g Au/ton were automatically re-analyzed on a duplicate pulp using fire assay preparation with a gravimetric finish on a 30g sub-sample.

In this sequence of holes, only one sample from MDDH 84 was checked; a duplicate of the original pulp was analyzed by fire assay with a gravimetric finish on a 30g charge.

For holes MDDH 114 through MDDH 162, Meridian continued to use Chemex as the principal lab using the same procedures described above. However, some check analyses were performed on selected samples from 14 of these holes. Most checks were done on duplicate sub-samples obtained from the original 250g of pulverized material; these showed some variation due to coarse gold. Some check analyses were done using a new pulp prepared from the coarse reject material; these showed a greater degree of variation between the original analysis and the check. Most analyses were FA-AA, but in some of the later holes a gravimetric finish was used. According to Ronning and Prenn (2004), "*Irwin concluded that the paucity of check sampling and the lack of screen fire assaying rendered the data produced by those drill campaigns inadequate for accurate estimation of the grades.*"

It is expected that the sample collection, preparation, and assay procedures conducted by Meridian would have been performed in accordance with accepted industry standards at that time. MDA's sample heterogeneity study (see Section 12.3.3) indicates that there are significant grade estimation concerns within all drill and sampling campaigns due to the coarse-gold nature of the mineralization. It is MDA's opinion that the Meridian drill results can be used in conjunction with the other historical drilling to provide an estimate of size and grade of the deposit and provide appropriate information for mine planning. The imprecision of the drill assay results is reflected in a resource classification of Inferred for those portions of the deposit located away from the underground workings and which rely solely on drill data for grade estimation.

11.1.3 Pre-1994 Underground Chip Sampling, Preparation and Analytical Procedures

Lacking original records, MDA has not determined any details of how rock-chip samples were processed prior to their delivery to the laboratory.

Campodonic (2000) deduced, based on his conversations with mine employees, that the samples were reduced in particle size and weight prior to being sent to the laboratory. The samples were apparently fed into a jaw crusher and then a roll crusher, followed by screening to the desired (but unstated) particle size. The screened material was run through a vibrating hopper feeding into a constant-speed rotating sample splitter. The sample splitter divided the sample into 15 bins. Every third bin was then combined, producing three samples.

Ronning and Prenn (2004) summarized what is known and their observations about the underground chip sampling are as follows.

Laboratory certificates are available for 511 chip samples processed by Chemex. The manner in which Chemex processed the samples can be inferred from codes on the laboratory certificates. There was some variation in detail from time to time, but in general the procedures were:

- 4 to 7kg of sample material was crushed until 70% of the material passed through a 10 mesh (2mm) screen, and a 250g split was obtained;
- Either a ring grinder or rotary pulverizer was used to reduce the 250g split to -150 mesh; and
- Gold was analyzed using a 30g sub-sample, fire assay preparation, and gravimetric measurement.

Laboratory certificates are available for 299 chip samples processed by Shasta. In a letter dated February 11, 1991, Shasta described the services that it would provide to the Sutter Gold Venture. They were to include:

- dry the samples at 60° C.
- mechanical and air cleaning of crushers and pulverizers between samples;
- entire sample crushed to -10 mesh by roll crushing each sample after jaw crushing. Pulps were to be rolled 40 repetitions before sub-sampling;
- 250g splits were to be taken from the crushed material and plate or ring-and-puck pulverized to -200 mesh;
- for gold fire assays, 30g sub-samples were fused;
- gold was to be measured gravimetrically, and a detection limit of 0.002 oz Au/ton attained;
- furnaces were to be mechanically cleaned every 24 hours; and
- for geochemical analyses of other elements, 3g sub-samples were used for atomic absorption analyses. Detection limits were to be:

silver	0.1 ppm
arsenic	3.0 ppm
antimony	3.0 ppm
copper	0.1 ppm
lead	0.2 ppm
zinc	0.5 ppm
iron	1.0 ppm (iron in sulfide)

On reviewing the laboratory certificates from Shasta, it appears that in almost all cases three separate sub-samples from each 250g pulp were analyzed for gold, and each of the three values was reported to SGM. SGM averaged the three gold values to obtain the value assigned to the sample.

11.2 Sutter Gold Mining Inc. (1994 through 2007)

All of the following information relates to the sampling performed by SGM in their exploration of the property and is taken from Payne's (2008) technical report. Data compilation for the SGM's surface and underground core drilling programs was performed by SGM's geological staff, with review and verification by Mark Payne. It is the author's opinion that SGM's sample collection, preparation, and assay procedures have been performed in accordance with accepted industry standards and they are adequate to support a classified resource estimate to be used for further mine planning.

11.2.1 Sutter Gold Mining Inc. Logging and Sample Selection

The 2006-2007 SGM exploration program was planned and supervised by Mark Payne, a consulting geologist and Qualified Person. All of the following information in Section 11.2 relates to the sampling performed by SGM in their exploration of the property and is taken from Payne's (2008) technical report. SGM has a core-handling and core-sampling protocol designed by Payne for use in sampling drill core from coarse-gold-bearing vein systems. SGM has implemented a quality assurance/quality control program ("QA/QC") to ensure sampling and analyses of all drill core are conducted in accordance with the best industry practices. The following procedures were implemented to insure proper handling and processing of core for the 2006-2007 Keystone exploration drilling and were identical to those adopted for the 2006 Lincoln-Comet underground drilling (Payne, 2010, written communication).

11.2.2 Core Logging and Photo Documentation

All core logging was conducted at the Lincoln Mine project site. When core was obtained by the duty geologist, it was inspected, and a rapid log was made which identified the obvious altered and mineralized intervals. The duty geologist inspected the highly mineralized sections of the core for visible free gold, which if found, was documented with digital photos. Then, the geological consultant oriented the north-orientation marking facing up and rolled and matched the drill cores through the whole box. The geologist measured core recoveries in feet and tenths of feet, then logged the core on a paper core-log sheet. Each box of unsampled core was photographed. The core was then placed in SGM's core-storage facility.

11.2.3 Determination of Sample Size

The observed presence of coarse-gold and the choice to use screened metallics fire assay analysis determined the minimum sample sizes to be selected from the drill cores. Sample intervals for mineralized core were in the range of approximately 3.0 to 3.3ft where practical, with the minimum being 2.9ft. This was designed to ensure an adequate sample size was available for multiple 500g or 1,000g screened metallics fire assays

11.2.4 Core Handling and Sampling Methodology

The logged core was marked with flagging at the start and end of the interval to be sampled, and an aluminum tag was attached to the box at the start of the sample interval, labeled with sample number and footage interval. The typical vein structure noted from drill holes and underground exposures was 3 to 5ft in true width. For veins or sulfide replacement intervals greater than 3ft in width, the sample length may be up to 4.5ft long in order to maintain sample size and not introduce undue wall rock dilution. For quartz intervals longer than 4.5ft, the interval was divided into two equal length samples.

The core was brought to the surface, inspected by a geologist for free gold, logged, photographed, oriented for drill-hole direction, taped to reduce fragmentation, and cut into halves with a wet-type tile saw with a diamond blade. The saw was set up to operate with a steady inflow of fresh water and continuous drainage out of the water tray to avoid contamination of core. The blade was also cleaned after cutting core with visible gold to prevent contamination.

The completed core log pages were copied daily by the duty geologist. The original core logs were stored at the core-storage facility, with a copy filed at the project office and a copy distributed to each staff geologist for internal quality-checking purposes. The duty geologist then sampled the mineralized portions of the core. Sample description books with numbered pages and identically numbered tear-off tags were used. As samples were taken, pertinent identifying information and the type of processing needed were noted for each sample. The numbered sample tag traveled with the sample throughout the entire sample prep, assaying, and replicate assaying process.

Samples of core containing visible gold, significant arsenopyrite, fault-bounded ribboned or banded vein quartz, or strong sulfide replacement mineralization, were analyzed by screened metallics fire assay. All other pyritic mineralized rock, phyllonite, and altered rock were analyzed by one-assay ton fire assay. Sample-description cards and core logs were collected and copied. Sample intervals with the sample number were recorded on the drill-core log by the duty geologist. Samples were collected at the end of the day and stored in a locked unit until samples were shipped to the assay lab. The duty geologist ensured all samples were present and accounted for, and loaded them into large shipping bags of 55-60 pounds, which were then wired shut. The shipping bags were marked with the company name and all sample numbers contained within. Samples were picked up at the site by the assay lab, or samples were transported directly to the lab by a trained SGM employee.

11.2.5 Sutter Gold Mining Inc. Sample Preparation, Analytical and QA/QC Procedures

11.2.5.1 Fire Assay Analysis

Half of the sawn core was shipped to American Assay Laboratories ("American Assay") in Sparks, Nevada, for preparation and analysis. The other half of the core is stored at the SGM core storage facility. All replicate fire assaying was also conducted by American Assay utilizing an approved method of blind re-submission. As of September 14, 2010, American Assay was not currently registered, but was working on their ISO 17025 certification (personal communication, 2010).

All gold analyses of strongly mineralized samples utilized the screened metallics fire assay ("SMF") method with a gravimetric finish. At the laboratory, the entire sample was crushed to 90% minus 10-mesh. A rotary splitter was used to obtain a 500g sample for pulverizing. A separate 60-100g rotary split of the coarse reject was made, and that sample was pulverized with a closed bowl-type grinder and shipped directly back to SGM. The screened metallics were collected as the plus fraction from a 150-mesh screen at the lab and were fire assayed. Two separate one-assay ton fire analyses of the minus 150-mesh fraction were performed and arithmetically averaged. The minus and plus 150-mesh results were then combined for a total SMF assay.

All gold analyses of altered and weakly mineralized samples were made by 30g fire assay ("1ATF") with a gravimetric finish.

American Assay internally re-assays at least 10% of all samples. Each batch of 50 samples to be fire assayed includes at least 5% lab standards and blanks.

Analytical problems encountered during the 2006-2007 drilling program were minor. They consisted of a few poor bead fusions for some very high-grade gold samples, and minor contaminations of gold were incorporated into abrasive blank material prepared directly after a few high-grade samples. The poor bead fusions were readily rectified by automatic tripling of the silver inquart for all samples submitted from the Lincoln Mine project. The minor gold contamination of 13 blank samples, which directly followed high-grade samples, was either the result of gold smearing in the LM-1 pulverizer, or particulate gold not removed from the 150-mesh screen being introduced into the following blank.

11.2.5.2 Chain of Custody

At the drill site, the core was placed into wooden core boxes by the drilling contractors. Only the designated duty geologist was authorized by the drilling contractors to examine or take custody of the boxed core. After the drilling contractors released custody of the drill core, access to the core was restricted to the project geological consultants. The boxed core was transported directly to a secure core-storage facility located at the mine site. Access to the core-storage facility was restricted to geological consultants and designated trained sample technicians.

SGM maintained the shortest possible chain of custody for samples from the drill site to the assay lab. Samples of the mineralized core were placed in a locked room by the sampler at the end of every work shift. Generally sample shipments were transported by designated personnel directly to the lab, and the coarse rejects and pulps were returned directly to the project site. Larger sample shipments were picked up at the project site by analytical lab personnel.

Coarse rejects and pulps were stored in a locked room at the site for future use as replicate assays. Coarse rejects with the appropriate color and abrasive characteristics were stored for use as blank material in replicate assay shipments.

11.2.5.3 Sutter Gold Mining Inc. QA/QC

SGM's QA/QC program included a replicate assaying program along with the routine insertion of hard abrasive blank sample material and three different reference standards. As an additional check on the analyses, a 60-100g sample was rotary split from the coarse reject (90% passing 10-mesh) at the laboratory, pulverized to 90% passing 150-mesh, and shipped to SGM. The split prepared at the lab was panned and used for rough comparisons to lab analytical results.

Each sample shipment included at least 10% inserted standards and blanks. The blank material consisted of unaltered and unmineralized metavolcanic drill core containing no visible quartz. Blanks were always inserted by the project's Qualified Person after every drill sample containing over 25% vein quartz by volume (Payne, 2008). Assays for blank samples were considered acceptable by SGM if they were less than, or equal to, twice the lower detection limit for gold (0.006oz Au/ton). Nineteen blanks exceeded 0.006oz Au/ton, including eight blanks exceeding 0.009oz Au/ton.

SGM prepared the reference standards from a homogenized mixture of gold-mineralized material collected from historic waste dumps on the property. The following information on the SGM standards is taken from Behre Dolbear (2007):

"*to be certified as an acceptable standard ("SRM"), the assays of sub-samples from the theoretically homogeneous standard must be within a 20 percent range. Of the 94 sub-samples analyzed, SGMC reports that 62 performed within 20 percent of the mean but in general performed poorly. Analyses of the minus 150 mesh fraction show a fifty percent range from the mean. About 1 in 20 samples were SRMs.*

The standard sample results show the difficulty of preparing homogeneous samples from the Lincoln-Comet type mineralization and that the determination of accuracy is dependent on the laboratory's internal standards and procedures. Behre Dolbear concludes from the information that the absolute accuracy of the core samples analyzed has not been determined from the SRMs."

The duplicate assaying procedure incorporated a minimum of 10% of all samples to be submitted as coarse rejects for replicate screened metallics fire assay. To eliminate bias, four different criteria were used in combination to select the samples to be shipped for assay replication:

(1) All mineralized samples identified as exhibiting specific characteristics known to occur in the historic mineralized shoots of the Lincoln Mine property;

(2) All samples with gold assays greater than or equal to 0.01oz Au/ton;

(3) All samples which panned significant gold when assays indicated otherwise were included in the replicate sample shipment. This was enacted only when screened metallics fire assays indicated that the majority of gold resided in the coarse fraction, which increased the potential for assay variance. As part of the same sub-set, any samples containing high percentages of gold in the plus fraction of a screened fire assay were submitted for replication regardless of the assay value if there was enough material available after completion of the first screened fire assay. SGM reports that in 2010 that protocol was still used for samples containing appreciable coarse particulate gold greater than 100 microns (Payne, 2010, written communication).

(4) In the event that samples based on criteria 1, 2, and 3, plus standards and blanks, totaled less than 20% of the total number of samples from the original sample shipment, the remainder of the replicate shipment was made up of randomly selected samples until the shipment totaled 20% of the primary assay lab sample shipment, regardless of their assay values.

11.2.6 Using Second Half of Cores

When the quantity of coarse reject was insufficient for completing the primary or replicate screened metallics fire assay analyses, the second half of the core could be utilized. The second half of the core that was identified for sampling was digitally photographed with the sample locations marked in the core box. Appropriate aluminum tags with the additional new sample numbers were affixed (Payne, 2008).

Where significant visible gold was noted on the drill core, but primary and replicate fire assay analyses did not reflect that, the second half of the core could be analyzed. To preserve the sample-size integrity, there was no quartering of the core in these instances.

11.3 Sutter Gold Mining Inc. (2012)

The following summary information on SGM's 2012 drill program is taken from an SGM in-house report (Zahony, 2012). MDA has not verified this information.

All drill core was logged by SGM personnel in the SGM core storage facility. Mineralized intervals identified during logging were split in half using a diamond rock saw with half of the core retained in the core box and the second half bagged and sent to ALS Minerals in Reno, NV. Samples were assayed for gold by fire assay with an atomic absorption finish. In all cases, fifty-gram pulp samples were utilized for the fire assay analysis. The core remaining from intervals sent for analysis is kept at the core storage facility with the remaining unmineralized core being discarded.

12.0 DATA VERIFICATION

12.1 Data Verification by Previous Authors

There is no remaining drill core, chip samples, coarse rejects, or pulp samples for the Lincoln-Comet resource area from the Callahan, Callahan-Pancana, or Meridian programs (Behre Dolbear, 2007). However, Ronning and Prenn (2004) reported that all of the original drill-hole logs are available. The file for each drill hole contains the log, related laboratory certificates, a listing of down-hole surveys, and the original camera films from the down-hole survey instrument. Ronning and Prenn reviewed about 11% of the logs and noted a possible error in recording the down-hole surveys in MDDH 89 that suggested the hole deviated by almost exactly 180° between the collar and the 100ft mark. They also found a data entry error in the digital data file.

In 2005-2006, the Callahan and Meridian drilling and underground data were verified for the 2006 resource estimate (Payne and Grunwald, 2006). Grunwald conducted confirmation sampling underground of four separate vein structures. Results of these assays ranged from significantly lower to significantly higher grades than those that had been previously obtained, but there were no signs of consistently high or consistently low assays that could indicate sample preparation problems or bias in sample collection (Payne and Grunwald, 2006). Grunwald took 20 chip/channel samples from the Stringbean Alley decline. Samples were cut across the trend of the veins and were about 2in wide and 1in deep. Depending on the length of the chip/channel, each sample varied from 5 to 20lb in weight. Where power was available, sampling was done with a Bosch 11230 EVS rotohammer with a bull-point chisel. Where there was no power, sampling was done with a bull-point chisel and 3lb sledgehammer. American Assay of Sparks, NV, carried out the sample preparation and screened metallics fire assays using the fire-gravimetric method. Payne and Grunwald (2006) reported that the assays of blanks submitted with the samples indicated carryover of free gold during sample preparation from preceding samples. In addition, one of the two standards (SRM 7.0) assayed 10% lower than the standard. None of these samples taken by Grunwald was used for the resource estimates in the 2006 technical report (Payne and Grunwald, 2006).

Behre Dolbear (2007) reviewed procedures and results for both historic and 2006 SGM sample collection, sample preparation, core sample assays, and chip/channel sample assays and concluded that based on the limited information available, "*the historical* [pre-SGM] *core drilling, core sample collection, and chip/channel sampling were done in accordance with past and current accepted industry procedures*." However, they also stated that "*based on what is known and not known regarding the preparation of the historical core and chip/channel samples, it is Behre Dolbear's opinion that the sample preparation procedures used by Callahan and Meridian were not appropriate or adequate to minimize the nugget effect and to produce 250 gram fire assay samples that are representative of the original samples. …In addition, an insufficient number of samples were assayed by screen fire assays to minimize the nugget effect*" (Behre Dolbear, 2007). Regarding SGM's work, they concluded that "*the 2006 on-site core drilling, logging, and sample collection methods meet or exceed accepted industry procedures and standards*" (Behre Dolbear, 2007). However, they also noted that "*the lack of homogeneous standard reference materials (SRMS) prevents the direct verification of assay accuracy*" (Behre Dolbear, 2007). Behre Dolbear (2007) concluded:

"It is Behre Dolbear's opinion that the historical assays can be used in conjunction with the assays from the 2006 Lincoln-Comet drilling to estimate an average grade of the deposit and provide appropriate

information for mine planning. Given the large number of historical core and chip/channel assays, Behre Dolbear concludes that the average grade of the deposit, if proper estimating procedures are used, will be reasonably accurate. The historical sample preparation and assay problems do not constitute a fatal flaw to the project. The risk to the resource estimate due to the unknown accuracy of the individual historical sample assays is moderate."

The SGM data were further verified in 2006-2007 for the 2008 estimate (Payne, 2008). Nineteen percent of the 2006-2007 drill hole samples for both the Keystone and Lincoln-Comet areas were selected for replicate check assays from the coarse reject material. According to Payne (2008), the composited total of all assays for 129 replicate Keystone samples was 3% lower overall compared to the composited primary assays. The composited total assays for 291 replicated Lincoln-Comet samples were 4% higher overall compared to the primary assays. One percent of the Keystone sample population and 2% of the Lincoln-Comet samples yielded screened fire assays containing 65% of the gold or greater in the coarse fraction. "*The majority of mineralized samples contain a significant quantity of fine-grained gold, which tempers the assay variance for individual samples. The 'nugget effect' is considered to be of moderate severity at the Sutter Gold Project*" (Payne, 2008).

12.2 Verification of Sutter Gold Mining Inc. Data by MDA

The following sections describe MDA's audit procedures and results completed in 2010 on the drill and underground sample data used in the current Lincoln-Comet and Keystone resource estimates. Except for the May and June 2015 site visits and project review with SGM personnel, MDA has not verified any of the post-resource estimate drill or underground sample data.

As discussed in Section 10.0, MDA uses the term historical to describe work completed on the Lincoln Mine project prior to the formation of SGM in 1994.

12.2.1 Database Audit and Reconstruction

For MDA's completion of the 2011 resource estimate, SGM provided MDA a digital database that included drill-hole collar coordinates and down hole survey, assay and geology data. The database includes the underground chip "channel" sample data which were constructed and listed in the database as short, horizontal drill holes. A significant part of MDA's 2011 audit was data research and then database reconstruction of the underground sample database.

12.2.1.1 Drill Hole Database Audit

Original data used for drill-hole database audit included digital files for each historic drill hole which contained the drill log, related laboratory certificates, a listing of down-hole surveys, and/or the original camera films from the down-hole survey instrument. Previous versions of the database were also checked when the original data were not available. The assay data for the SGM drilling were downloaded directly from American Assay and checked directly against the current database.

The underground drill-hole collar coordinates were revised by SGM in 2009 after a detailed survey of the Lincoln-Comet deposit underground workings. The survey resulted in a shift in most of the underground drill-hole collars of less than 5ft in the east and north coordinates, but up to 15ft in elevation. Upon the completion of the survey, MDA checked approximately 15% of the drill-hole collar coordinate data against the drill log hand-written collar data and then against the new 2009 collar survey data. No errors were found in the collar data.

Approximately 30% of the down-hole surveys were checked against either the original camera photos, or the hand-written survey notes located on the drill logs. Thirty-one survey readings were corrected (12% of the total number audited) with most of the errors due to an incorrect reading of the azimuth or minor changes in the interpretation of the camera readings.

The assay database includes the original assays plus all available check assay data. The “final” sample interval assay value to be used in the resource estimate is an average of all of the analyses. All historic assaying used a 1ATF technique. MDA’s verification of the historic assay data included checking approximately 30% of the sample intervals against the drill logs to confirm sample “from and “to” footage data, and then checking the assay values against the original assay certificates. In addition, all of the assay certificates were reviewed to confirm the inclusion of the check assay data. A total of 16 sample interval or assay value errors were noted and corrected, for an acceptable error rate of about 1%. The audit also resulted in the addition of assay data for 285 sample intervals. Much of the added data were check assay values, though original assay data were also added to the bottom of three drill holes.

The audit of SGM’s drill hole assay data consisted of a complete digital check of the data against assay certificates downloaded directly by MDA from American Assay. MDA also checked all of the sample-type designations against sample footages and assay values to confirm the presence of the QA/QC samples and the correct drill-hole interval data. The audit resulted in corrections to three assay values within the database, though 10 sample types (blank, standard, drill-hole sample, etc.) and seven assay types (1ATF versus SMF) were corrected.

The drill-hole geology data were checked for gaps or overlaps in the interval data and missing or incorrect geology coding. The data were checked using drill logs and cross-sectional plots; corrections or additions were made to six intervals. Additions and corrections to the geology data were made after the initial audit during the geologic modeling process. More detailed vein locations and designations were added to the database by SGM and these were all checked by MDA against the sectional interpretations.

12.2.1.2 Underground Sample Data Verification

The underground rock-chip channel samples are included in the database as horizontal drill holes with the orientation of the samples noted in the down-hole survey file. Where two or more channel samples form a continuous sequence of samples taken across a face, these samples are linked together with each sample representing a “down-hole” interval. The original database provided to MDA included sample intervals based on wallrock and vein geology contacts and not on the original chip-sample data. This resulted in artificial database intervals that either split the original sample, and corresponding assay value, into two database intervals or, in some cases combined original intervals into larger database intervals. Where the chip sample did not cover the full width of a particular vein, or the samples bracketed the vein boundaries, sample intervals with no sample data were inserted into the database so the full width of the vein could be conveyed in the database.

Before auditing the underground sample data, MDA reconstructed the sample database by revising all of the artificial interval breaks and lengths and changing back to the original sample interval data. Much of the re-construction was done by spreadsheet recognition of combined or split intervals. Where there was uncertainty as to the original interval, the data were checked against previous databases and/or the original sample card data if the latter were available. The database re-construction resulted in a reduction in total samples intervals from 1035 to 898. The reconstructed database still contains 120 intervals with no sample data that were left in the database to provide geologic information on vein location and widths.

MDA verified the collar location and orientation (down-hole survey data) of the underground samples using the digital data and the location data within the original sample cards. Upon completion of the 2009 survey of the underground workings, the sample locations were then plotted on-screen and compared with the 3-dimensional solid of the surveyed underground workings. Any spatial discrepancies were revised to better correlate with the underground workings.

The underground sample interval lengths and assay data were verified using original sample card data, digital copies of assay certificates, and previous database values when certificates are not available. All of the assays were compared against the master assay database compiled by SGM. The detailed audit resulted in changes to 90 assay values, most due to the inclusion of additional check assay data and a subsequent change in the final average value, and a change in 19 sample interval lengths based on sample card data.

12.2.2 Site Visits

MDA visited the site on March 19, 2009 and June 11, 2009. Project data and geology were reviewed with the project staff, and the underground workings were toured. The pre-existing Lincoln-Comet geologic model and resource estimate were reviewed in detail with Mark Payne (consulting geologist), who has been intimately involved with the project and who was the author of the 2008 Technical Report. The underground tour included viewing a number of vein exposures which provided significant insight into vein geometry and continuity.

In preparation for this updated technical report and the inclusion of the Keystone resource estimate, MDA visited the site on May 12 and 13, 2015 and June 20, 2015. The project status was reviewed with SGM personnel and MDA toured the underground development and surface mill facilities. While underground, MDA was able to observe evidence of both vertical and horizontal continuity along the primary mineralized veins.

12.2.3 Data Audit Summary

The audited and partially reconstructed project database is adequate for use in the development of a classified resource estimate and for further mine planning. There is some uncertainty as to the location of some of the underground samples, and some of the original historic data are missing, which limits the audit completeness, but the risk to the estimate is considered to be low.

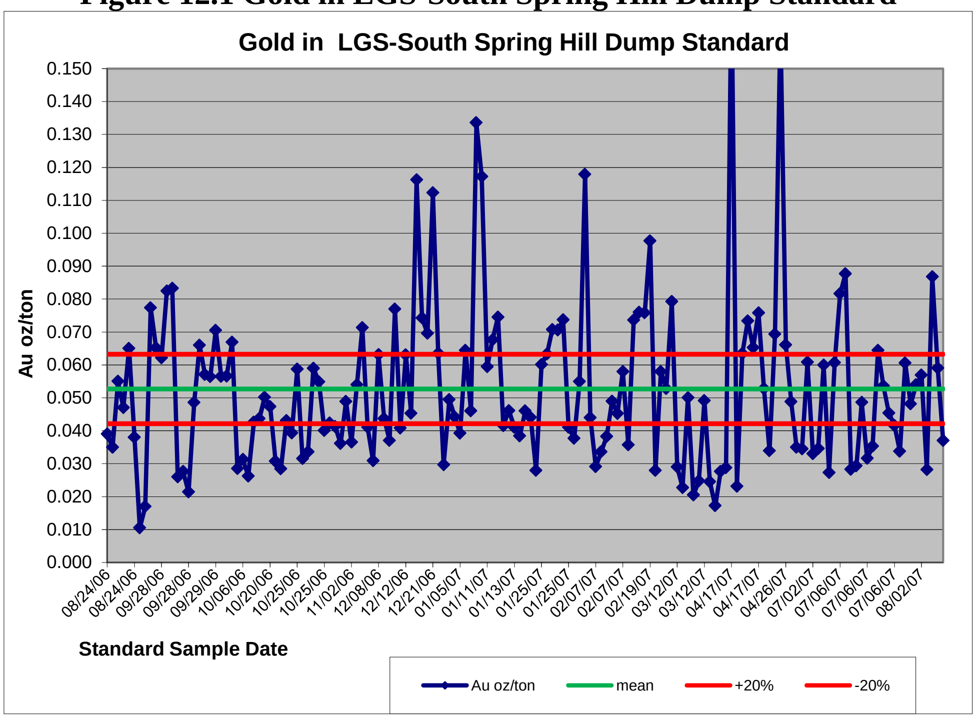
12.3 Quality Assurance/Quality Control

12.3.1 Standards

The Lincoln Mine database includes gold analyses for 212 reference standard samples inserted into the sample stream during SGM's 2006-2007 Lincoln-Comet and Keystone drill and underground sampling program. The standards are used to check the accuracy of the laboratory analyses. SGM created three primary standards by collecting and homogenizing large grab samples from three mine dumps on the Lincoln Mine property. Unfortunately, the analytical results for these standards were highly variable and the material collected performed poorly as acceptable standards.

As an example of the high variability, Figure 12.1 shows the gold values over time for the most used standard (LGS-South Spring Hill Dump). Included on the figure are the mean assay value (green line) and the +20 percent and -20 percent limits above and below the mean value. Of the 155 total analyses of this standard, 102 values are outside the plus or minus 20 percent limits. These data confirm the statements in Behre Dolbear (2007) that discuss the difficulty in creating acceptable standard samples from the Lincoln Mine mineralization. The lack of standard data means that the accuracy of the assay data is determined solely by the internal American Assay standards. The risk to the estimate is considered low.

Figure 12.1 Gold in LGS-South Spring Hill Dump Standard



12.3.2 Blanks

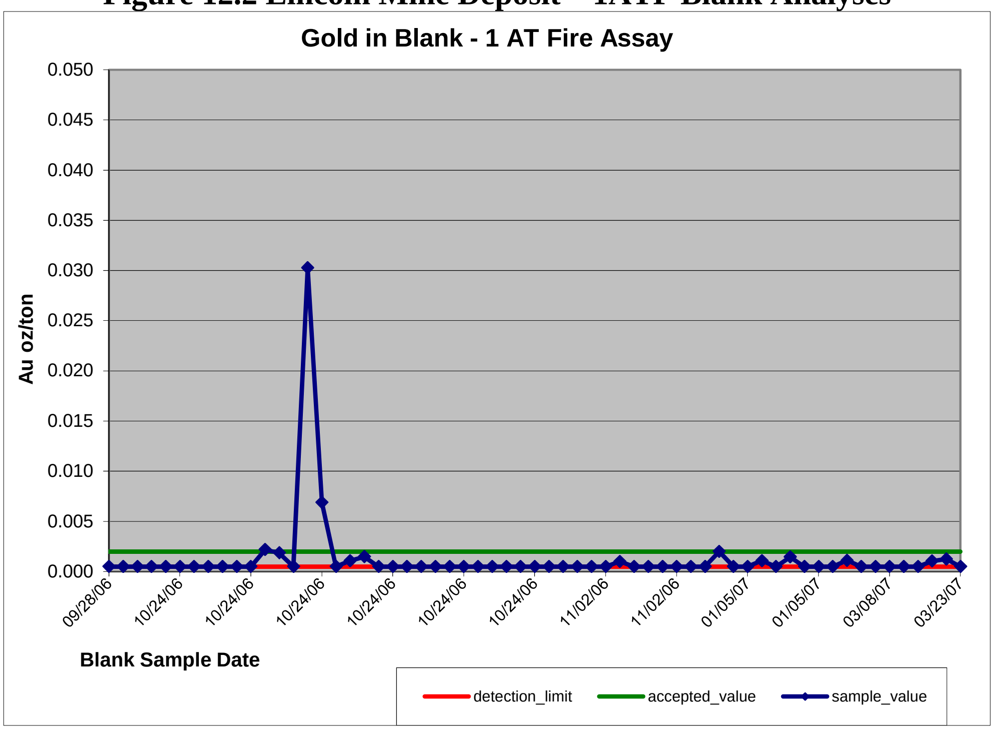
The Lincoln Mine database includes 567 blank analyses inserted into the sample stream sent to American Assay during SGM's 2006-2007 drill and underground sampling program. Blanks are inserted into the sample stream to check for contamination with gold during sample preparation due to the insufficient cleaning of the crushing, pulverizing, and sieving equipment. The blanks were inserted by the project's Qualified Person after every drill and underground sample containing over 25% vein quartz by volume (Payne, 2008). There is no record of any blank analyses for the pre-SGM work on the Lincoln Mine project.

As discussed in Section 11.2.5.3, the blank material used by SGM consisted of Lincoln Mine project drill core of unaltered and un-mineralized metavolcanic that contained no visible quartz. MDA has no record of a round-robin assay analyses of the core used in the blank program and can only evaluate the suitability of the blank material from the resultant analyses.

The blanks were assayed using both 1ATF and SMF techniques. A total of 61 1ATF blanks were analyzed. Of these, 35 were inserted into the sample stream directly after a drill of underground sample, and the remaining 26, including a continuous sequence of 21 blank samples, were assayed following a standard sample or another blank sample. The large majority of the 506 SMF blank samples were inserted directly after a strongly mineralized drill or underground sample and these results are a better indicator of contamination within the lab.

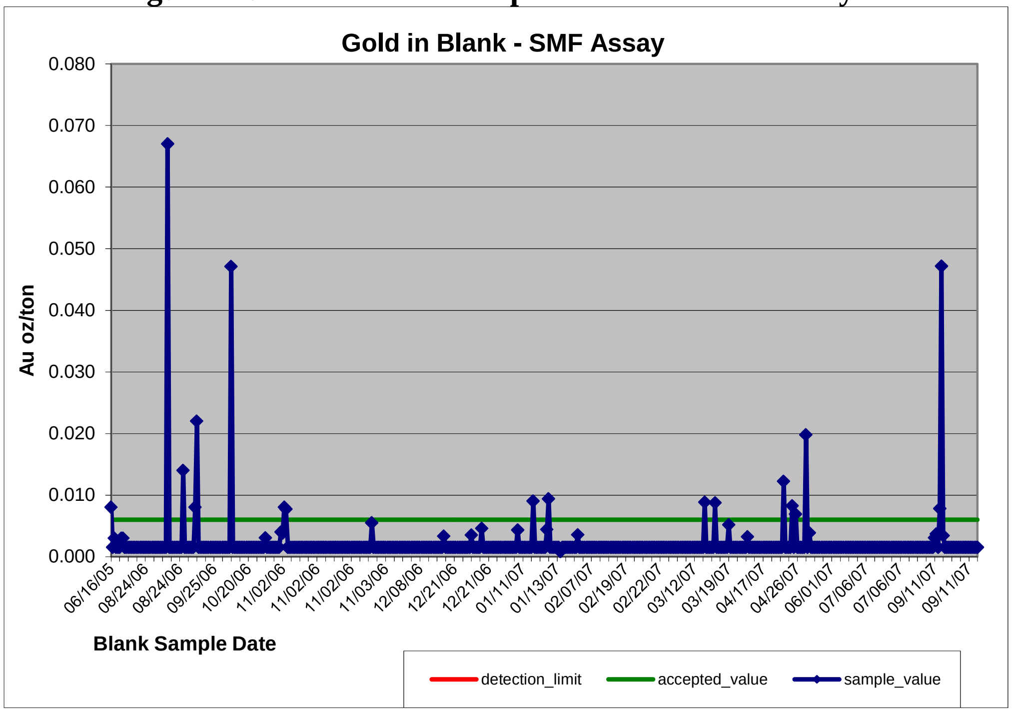
MDA plotted the blank analyses over time for the two assay types and the results are shown in Figure 12.2 and Figure 12.3. For both techniques, the blank values were predominantly at or below the detection limit for that specific technique; 0.001oz Au/ton for the 1ATF analyses and 0.003oz Au/ton for the SMF analyses. An accepted value is considered to be a blank value at or below twice the detection limit for that technique, which is noted as a green line in both figures. Two successive 1ATF blanks returned gold values above the accepted limit (see Figure 12.2). The initial high "failure" was preceded in the original sample stream by another blank that assayed less than detection, so the high blank value cannot be attributed to contamination with gold during sample preparation; it could be an analytical error, a weakly mineralized blank, or a clerical error (this sample was actually a standard). The second failure directly follows the first in the lab sample sequence and could reflect contamination. Overall, there is a low number and low level of contamination, and the results indicate a low risk to the resource estimate.

Figure 12.2 Lincoln Mine Deposit – 1ATF Blank Analyses



The SMF blank results in Figure 12.3 show a 3 percent failure rate with eighteen blanks that returned values above the 0.006oz Au/ton accepted limit. Of the eighteen failures, twelve were preceded in the sample stream by drill samples that assayed greater than 0.1oz Au/ton. The results do indicate that there is a low level of contamination within the drill sample results, but the potential error in gold content is low compared to the cutoff gold grades used in the current resource estimate. The risk to the estimate is considered low.

Figure 12.3 Lincoln Mine Deposit – SMF Blank Analyses



12.3.3 MDA Sample-Grade Reproducibility Study

The Lincoln-Comet and Keystone deposits are characterized by the prevalence of coarse gold and the resulting “nugget” effect, which creates risk in estimating a locally accurate resource. To ascertain the degree of risk, MDA completed a sample-grade reproducibility study in 2009 to better define the sub-sampling gold-grade variability. The 2009 study focused on the Lincoln-Comet mineralization since there are both drill and underground samples available for analyses and comparison. Due to the similarities in mineralization style, the author believes that the results of this study are likely to be representative of both the Lincoln-Comet and Keystone mineralization. The following sections provide the summary and then the more detailed results from this study.

12.3.3.1 Summary

Assay results from drill and underground sampling programs at the Lincoln-Comet area have indicated high variability in gold grades within the vein material, most likely due to the presence of coarse gold or possibly to gold occurring in coarse clots. This high variability occurs at all sub-sample stages from pulps all the way up to, it has been proposed, a macro or mining-round scale within and along the mineralized veins. The estimation of a *locally* accurate resource will, therefore, be difficult to achieve due to this inherent high sample-grade variability, imparting risk from using assay values that are potentially not representative with respect to the localized volume of rock.

In August 2009, the author (Tietz, 2009) performed statistical analyses on check assay data for combined drill-core and underground-chip samples, and for muck samples, to study the sample-grade reproducibility issue. While the drill-core and chip samples are mainly selective samples with a presumed homogeneous geologic character, the muck samples would generally be more heterogeneous in character because each round would likely contain both mineralized vein and unmineralized wallrock. The muck samples are, of course, also much larger than the drill or chip samples. This study analyzed three types of samples:

- pulp duplicates – 30g sample pulps taken from the same 250g to 500g pulverized sample;
- pulp replicates – newly pulverized splits taken from the sample's coarse reject; and
- twins – samples re-taken at the same location (i.e., the remaining half splits from core or underground samples taken from the same locations).

Table 12.1 summarizes the average assay variability for the duplicate, replicate, and twin pairs of drill-core and underground-chip sample data at various gold grades. Table 12.2 summarizes the same, but for muck samples. The data are further sub-divided by assay type, namely one-assay-ton fire assay ("1ATF") versus screened metallics fire ("SMF"). The range in the average variability noted in the tables for each sample/assay type reflects the presence of individual assay variability values that often are >1,000%. These high individual values can skew the data set average, but likely are real values that reflect the "nugget" aspect of the Lincoln-Comet mineralization.

Overall, there is an inherent variability in sample-grade results, ranging from an average of ~20% for pulp duplicates, to >~200% for twin samples. This variability reflects sampling and sub-sampling results only, and not spatial variability along or within a vein. The *apparent* variability along and within the vein is increased with the existing sub-sampling variability.

Table 12.1 Drill Core and Underground Chip Sample Variability

Au range (oz/ton)	Assay Type	Average Variability (%)		
		Duplicate	Replicate	Twin
>0.05	SMF	NA	50-65	100-200
>0.05	1ATF	15-25	60-100	NA
>0.1	SMF	NA	40-50	110-270
>0.1	1ATF	15-20	70-110	NA
0.05 - 0.2	SMF	NA	30-60	60-100*
0.05 - 0.2	1ATF	15-25	50-100	NA

* small sample population

Table 12.2 Muck Sample Variability

Au range (oz/ton)	Assay Type	Average Variability (%)		
		Duplicate	Replicate	Twin
>0.05	SMF	NA	60-70*	50-100
>0.05	1ATF	NA	75-90	100-150*
>0.1	SMF	NA	60-70*	50-70
>0.1	1ATF	NA	80-100	120-220*
0.05 - 0.2	SMF	NA	30-70*	30-50
0.05 - 0.2	1ATF	NA	50-75	100-150*

* small sample population

The data in Table 12.1 show an increase of 50%-70% in sample grade variability between the duplicate and replicate stages. This suggests that the effect of not pulverizing the entire coarse reject increases variability by two to five times. The high variability observed using the current sampling and assaying procedures, especially in the 1ATF samples, reduces confidence that any single assay can fairly reflect a sample's grade, and may potentially affect resource classification. It is noteworthy that twin samples add another 20% to 180% variability in gold assay results. Based on these results, the sub-sampling procedures should be modified to approach a complete crush and pulverization of the entire sample to get reproducibility down to a level where one can have reasonable confidence in analytical results.

Within the replicate data sets, the variability within the 1ATF data is consistently 20% higher than the replicate samples assayed by metallic screen techniques, evidence that metallic screen assaying is the preferred assaying technique. This increase in variability of 1ATF assay results over metallic screen results is accentuated in the muck sample twin data. For all sample types, the 1ATF assays are generally lower in grade than the comparable metallic screen values. This is true across all grade ranges. It is important to note that this study would not have recognized sub-sampling or analytical bias by lab, because of the manner in which the data exist,.

Any sampling at the Lincoln Mine project, including underground bulk-sampling, is subject to the high assay variability and consequent reduction of confidence in the assays to fairly reflect a sample's true grade. To lessen grade variability and increase confidence in the results, MDA recommends that a larger portion of the sample, or the entire sample, be pulverized and metallic screen assay techniques be used.

A likely serious consequence of this high sample grade variability for future mining or bulk sampling is that using the standard 1ATF assaying and/or sub-sampling, as was done in the past, will lead to waste versus "ore grade" mis-classification during mining. This could be ameliorated by using metallic screen assays and sub-sampling and smaller grind sizes.

An issue raised by this study was the general lack of twin data and specifically the twin data results for drill hole DDH-195. This is the only hole on the property with any twin data, and the results showed a consistent pattern of lower SMF twin values as compared to the original assays. Fourteen of the 18 twin pairs within this drill hole returned lower twin values, including three assay pairs which had a >500% negative difference. The variability is not surprising, but the possible sampling/assaying bias is a concern. As a check on the DDH-195 twin results, MDA looked at the eight sample intervals within

DDH-195 which have both twin and replicate assays. The results show that the mean population grades for the replicate and twin data show differences with the original data of -56% and -65%, respectively. The similarity of the replicate and twin results indicates a potential high bias within the original data. Much more data are needed before any firm conclusions can be drawn, and it is recommended that additional twin and replicate analyses be conducted on both this hole and other SGM core holes completed and assayed during the same time period as DDH-195.

The subsections that follow describe pulp duplicate, pulp replicate, and twin analyses results in greater detail.

12.3.3.2 Pulp Duplicate Analyses and Results

The SGM database compiled for this work contains a total of 888 pulp-duplicate analyses from drill-hole and underground-chip samples. These duplicates are from 53 original chip samples and 833 original half-split drill-core samples. The majority of duplicates (835 total assay pairs) are 1ATF pulp duplicates paired with a similar technique original assay. There are also a few duplicate/original pairs of 2-assay-ton fire assay ("2ATF") values and some mixed pairs of 1ATF *vs.* 2ATF values; neither of the latter combinations has a large enough population to be statistically relevant, and in the case of the 1ATF/2ATF pairs, the pairing of different assay techniques introduces another variable complicating interpretations of the 1ATF/1ATF pairs. No review of the 1ATF/2ATF pairs was done, and they were not included in the 1ATF/1ATF pairs analysis.

Multiple laboratories were used for these analyses with many of the duplicates assayed at the same laboratory as the original, while a significant portion was assayed at a second laboratory. This study did not look into laboratory variability, and consequently the variability noted in this report includes potential sub-sampling, sample preparation, and analytical variability between laboratories. Further analyses of the data broken out by specific laboratories are warranted.

Though not a critical issue for this study, it was found that the majority of the pulp-duplicate values are in-house, non-blind lab duplicates (the same lab assayed the original and duplicate with both samples having the same sample number).

Table 12.3 andTable 12.4 results show low mean and median population differences with similarly low relative difference values. Only the 0.01oz Au/ton and 0.05oz Au/ton relative difference values for the full data set (Table 12.3) show a negative bias (-5% and -6%, respectively) but, as shown by the <1% values for the same cutoffs in Table 12.4, these values result from the overly weighted influence of the outliers. Not unexpectedly, these data indicate sample bias is not a factor within the pulp duplicate data set and the original and duplicate assay populations are very similar.

The mean values for the absolute value of the relative difference (the measure of variability) range from 16% to 27% across both tables with the mean values above the 0.05oz Au/ton and 0.2oz Au/ton cutoff grades in Table 12.4 at 16% and 17%. For pulp duplicate samples, these are high values.

Table 12.3 Lincoln-Comet Pulp Duplicate 1ATF Analyses – All Data

>0.01 opt	Original	Pulp_Dupl	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	643	643		643	643
Mean	0.467	0.466	0%	-5%	27%
Median	0.111	0.106	-5%	0%	11%
Std. Dev.	1.221	1.197			
CV	2.613	2.568			
Min.	0.003	0.010		-1400%	0%
Max.	12.750	12.075		257%	1400%

>0.05 opt	Original	Pulp_Dupl	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	434	434		434	434
Mean	0.679	0.678	0%	-6%	23%
Median	0.205	0.204	0%	0%	9%
Std. Dev.	1.440	1.409			
CV	2.119	2.080			
Min.	0.019	0.050		-1400%	0%
Max.	12.750	12.075		156%	1400%

>0.2 opt	Original	Pulp_Dupl	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	221	221		221	221
Mean	1.228	1.223	0%	-1%	16%
Median	0.620	0.627	1%	1%	7%
Std. Dev.	1.860	1.816			
CV	1.515	1.485			
Min.	0.157	0.201		-134%	0%
Max.	12.750	12.075		102%	134%

Table 12.4 Lincoln-Comet Pulp Duplicate 1ATF Analyses – >250% Difference Pairs Removed

>0.01 opt	Original	Pulp_Dupl	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	636	636		636	636
Mean	0.472	0.471	0%	1%	21%
Median	0.112	0.107	-4%	0%	10%
Std. Dev.	1.227	1.203			
CV	2.598	2.556			
Min.	0.006	0.010		-183%	0%
Max.	12.750	12.075		182%	183%

>0.05 opt	Original	Pulp_Dupl	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	431	431		431	431
Mean	0.684	0.681	0%	0%	17%
Median	0.208	0.206	-1%	1%	8%
Std. Dev.	1.443	1.414			
CV	2.111	2.075			
Min.	0.027	0.050		-181%	0%
Max.	12.750	12.075		156%	181%

>0.2 opt	Original	Pulp_Dupl	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	221	221		221	221
Mean	1.228	1.223	0%	-1%	16%
Median	0.620	0.627	1%	1%	7%
Std. Dev.	1.860	1.816			
CV	1.515	1.485			
Min.	0.157	0.201		-134%	0%
Max.	12.750	12.075		102%	134%

Figure 12.4 and Figure 12.5 show the absolute value data (vertical scale) plotted against the mean value of the original/duplicate pair (horizontal scale). Trend lines of the absolute value data are shown in red. Figure 12.4 shows the data up to a grade of 2.0oz Au/ton, while Figure 12.5 shows that portion of the data within the 0 to 0.2oz Au/ton grade range. Figure 12.4 is essentially all of the data, missing only a few higher-grade assay pairs, and these additional pairs do not change the observed trend line. Figure 12.5 covers the grade range of the expected cutoff for the resource estimate and later production mining. Variability within this grade range is critical in providing insight into possible “ore-grade” versus waste mis-classification during mining.

The pulp duplicate variability within the higher-grade ranges (Figure 12.4) is fairly constant at 10%-15%, from about 0.1oz Au/ton to 0.7oz Au/ton, and then gradually rises to near 40% at 1.6oz Au/ton, possibly indicating a minor “nugget” effect at the pulp stage above 0.7oz Au/ton. The data are limited, but above 1.6oz Au/ton, variability drops back down to 10%-15% suggesting that the pulp once again becomes fairly homogeneous. This could be an artifact due to not having enough data points.

Figure 12.5 shows that at lower gold grades pulp variability has a step-down character with fairly sharp breaks at 0.03oz Au/ton and 0.1oz Au/ton. Below 0.03oz Au/ton, variability is between 30% and 50%, from 0.03 to 0.1oz Au/ton, it lessens to about 20%-30%, and above 0.1oz Au/ton, variability drops to around 10%-15%. As indicated in Figure 12.4 the 10%-15% variability remains constant up to about 0.7oz Au/ton. The higher variability within the lower grade ranges possibly results from coarse gold in not-large-enough concentrations to be statistically reproducible. At higher grades, the gold content is high enough to be statistically reproducible when assaying the sample pulp.

Figure 12.4 Pulp Duplicate Absolute Relative Difference – 0 to 2.0oz Au/ton Grade Range

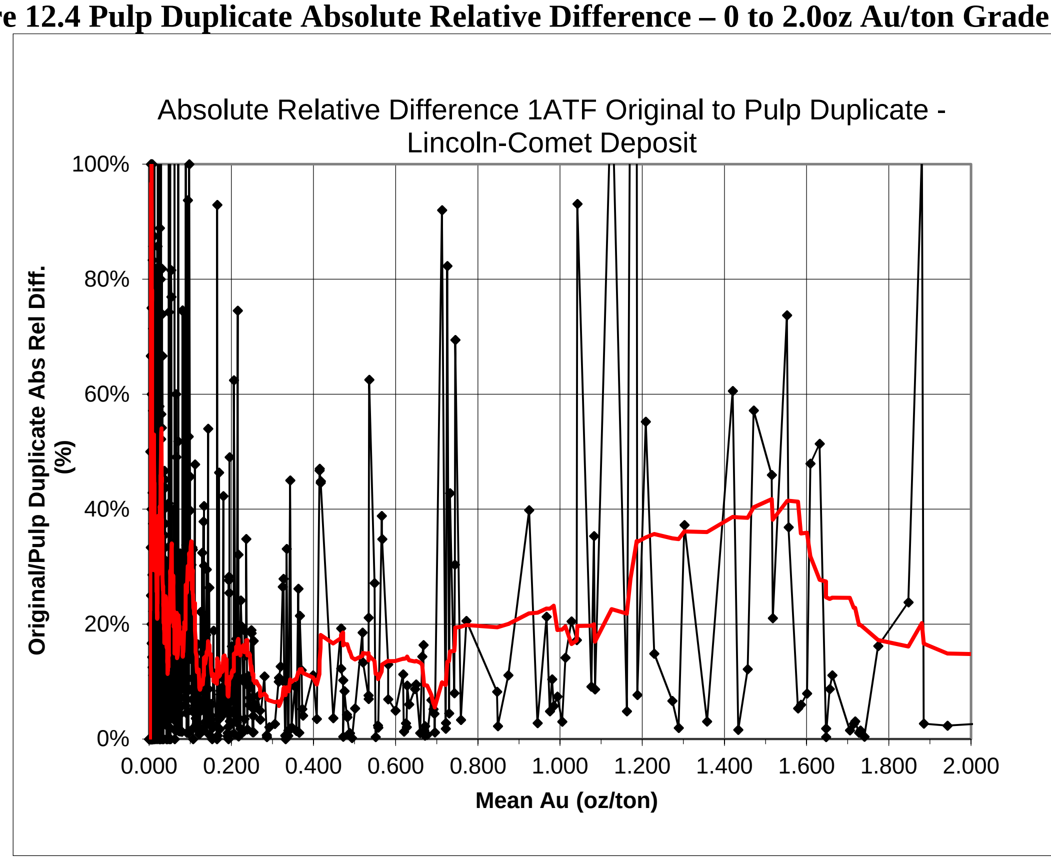
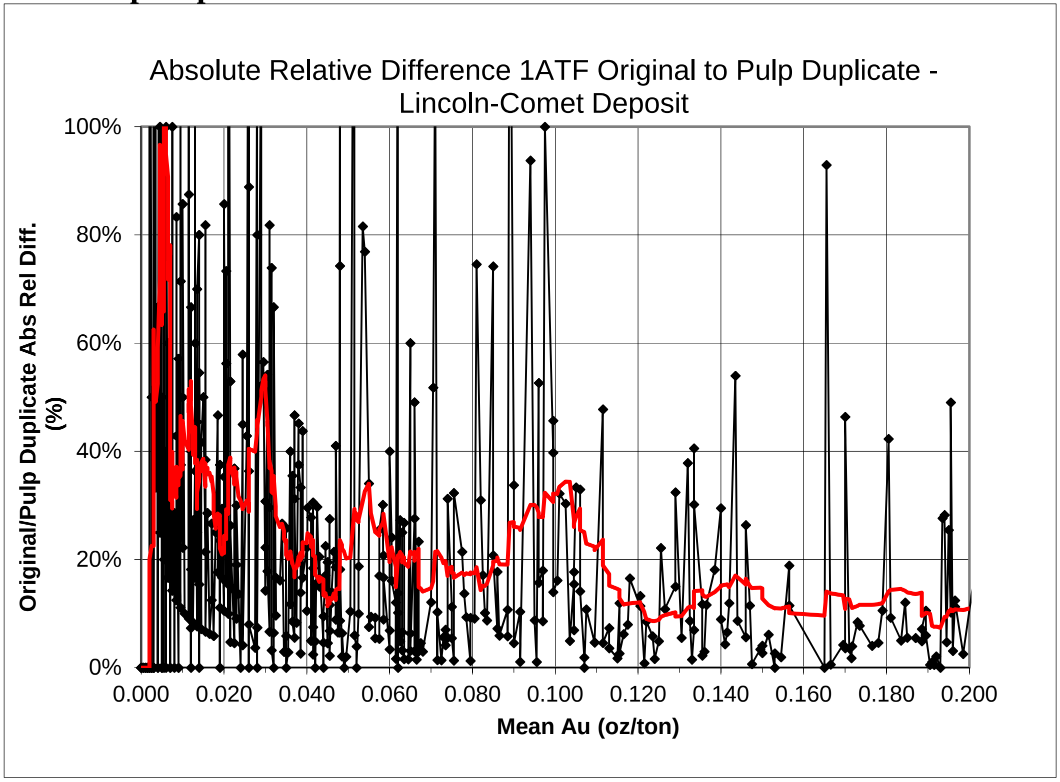


Figure 12.5 Pulp Duplicate Absolute Relative Difference – 0 to 0.2oz Au/ton Grade Range



12.3.3.3 Pulp Replicate Analyses and Results

The SGM database contains 2,991 pulp replicate analyses, which for the following discussion are divided into one group consisting of drill-core and underground-chip samples, and a second group consisting of muck samples collected from both underground trucks and larger muck-pile samples. Analyses are separated into 1 ATF and metallic screen fire ("SMF") assays in order to assess the variability in assay technique.

As compared to the duplicate analyses, which were often chosen at random by the laboratory responsible for the original assay, the replicates were chosen for analyses by project personnel. This results in a selection bias towards higher average gold grade samples and the consequent possible introduction of bias; *i.e.*, selecting mostly higher-grade samples for check analyses will result in a statistical tendency for lower-grade check assay values. The converse is also true; repeat analyses of mostly low-grade material should show generally higher-grade values.

12.3.3.4 Replicate Analyses – Drill Core and Underground Chip – 1ATF Analyses

Assaying by 1 ATF technique was done on 1,161 replicate/original pairs of historic drill-core and underground-chip samples. The statistical data are presented in Table 12.5 and Table 12.6; graphical presentation of the variability is presented in Figure 12.6 and Table 12.7. In contrast to the 250% difference level which marked the cutoff for the removal of "outliers" within the duplicate analyses, an assay difference of 500% is used for the replicate analyses. Even at this high variability, a total of 48 assay pairs were removed from the data, including three extreme outliers with >10,000% difference. The removal of this many samples is reflected in the significant difference in statistics Table 12.5 (all data except the three extreme outliers) and Table 12.6 (48 outliers removed). There is a noticeable negative bias when looking at all of the data (negative values for the "Diff" and Rel. Diff." columns), which is significantly tempered when the outliers are removed. The assay variability, as indicated by the "A. V. Rel. Diff." column, is almost double at all cutoffs for the full population, versus the population with outliers removed: about 130% down to 70%. If the outlier cutoff is raised to 1000% difference, which results in the removal of 29 assay pairs, the average variability at the various cutoffs increases by about 20%. Table 12.6 (48 outliers removed) shows the minimum variability in the replicate population and I dicates that the actual variability could be up to and over 100%. This is in contrast to the pulp duplicate results, which showed variability values in the 15%-20% range for the same type samples and same assay technique.

Table 12.5 and Table 12.6 do indicate that there is not an appreciable change in variability at the various cutoff grades. This is also reflected in Figure 12.6 and Figure 12.7, which plot the variability against the mean grade for the sample population with outliers removed. Figure 12.6 shows the data over the full grade range (up to 10oz Au/ton), while Figure 12.7 shows the data in the lower-grade ranges (up to 0.5oz Au/ton). In both, the high variability of individual samples is indicated, and the trend line shows a fairly constant rise from 50% at about 0.15oz Au/ton, up to 100% at just past 0.4oz Au/ton.

Table 12.5 Drill Hole/UG Chip - Replicate 1 ATF Analyses – All Data

>0.05	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	840	840		840	840
Mean	0.816	0.781	-4%	-18%	120%
Median	0.255	0.236	-7%	-2%	38%
Std. Dev.	1.788	1.710			
CV	2.192	2.189			
Min.	0.011	0.007		-4614%	0%
Max.	18.455	18.539		3391%	4614%

>0.1	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	683	683		683	683
Mean	0.986	0.944	-4%	-22%	130%
Median	0.350	0.345	-1%	-2%	39%
Std. Dev.	1.944	1.860			
CV	1.972	1.971			
Min.	0.014	0.007		-4614%	0%
Max.	18.455	18.539		3391%	4614%

>0.2	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	488	488		488	488
Mean	1.321	1.265	-4%	-20%	139%
Median	0.631	0.617	-2%	-2%	42%
Std. Dev.	2.213	2.117			
CV	1.675	1.674			
Min.	0.095	0.047		-2793%	0%
Max.	18.455	18.539		3391%	3391%

Table 12.6 Drill Hole/UG Chip - Replicate 1 ATF Analyses – >500% Difference Pairs Removed

>0.05	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	805	805		805	805
Mean	0.772	0.772	0%	-5%	68%
Median	0.253	0.236	-7%	-1%	36%
Std. Dev.	1.690	1.711			
CV	2.190	2.216			
Min.	0.023	0.018		-478%	0%
Max.	18.455	18.539		468%	478%

>0.1	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	652	652		652	652
Mean	0.935	0.935	0%	-6%	70%
Median	0.342	0.345	1%	-2%	36%
Std. Dev.	1.840	1.864			
CV	1.969	1.993			
Min.	0.034	0.032		-459%	0%
Max.	18.455	18.539		468%	468%

>0.2	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	460	460		460	460
Mean	1.264	1.267	0%	-4%	72%
Median	0.630	0.624	-1%	-1%	38%
Std. Dev.	2.106	2.134			
CV	1.666	1.684			
Min.	0.109	0.089		-443%	0%
Max.	18.455	18.539		455%	455%

Figure 12.6 Drill Hole/UG Chip - Replicate 1ATF Absolute Rel. Diff. – 0 to 10.0oz Au/ton Range
(48 outliers removed from analysis)

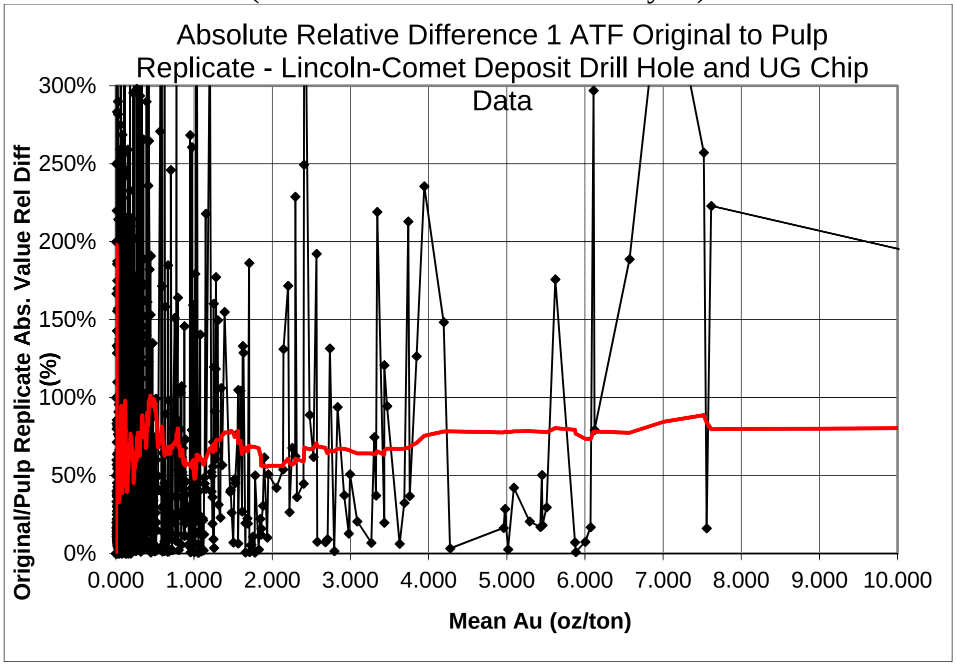
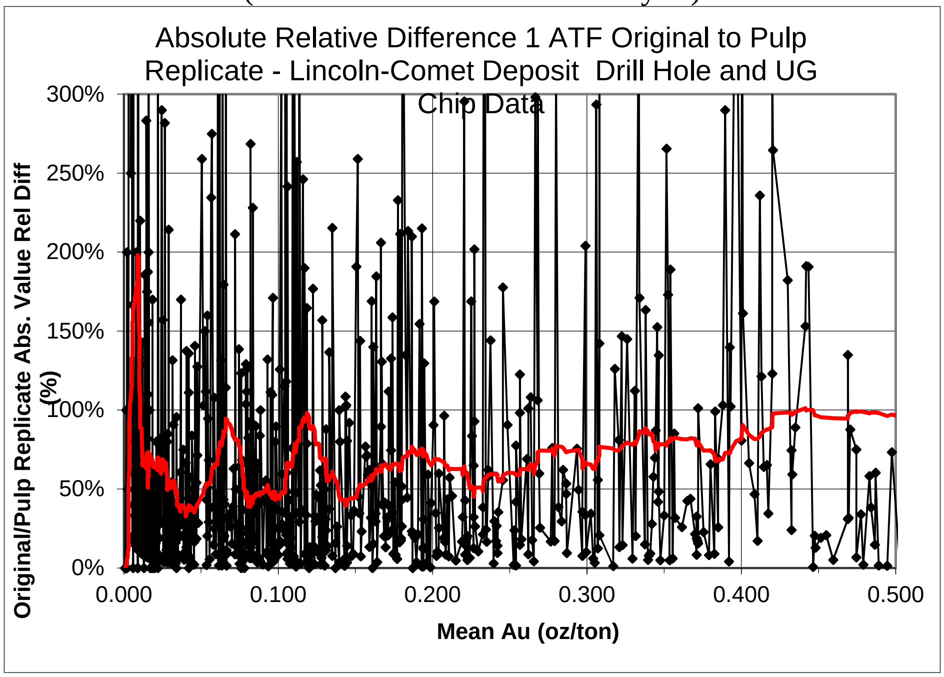


Figure 12.7 Drill Hole/UG Chip - Replicate 1ATF Absolute Rel. Diff. – 0 to 0.5oz Au/ton Range
(48 outliers removed from analysis)



12.3.3.5 Replicate Analyses – Drill Core and Underground Chip – SMF Analyses

Two hundred and seventy seven replicate/original pairs of SGM drill-core and some underground-chip samples were assayed by SMF technique. Results of statistical analyses are presented in Table 12.7, while the graphical representation of variability by mean grade are in Figure 12.8 and Figure 12.9. Similar cutoffs are shown in Table 12.7 as for the 1ATF analyses, and the same 500% difference determines the level to remove outliers. Just four outliers were removed from the SMF data, with the two highest percentage outliers occurring at grades <0.05oz Au/ton. Table 12.7 has a column added on the right side showing the variability of the population with the outliers included.

results show a noticeable contrast between the difference in mean population values ("Diff" column) and the mean value of the relative difference ("Rel. Diff." column) of the individual assay pairs. The mean of the replicate population is significantly higher (>40%) than the mean of the original population, but the relative difference of the individual pairs is low and, in fact, shows a trend to lower individual replicate assay values in the higher Au grades. Six of the eight highest mean grade assay pairs (>2.0 oz Au/ton) within the data set are from Meridian underground chip samples, and all six have a higher gold grade, often significantly higher (>100% difference), in the replicate versus the original. These high-grade assay values statistically dominate the replicate population and result in the observed significant increase in population grade. A review of the assay certificates for these chip-sample assay data indicate that there can be no distinction made between an original sample versus replicate, since all samples were split and analyzed at the same time at the same facilities. The population bias seen in the current data is created by having the sample order listed on the assay certificate, and subsequently in the SGM database, by increasing SMF gold grade with the higher assay value sample of the original-replicate pair always designated as the replicate analysis. This has resulted in the replicate always being higher grade than the original. If these six assay pairs are removed from the data set, the difference in assay populations is comparable to the relative difference values.

The variability in the SMF analyses with outliers excluded averages about 50%, with the high value of 60% at the 0.2 oz Au/ton cutoff. This increase in variability with grade is caused by the dominance of the significantly higher-grade replicates at mean grades over 2.0 oz Au/ton, as discussed above. If the eight highest grade samples are removed from the data set, the variability at the 0.2 oz Au/ton cutoff decreases to less than 50%. The SMF variability against grade is shown in Figure 12.8 and Figure 12.9. Over the full grade range in Figure 12.8, the high variability within the few samples above 2.0oz Au/ton is in evidence. It is not known if this increased variability at high grade is real or a function of the small data set. Within the lower grade ranges (Figure 12.9), the fairly constant trend of less than 50% variability is indicated above about 0.07oz Au/ton.

The SMF results show a pronounced decrease in variability compared to the previous 1ATF results. Compared to the 48 outliers with differences over 500% in the 1ATF data set, there are only four outliers in the SMF data. At the various cutoffs shown in Table 12.7 the SMF variability with outliers excluded is about 20% less than the 1ATF values (Table 12.6), while, if the outliers are included, the SMF variability is half that of the 1ATF results (Table 12.5). Within the expected mining cutoff grade ranges, the less than 50% SMF variability is in contrast to the 50% -100% variability indicated for the 1ATF population (Figure 12.7). All of the above data reinforce the preference for the use of SMF analyses in future sampling programs.

Table 12.7 Drill Hole/UG Chip - Replicate SMF Analyses

>0.05 opt Au	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.	A.V. Rel. Diff.
Count	83	83		83	83	85
Mean	0.570	0.837	47%	15%	52%	64%
Median	0.118	0.121	2%	-2%	19%	22%
Std. Dev.	1.174	2.316				
CV	2.058	2.769				
Min.	0.016	0.039		-186%	0%	0%
Max.	5.379	11.205		405%	405%	660%
>0.1 opt Au	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	51	51		51	51	52
Mean	0.890	1.321	48%	4%	40%	52%
Median	0.195	0.177	-9%	-3%	18%	18%
Std. Dev.	1.410	2.860				
CV	1.584	2.165				
Min.	0.088	0.093		-186%	0%	0%
Max.	5.379	11.205		226%	226%	660%
>0.2 opt Au	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	25	25		25	25	25
Mean	1.679	2.557	52%	7%	60%	60%
Median	0.826	0.802	-3%	2%	41%	41%
Std. Dev.	1.694	3.730				
CV	1.009	1.459				
Min.	0.202	0.145		-186%	0%	0%
Max.	5.379	11.205		226%	226%	226%

Figure 12.8 Drill Hole/UG Chip - Replicate SMF Absolute Rel. Diff. – 0 to 8.0oz Au/ton Range

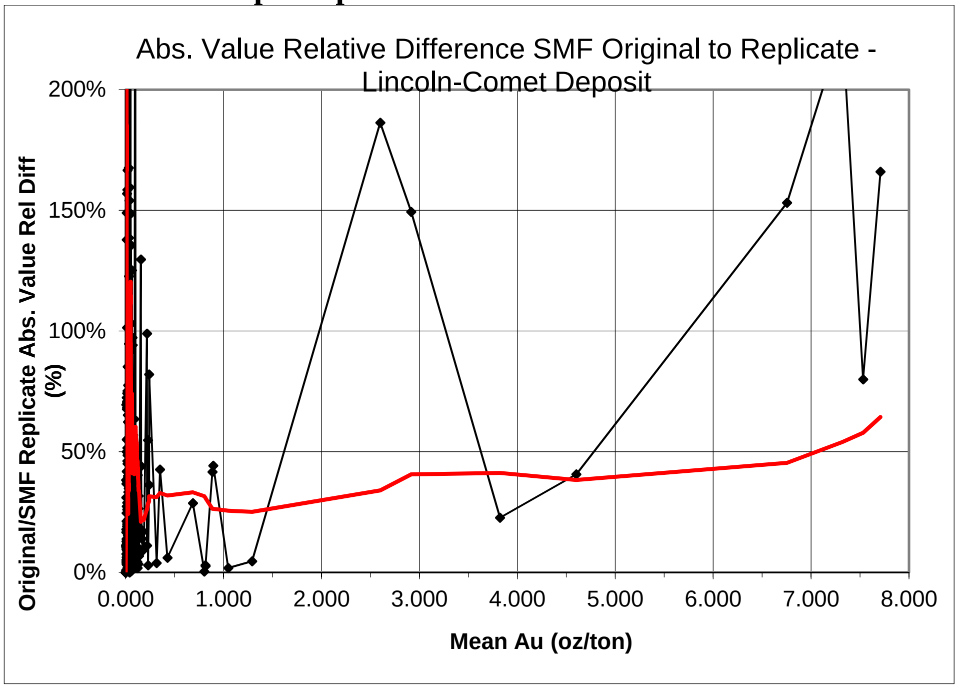
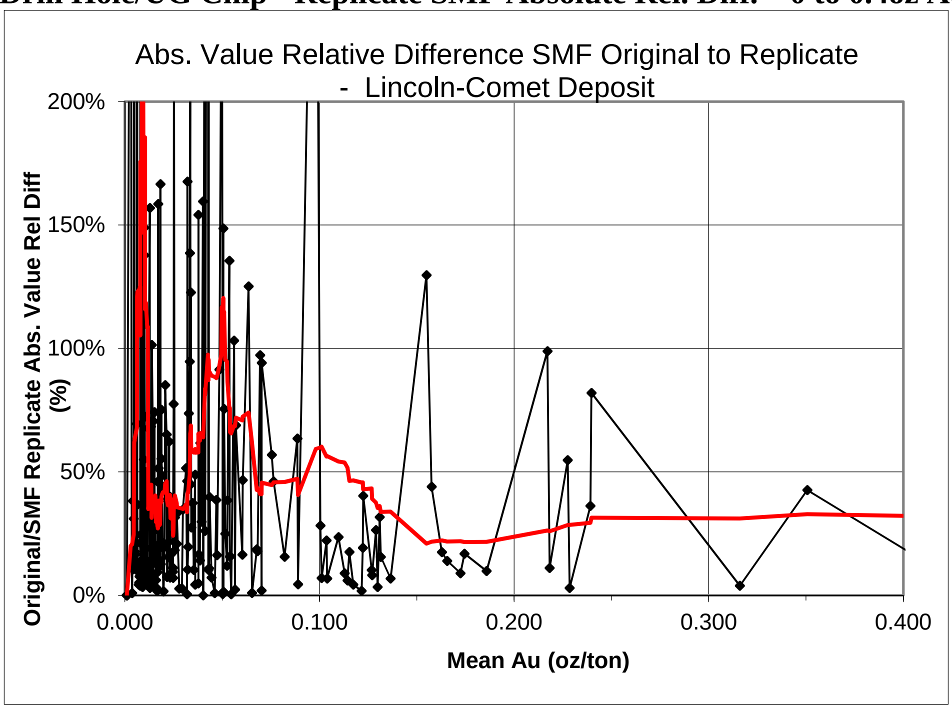


Figure 12.9 Drill Hole/UG Chip - Replicate SMF Absolute Rel. Diff. – 0 to 0.4oz Au/ton Range



12.3.3.6 Replicate Analyses – Drill Core and Underground Chip – SMF/1ATF Analyses

Two hundred and six replicate/original pairs of mostly Sutter Gold drill-core and underground-chip samples were assayed by both SMF and 1ATF techniques. Either the original was 1ATF and the replicate SMF or vice versa. For this study, the data were standardized with the SMF as the first assay and the 1ATF the second assay. This would allow for population differences and variations between assay types to be reflected in the statistical and graphical results. The statistical data are presented in Table 12.8, while the graphical representation of variability by mean grade is in Figure 12.10 and Figure 12.11. Seven outliers above 500% difference were removed from the data set, two of which exceeded 4,800% difference and one was 9,700%. The right-hand column in Table 12.8 shows the effect of these outliers on the variability values.

The 35%-40% variability indicated in Table 12.8 and shown graphically in Figure 12.10 and Figure 12.11 are somewhat surprising, but the small data set makes the conclusion somewhat suspect. The SMF/1ATF variability would be expected to fall in between the 1ATF/1ATF and the SMF/SMF pairs, but these data, when outliers are excluded, have consistently lower values than the SMF/SMF pairs. The various data sets do use different sample populations, and these spatial differences in sample origin could be reflected in the changing variability between assay pairs.

The population difference and the relative difference values in Table 12.8 indicate that the 1ATF assays are generally lower in grade than the comparable SMF values. A plot of the relative difference value against the mean gold grade is shown in Figure 12.12. Except for a minor area of positive relative difference (higher 1ATF assay values) at about 0.1oz Au/ton, the trend line remains negative throughout all grade ranges, indicating lower 1ATF assay values versus the SMF original.

Table 12.8 Drill Hole/UG Chip - Replicate SMF/1ATF Analyses

w/o 7 flyers (>500% A. V. Rel. Diff.)						All Data
>0.01	**SMF**	**1AFT**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	159	159		159	159	169
Mean	0.227	0.190	-16%	-11%	41%	175%
Median	0.082	0.083	1%	1%	20%	22%
Std. Dev.	0.526	0.336				
CV	2.320	1.768				
Min.	0.010	0.009		-465%	0%	0%
Max.	5.093	2.299		263%	465%	9700%
>0.05	**SMF**	**1AFT**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	111	111		111	111	115
Mean	0.312	0.260	-17%	-4%	40%	68%
Median	0.115	0.118	3%	2%	20%	21%
Std. Dev.	0.611	0.381				
CV	1.958	1.464				
Min.	0.040	0.034		-375%	0%	0%
Max.	5.093	2.299		263%	375%	1180%
>0.1	**SMF**	**1AFT**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	66	66		66	66	68
Mean	0.476	0.387	-19%	-12%	37%	65%
Median	0.227	0.203	-10%	2%	19%	19%
Std. Dev.	0.751	0.453				
CV	1.578	1.172				
Min.	0.080	0.072		-375%	0%	0%
Max.	5.093	2.299		72%	375%	1180%

Figure 12.10 Drill Hole/UG Chip - Replicate SMF/1ATF Absolute Rel. Diff. – 0 to 3.0oz Au/ton Range

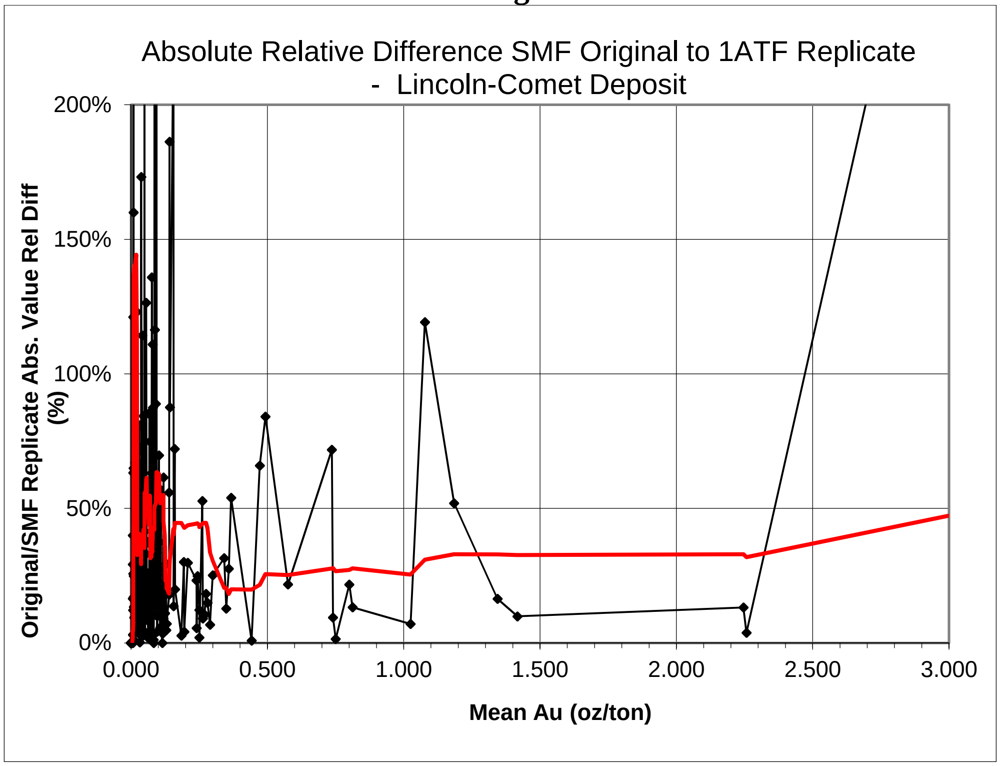


Figure 12.11 Drill Hole/UG Chip - Replicate SMF/1ATF Absolute Rel. Diff. – 0 to 0.4oz Au/ton Range

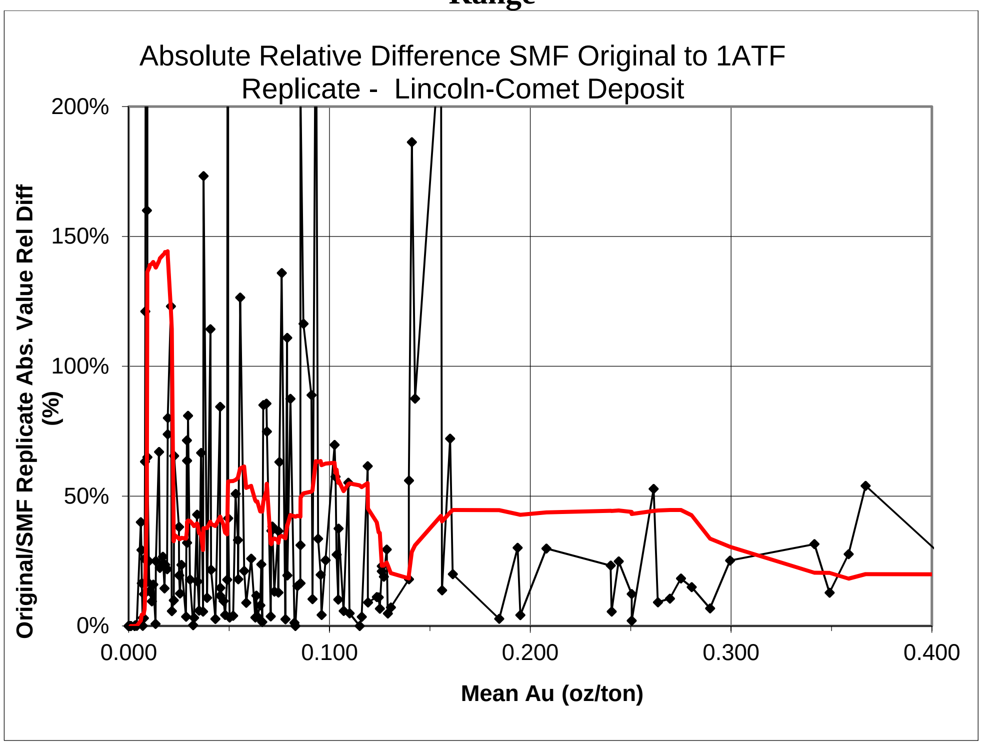
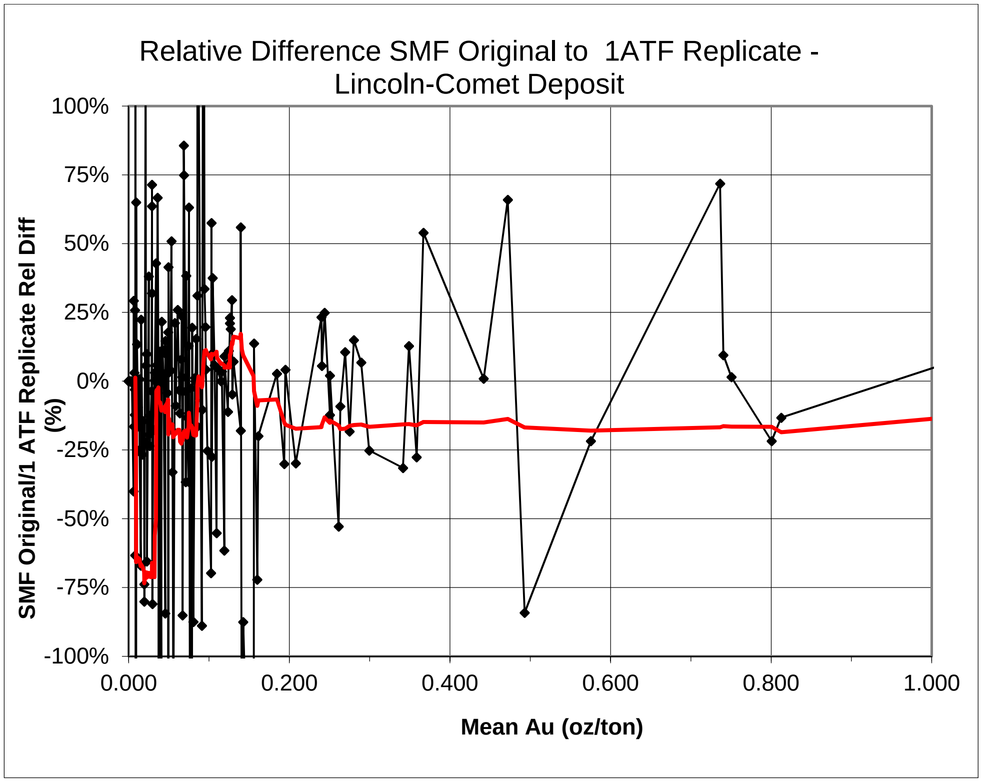


Figure 12.12 Drill Hole/UG Chip - Replicate SMF/1ATF Rel. Diff. – 0 to 1.0oz Au/ton Range



12.3.3.7 Replicate Analyses – Muck Samples – 1ATF Analyses

A total of 1,265 replicate/original pairs of muck samples were assayed by 1 ATF technique. The replicates, up to nine per single sample, were from 286 grab samples collected from single round muck piles or from material in "ore-grade" trucks. The muck grab samples weighed about 100lb and were crushed to 1/4in on-site. The crushed coarse reject material was then split into the individual replicate samples, each weighing about 15lb, and sent to the off-site laboratory for further preparation and assaying. The statistical data for the muck samples are presented in Table 12.9 and Table 12.10, while the graphical representation of the variability by mean grade is in Figure 12.13 and Figure 12.14. The same 500% difference cutoff is used for the removal of 35 outliers. As with the 1ATF pairs of the drill core and chip samples (Section 12.3.3.4), the large number of outliers indicates a "nugget" issue in sampled material and that the variability indicated in Table 12.10 is likely a minimum for this sample and assay type. The muck-sample results are similar to the drill-core and chip-sample variability values except for the higher variability at the 0.2oz Au/ton cutoff (101% in Table 12.10 versus 72% in Table 12.6).

Figure 12.13 and Figure 12.14 show the high individual muck-sample variability and the changes in variability with grade. The graphs show a similar, though more exaggerated, trend as the drill-hole and chip-sample data. There is an increasing variability from about 50% at 0.1oz Au/ton up to a high of about 125% between 0.3oz Au/ton and 0.4oz Au/ton. After dropping slightly, the trend line stays constant at about 100% past 0.4oz Au/ton. The increased variability of the muck samples at higher grades might be caused by the decreased sample reproducibility of the muck samples. The muck sample would contain both mineralized and un-mineralized rock, while the drill core and chip samples are often selectively sampled for a single rock type, and this differing sample reproducibility is reflected in the higher replicate variability.

Table 12.9 Muck Sample - Replicate 1 ATF Analyses – All Data

>0.05	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	1256	1256		1256	1256
Mean	0.261	0.266	2%	-3%	98%
Median	0.180	0.171	-5%	1%	50%
Std. Dev.	0.282	0.373			
CV	1.080	1.402			
Min.	0.007	0.007		-1989%	0%
Max.	2.949	5.149		3179%	3179%

>0.1	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	1106	1106		1106	1106
Mean	0.285	0.291	2%	-4%	101%
Median	0.192	0.189	-2%	1%	52%
Std. Dev.	0.292	0.391			
CV	1.022	1.344			
Min.	0.024	0.049		-1989%	0%
Max.	2.949	5.149		3179%	3179%

>0.2	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	575	575		575	575
Mean	0.415	0.422	2%	-15%	136%
Median	0.324	0.299	-8%	-4%	69%
Std. Dev.	0.356	0.506			
CV	0.857	1.200			
Min.	0.024	0.069		-1989%	0%
Max.	2.949	5.149		3179%	3179%

Table 12.10 Muck Sample - Replicate 1 ATF Analyses – >500% Difference Pairs Removed

>0.05	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	1222	1222		1222	1222
Mean	0.254	0.263	3%	-2%	77%
Median	0.180	0.173	-4%	1%	49%
Std. Dev.	0.271	0.371			
CV	1.064	1.408			
Min.	0.024	0.026		-487%	0%
Max.	2.949	5.149		481%	487%

>0.1	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	1075	1075		1075	1075
Mean	0.278	0.288	3%	-4%	81%
Median	0.192	0.189	-2%	1%	50%
Std. Dev.	0.280	0.389			
CV	1.008	1.350			
Min.	0.053	0.049		-487%	0%
Max.	2.949	5.149		481%	487%

>0.2	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	546	546		546	546
Mean	0.407	0.423	4%	-10%	101%
Median	0.322	0.301	-7%	-2%	61%
Std. Dev.	0.345	0.508			
CV	0.848	1.201			
Min.	0.074	0.069		-487%	0%
Max.	2.949	5.149		481%	487%

Figure 12.13 Muck Samples - Replicate 1ATF Absolute Rel. Diff. – 0 to 2.0oz Au/ton Range

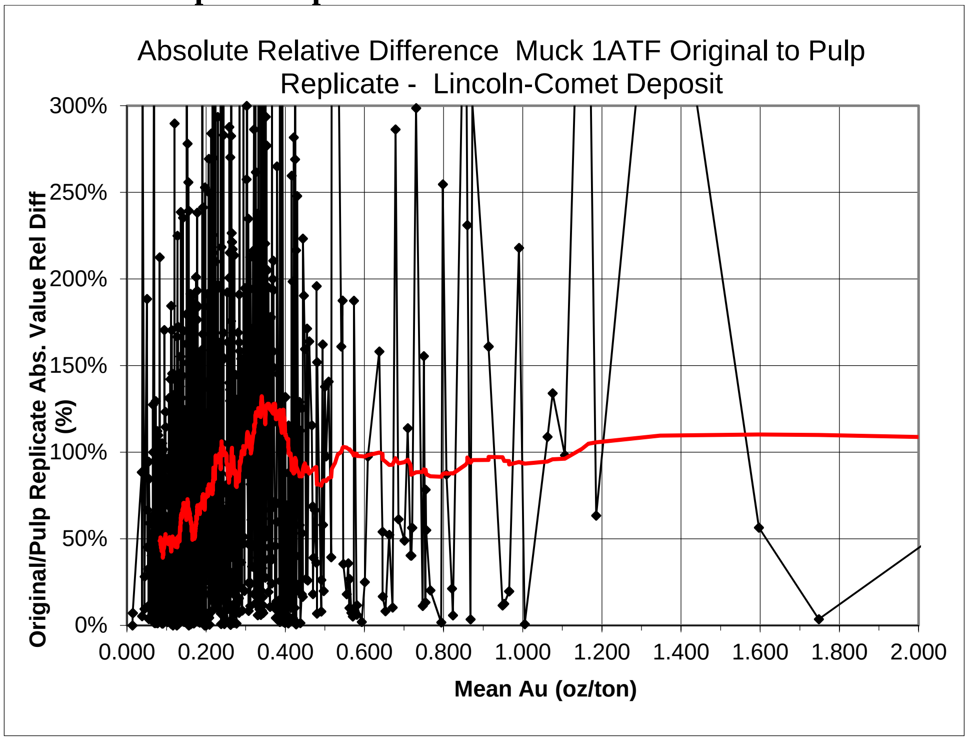
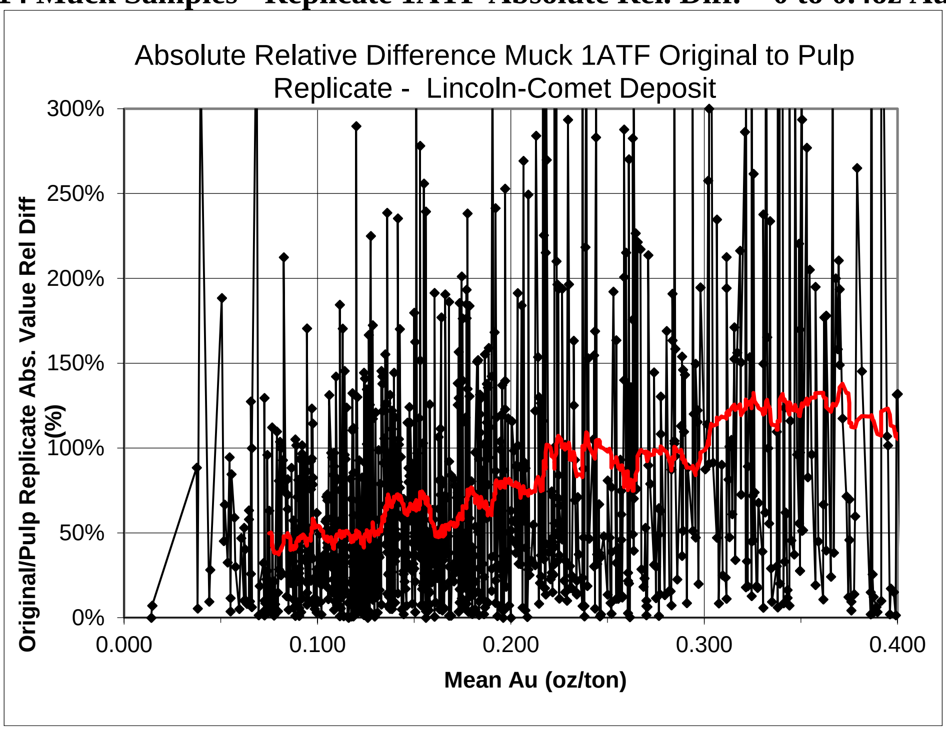


Figure 12.14 Muck Samples - Replicate 1ATF Absolute Rel. Diff. – 0 to 0.4oz Au/ton Range



12.3.3.8 Replicate Analyses – Muck Samples – SMF Analyses

A small number of replicate/original pairs (59 total pairs) of historic muck samples were assayed by SMF technique. The database indicates that each pair represents one individual sample. The statistical data for the SMF muck sample replicates are presented in Table 12.11. Almost all of the pairs have a mean grade between 0.1oz Au/ton and 0.3oz Au/ton, so the table includes data at only the 0.1oz Au/ton and 0.2oz Au/ton cutoffs. Two outliers over 500% were removed from the data set; one extreme value is over 6,100%.

For the muck samples, there is no real distinction between an "original" versus a replicate since all samples were split and analyzed at the same time at the same facilities. Any bias seen in the SMF replicate data, as reflected in the population difference and relative difference values, is a function of the small population size, how the samples are listed in the database, and the over-weighting of the few high-grade assays.

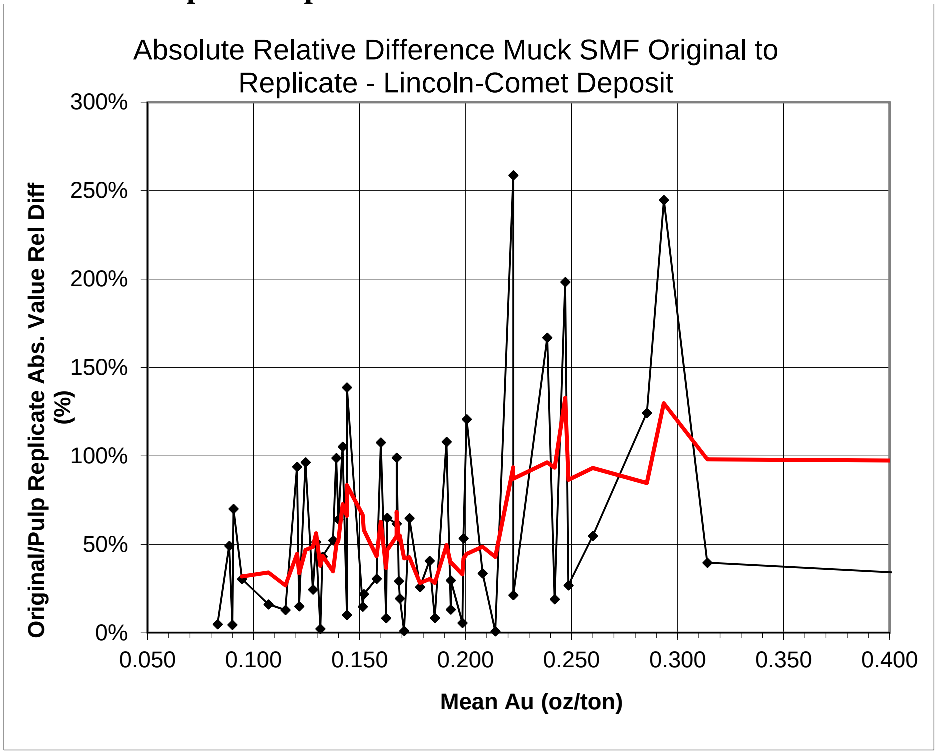
The variability of the SMF muck samples is about 10% to 20% less the 1 ATF muck samples. The graph of variability against mean grade (Figure 12.15) shows a similar progression as the 1ATF data. SMF variability is 40%-50% at the lower grades and then increases to near 100% above 0.2oz Au/ton, though the small data set makes any conclusions preliminary at best.

Table 12.11 Muck Sample - Replicate SMF Analyses – >500% Difference Pairs Removed

>0.1	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	51	51		51	51
Mean	0.219	0.239	9%	1%	60%
Median	0.176	0.169	-4%	8%	41%
Std. Dev.	0.172	0.256			
CV	0.785	1.069			
Min.	0.082	0.097		-259%	1%
Max.	0.869	1.351		245%	259%

>0.2	Original	Replicate	Diff	Rel. Diff.	A.V. Rel. Diff.
Count	16	16		16	16
Mean	0.369	0.410	11%	-18%	90%
Median	0.270	0.242	-10%	-10%	55%
Std. Dev.	0.241	0.412			
CV	0.654	1.003			
Min.	0.132	0.097		-259%	1%
Max.	0.869	1.351		245%	259%

Figure 12.15 Muck Samples - Replicate SMF Absolute Rel. Diff. – 0 to 0.4oz Au/ton Range



12.3.3.9 Twin Sample Analyses and Results

For the Lincoln-Comet area, twin samples consist of the remaining half core for the drill holes, a second underground chip sample across the same channel location, or a second hand grab from a muck pile. The twin assay should show the highest variability as compared to the duplicate or replicate values.

The SGM database contains a total of 157 twin samples: 127 muck twins and 30 drill-core and underground-chip twins. These data are predominantly SMF analyses of muck samples, with limited 1ATF muck, and SMF drill-core and underground-chip samples.

12.3.3.10 Twin Analyses – Muck Samples – SMF Analyses

There are 81 SMF twin pairs of muck samples within the database. The statistical data are presented in Table 12.12, while the graphical representation of variability is in Figure 12.16. Four outliers, with a high of 1,329%, were removed from the data set, though the variability results with these outliers included are shown in the right-hand column in Table 12.12 All of the outliers had mean values <0.1 oz Au/ton, so the population statistics are the same above this gold grade.

As with the replicate muck samples, there is no true distinction between an “original” versus a twin, since all muck samples were collected at the same time. Any bias seen in the data, as reflected in the population difference and relative difference values, is a function of how the samples are listed in the database and is not meaningful for this study.

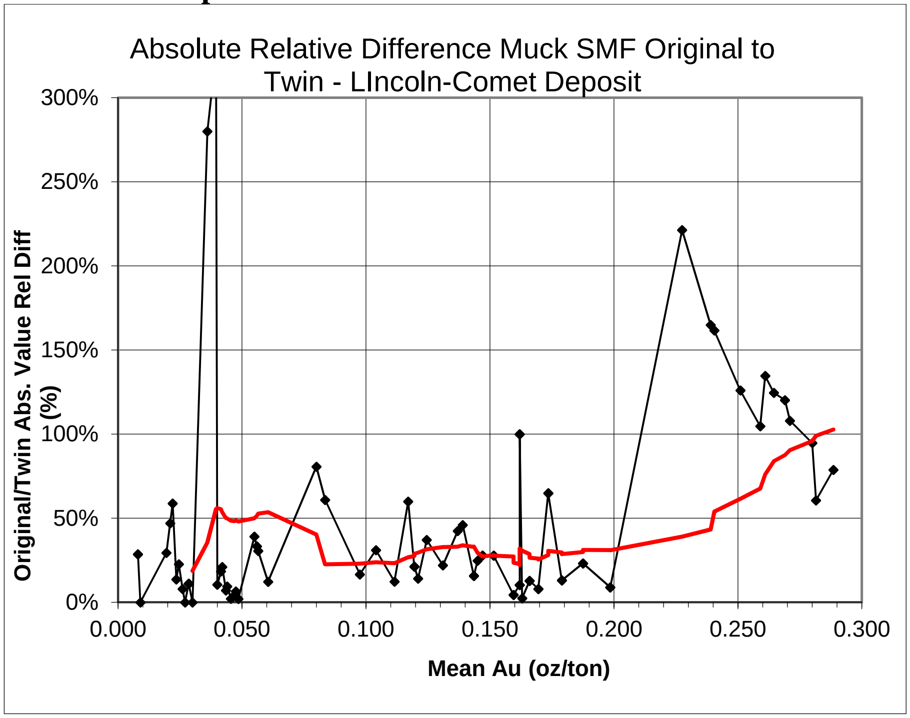
Figure 12.16 shows the trend of the variability up to 0.3oz Au/ton, which covers the full grade range of the data. The values are consistently low up to 0.2oz Au/ton, where there is a significant increase in variability up to and over 100%. More data are needed at the higher grade ranges.

The variability for the SMF muck twins shown is surprisingly lower than the SMF muck replicate sample grade variability. MDA cannot explain this but suggests that it could be a function of the small data sets or could reflect a primary difference in the muck samples used in each data set or there could be sample reproducibility differences due to rock type or spatial differences within the deposit.

Table 12.12 Muck Samples - Twin SMF Analyses

w/o 4 flyers (>500% A. V. Rel. Diff.)						All Data
>0.01	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	77	77		77	77	81
Mean	0.141	0.113	-20%	-12%	47%	93%
Median	0.131	0.129	-2%	-2%	22%	23%
Std. Dev.	0.110	0.063				
CV	0.778	0.558				
Min.	0.015	0.017		-221%	0%	0%
Max.	0.370	0.216		327%	327%	1329%
>0.05	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	54	54		54	54	58
Mean	0.187	0.146	-22%	-27%	50%	115%
Median	0.159	0.155	-2%	-14%	28%	31%
Std. Dev.	0.099	0.044				
CV	0.530	0.302				
Min.	0.048	0.046		-221%	2%	2%
Max.	0.370	0.216		81%	221%	1329%
>0.1	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	47	47		47	47	47
Mean	0.206	0.157	-24%	-33%	52%	52%
Median	0.168	0.163	-3%	-16%	28%	28%
Std. Dev.	0.093	0.036				
CV	0.454	0.227				
Min.	0.105	0.090		-221%	2%	2%
Max.	0.370	0.216		65%	221%	221%

Figure 12.16 Muck Samples - Twin SMF Absolute Rel. Diff. – 0 to 0.3oz Au/ton Range



12.3.3.11 Twin Analyses – Muck Samples – SMF/1ATF Analyses

There are 43 twin pairs of muck samples with mixed SMF and 1ATF assays. For this study, the data were standardized with the SMF as the first assay and the 1ATF the second assay. The statistical data are presented in Table 12.13, while the graphical representation of variability is in Figure 12.17. Three outliers above 500% difference were removed from the data set (one with a high of 1567%). The right-hand column in Table 12.13 shows the effect of including these outliers on the variability values.

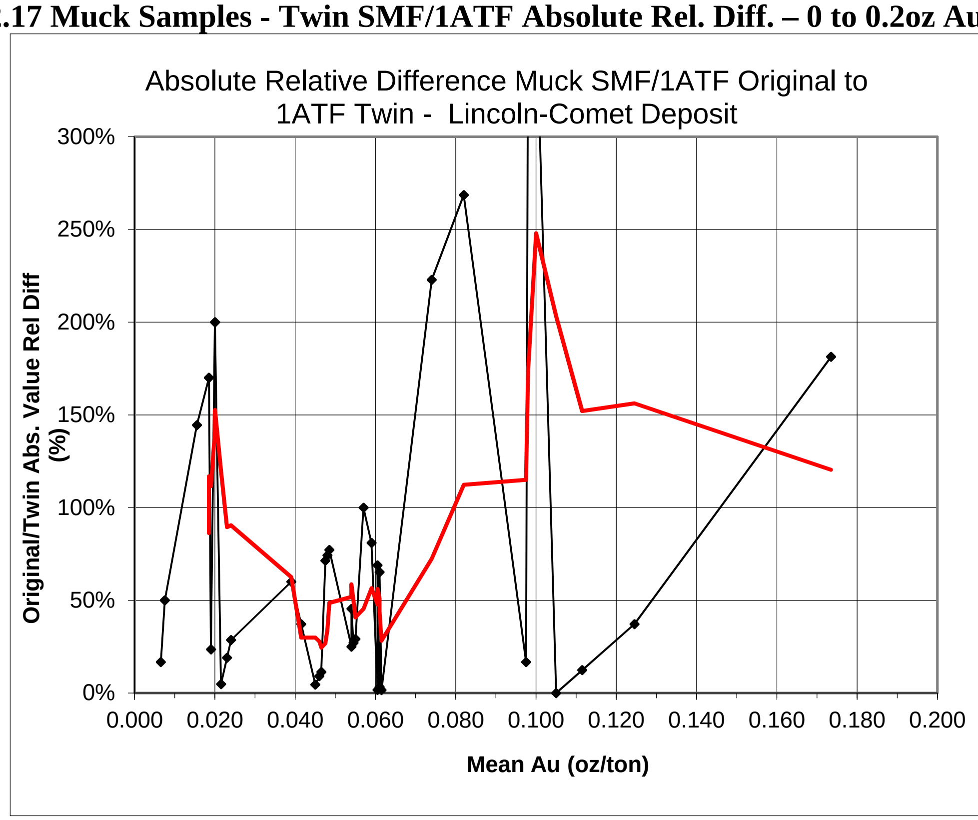
The population difference and the relative difference values in Table 12.13 indicate that the muck 1ATF assays are lower in grade than the comparable SMF values. This is true across all grade ranges and provides additional support for the use of SMF on the project.

The variability numbers are all significantly higher and more erratic (as seen in Figure 12.17) than the muck sample SMF-only twins, as would be expected.

Table 12.13 Muck Samples - Twin SMF/1ATF Analyses

w/o 3 flyers (>500% A. V. Rel. Diff.)						All Data
>0.01	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	37	37		37	37	39
Mean	0.068	0.048	-29%	-60%	87%	158%
Median	0.060	0.044	-27%	-27%	45%	60%
Std. Dev.	0.051	0.029				
CV	0.745	0.598				
Min.	0.009	0.010		-371%	0%	0%
Max.	0.256	0.105		144%	371%	1567%
>0.05	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	20	20		20	20	22
Mean	0.094	0.066	-29%	-62%	96%	220%
Median	0.063	0.062	-2%	-19%	41%	55%
Std. Dev.	0.056	0.026				
CV	0.591	0.386				
Min.	0.038	0.035		-371%	0%	0%
Max.	0.256	0.105		100%	371%	1567%
>0.1	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	5	5		5	5	6
Mean	0.158	0.088	-44%	-120%	120%	327%
Median	0.144	0.105	-27%	-37%	37%	109%
Std. Dev.	0.060	0.030				
CV	0.379	0.344				
Min.	0.105	0.035		-371%	0%	0%
Max.	0.256	0.105		0%	371%	1357%

Figure 12.17 Muck Samples - Twin SMF/1ATF Absolute Rel. Diff. – 0 to 0.2oz Au/ton Range



12.3.3.12 Twin Analyses – Drill Core and Underground Chip – SMF Analyses

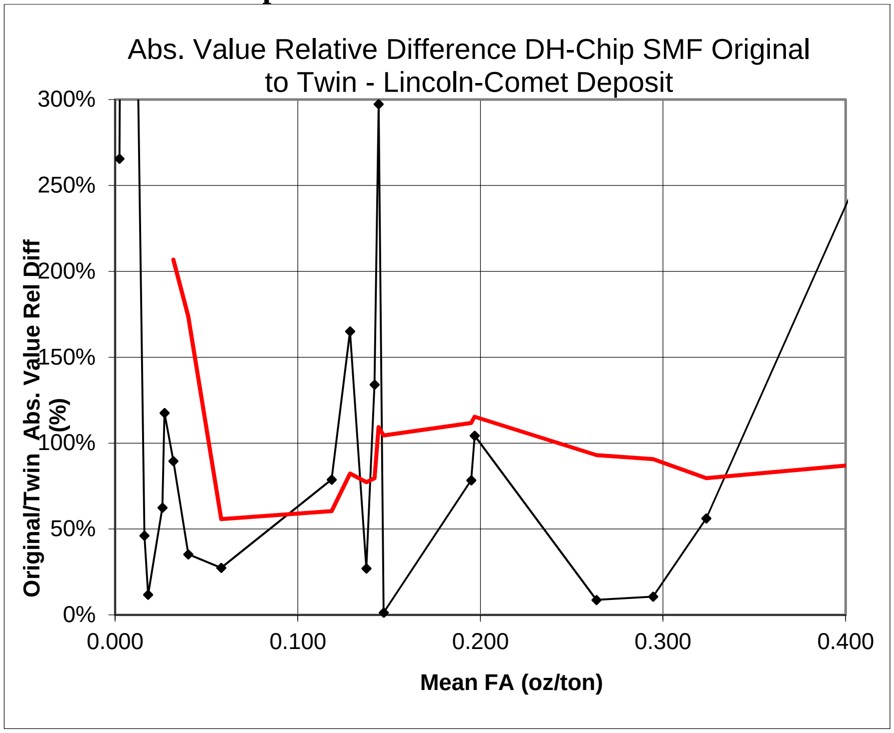
There are just 25 SMF twin pairs of drill-core and underground-chip samples within the database. The statistical data are presented in Table 12.14 and graphically in Figure 12.18. Three outliers, with a high of 2,928%, were removed from the data set, though the variability results with these outliers included are shown in the right-hand column in Table 12.14. The limited data set results in a significant increase in the already high variability when the outliers are included. Much more data are needed before any firm conclusions can be drawn for this population.

Figure 12.18 shows the trend of the variability at grades below 0.4oz Au/ton. The values are erratic, but the initial indications are that variability will likely be <100% within this grade range.

Table 12.14 Drill Hole/UG Chip - Twin SMF Analyses

w/o 3 flyers (>500% A. V. Rel. Diff.)						All Data
>0.01	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	20	20		20	20	23
Mean	0.356	0.165	-54%	-48%	101%	265%
Median	0.150	0.125	-17%	-37%	70%	79%
Std. Dev.	0.843	0.218				
CV	2.368	1.326				
Min.	0.019	0.013		-392%	1%	1%
Max.	3.854	1.011		297%	392%	2928%
>0.05	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	14	14		14	14	15
Mean	0.496	0.225	-55%	-56%	119%	306%
Median	0.218	0.151	-31%	-42%	79%	79%
Std. Dev.	0.983	0.238				
CV	1.982	1.059				
Min.	0.058	0.051		-392%	1%	1%
Max.	3.854	1.011		297%	392%	2928%
>0.1	**Original**	**Replicate**	**Diff**	**Rel. Diff.**	**A.V. Rel. Diff.**	**A.V. Rel. Diff.**
Count	13	13		13	13	14
Mean	0.529	0.238	-55%	-58%	126%	326%
Median	0.250	0.157	-37%	-56%	79%	92%
Std. Dev.	1.015	0.242				
CV	1.918	1.017				
Min.	0.058	0.070		-392%	1%	1%
Max.	3.854	1.011		297%	392%	2928%

Figure 12.18 Drill Hole/UG Chip - Twin SMF Absolute Rel. Diff. – 0 to 0.4oz Au/ton Range



The strongly negative population difference and relative difference values in Table 12.14 are caused by a consistent pattern of lower SMF twin values, as compared to the original twin assays for drill hole DDH-195. Fourteen of the 18 twin pairs within this drill hole returned lower twin values, including three assay pairs which had a >500% negative difference. The variability is not surprising, but the prevalence of lower-grade twin values suggests a possible sampling or assaying bias within one of the sample sequences. Due to the limited twin analyses and the lack of standards or check analyses to check lab accuracy, it cannot be determined which sequence of samples has a potential problem.

As a check on the DDH-195 twin results, MDA looked at the eight sample intervals within DDH-195 which have both twin and replicate assays and have mean assay values over 0.01oz Au/ton. Table 12.15 shows the original versus replicate results in the first row and the original versus the twin sample results in the second row. One >500% difference outlier occurs in each data set, though instead of removing these outliers thereby making the sample populations even smaller, the outlier values were reduced to 500%. The results show that both the replicate and twin data have similar large differences with the original data. Mean population grades show differences of -56% and -65% for the replicate and twin data, respectively, while the relative difference values are -97% and -114%. The similarity of the replicate and twin results indicates a potential high bias within the original data. Much more data are needed before any firm conclusions can be drawn, and it is recommended that additional twin and replicate analyses be conducted on both this hole and other Lincoln-Comet core holes completed and assayed during this time period.

Table 12.15 DDH-195 Check Sample Comparison – Replicate versus Twin Assays for Eight Sample Intervals

Mean Grade (oz Au/ton)					
Original	**Replicate**	**Twin**	**Diff**	**Rel. Diff.**	**Abs. Diff.**
0.680	0.298	-	-56%	-97%	132%
0.680	-	0.236	-65%	-114%	205%

12.3.4 Core Recovery versus Metal Grade Analyses

SGM provided MDA with the core recovery data for 169 of the 194 core holes used in the current Lincoln-Comet resource estimate. MDA checked the recovery data calculations and spot-checked the measurements against the core photos. No calculation errors were noted and only minor discrepancies in core recovery readings were in evidence.

MDA analyzed the relationship between gold grade and core recovery. Figure 12.19 shows the relationship between gold grades and core recovery for those sample intervals assaying greater than 0.01oz Au/ton. The gold grade and number (“Count) of core recovery intervals are presented in the left-hand and right-hand y-axis, respectively. These values are sorted into core recovery “bins” of regular 10 percent intervals as noted along the x-axis. (Each bin represents all intervals within each 10 percent interval; for example, recovery column “80” shows the average gold value and number of sample intervals for all intervals with core recovery values between 80 and 89 percent.)

The data in Figure 12.19 shows a noticeable increase in average gold grade as core recovery decreases from 100 percent to 60 percent. The number of sample intervals decreases significantly once recovery values drop below 90 percent but the trend in increasing gold grade is seen in the 60, 70, and 80 percent columns. Due to the small number of intervals with recoveries below 50 percent, the observed decrease in gold grade is not believed to be statistically relevant.

Figure 12.19 Core Recovery versus Gold Grade Comparison
Sample Intervals assaying >0.01 oz Au/ton

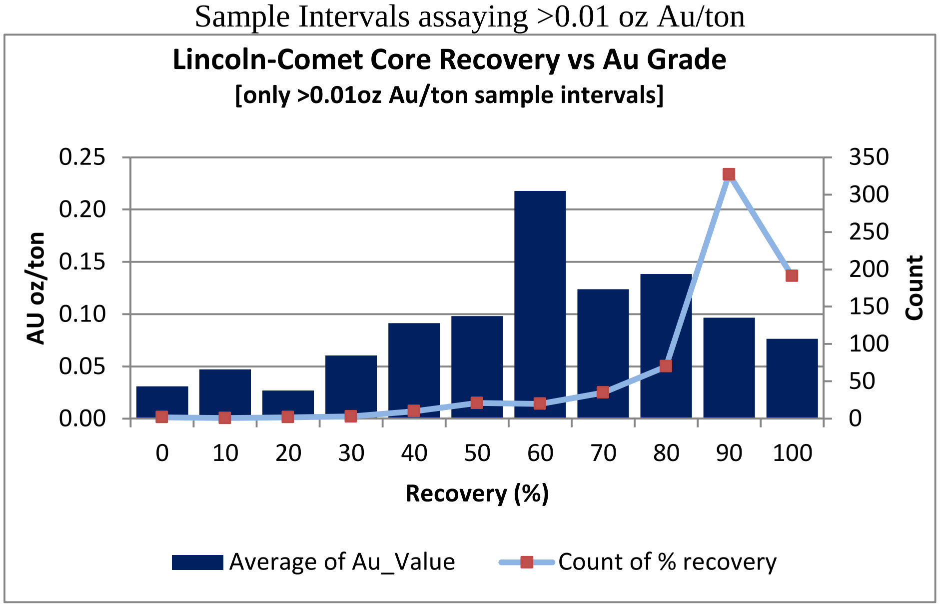
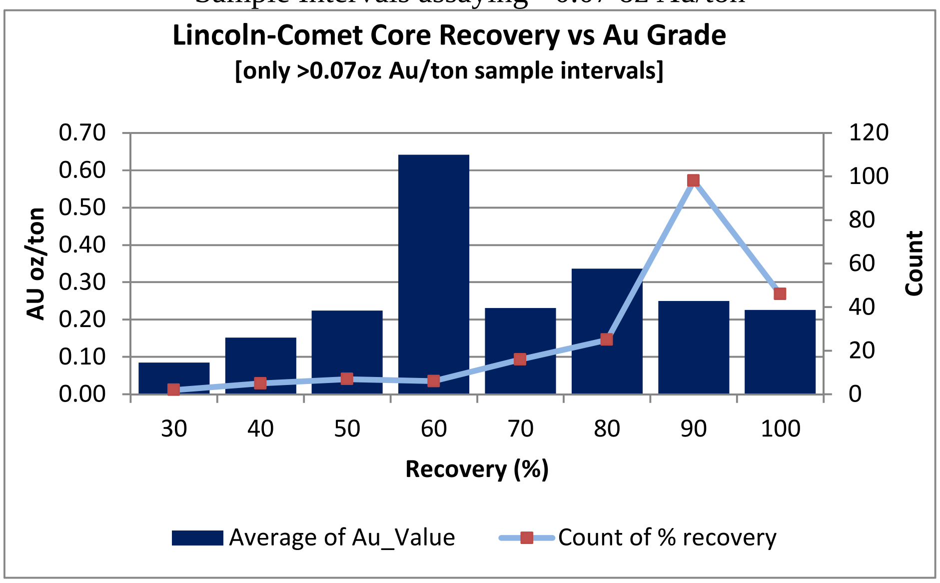


Figure 12.20 shows the same relationship between gold grades and core recovery but the data has been filtered to show only those sample intervals assaying greater than 0.07oz Au/ton to better represent the core recovery effect on significant gold grades. The sample count is much lower but the same trend of increasing grade with decreasing recovery is seen in this figure. Much of the pronounced increase in grade associated with core recovery values in the 60 percent range results from one sample interval that assays over 2.0oz Au/ton. Removing this one interval from the sample population drops the average grade to just under 0.3oz Au/ton which still corroborates the general trend of increased grade with moderate recoveries.

The inverse relationship between core recovery and grade observed in Figure 12.19 and Figure 12.20 is reflected in the drill core where the mineralized veins are often more fractured and the core more broken than in the lower-grade country rock. As the gold mineralization is often associated with late shears within the more massive quartz veins, it is not yet understood if the increasing grade with lower core recovery is reflective of a selective increase in grade with core loss, or whether the grade-recovery relationship is a natural reflection of the fractured nature of the more strongly mineralized veins.

Figure 12.20 Core Recovery versus Gold Grade Comparison
Sample Intervals assaying >0.07 oz Au/ton



Overall, the core recovery data indicate moderate to good core recovery within the mineralized horizons and support the estimation of the Lincoln-Comet resource.

12.4 Data Verification Summary and Conclusions

MDA conducted various verification procedures on the Lincoln-Comet drill-hole and underground sample data to be used in the current resource estimate. These procedures included four site visits, an audit of all historic and SGM data, a review and analyses of much of the QA/QC data, and core recovery versus gold grade studies.

The database audit included a detailed audit and reconstruction of all available underground chip sample data. Only a few significant errors in the database were noted and corrected. There is minor uncertainty as to the location of some of the underground samples, and portions of the original pre-1994 data are missing, but the risk to the estimate is believed to be low. MDA considers the project database to be adequate for use in the development of a classified resource estimate and for further mine planning.

There is limited blank sample and no acceptable standard sample quality control analyses on the project assay data. The gold-grade reproducibility study has indicated high variability in gold grades within the vein material, most likely due to the presence of coarse gold or possibly to gold occurring in coarse clots. This high variability occurs at all sub-sample stages from pulps all the way up to, it has been proposed, a macro or mining-round scale within and along the mineralized veins. The estimation of a *locally* accurate resource will, therefore, be difficult to achieve due to this inherent high sample-grade variability. Moderate to high risk is imparted from using assay values that are potentially not representative of the localized volume of rock. This risk can be lessened to some extent by employing sample preparation techniques that pulverize the entire sample and then analyze by metallic screen fire assay.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Section 13.0 was prepared by Dr. Corby G. Anderson, QP, CEng, FIChemE, with Allihies Engineering Inc. of Butte, Montana. Allihies Engineering Inc. was provided with metallurgical testing, analysis, and processing, piloting, and engineering studies related to the Sutter Creek Gold Mine and was asked to perform a professional review of these materials in support of this Technical Report in accordance with NI 43-101. This was completed, and a summary of this review is included below. In conclusion, a substantial body of fundamental and applied testing and engineering metallurgical studies from the laboratory through pilot and industrial process scale are available to support the preparation of a high-quality, advanced NI 43-101 Technical Report.

In this section, "ore" is used as a descriptive metallurgical term and is not intended to reflect an economic classification of material to NI 43-101 standards.

13.1 Introduction

Behre Dolbear (2007) provided the following summary of metallurgical testing from the Sutter gold project:

> *"Since 1989, samples from the Sutter Creek Gold Mine have been investigated at HRI, Dawson Laboratories, and Kappes Cassiday, Inc. Both bench-scale and pilot plant scale testing were conducted. The metallurgical response, as measured at each laboratory, was similar. Gravity recoveries, at relatively coarse grinds, typically ranged from 23 to 54 percent. In the most extreme example, gravity recoveries as high as 90 percent were achieved. Typical flotation recoveries (based on the overall plant feed) contributed an additional 45 to 74 percent gold recovery. Overall recoveries were generally greater than 90 percent."*

Behre Dolbear (2007) recommended that additional pilot-scale testing be undertaken that incorporates complete locked cycle tests utilizing both gravity and flotation circuits. They opined that the prior test programs were conducted on samples that were not necessarily representative of the resources as they were then defined. Specifically, metallurgical work more recent than 1989 has been done on higher gold head grades exceeding 0.5oz Au/ton, which may have biased the gravity gold recovery on the high side.

SGM contracted McClelland Laboratories, Inc., through Mr. Herb Osborne, an independent metallurgical consultant, to complete the locked cycle tests and to provide final laboratory analysis for mill circuit development. McClelland Laboratories, Inc. completed this work in 2009, which provides the majority basis for metallurgical recovery estimates and mill design (see Table 13.7).

The following information in Sections 13.2 through 13.8 is taken from Ronning and Prenn (2004) of Mine Development Associates, and from Behre Dolbear (2007), who summarized the numerous studies on metallurgy and mineral processing undertaken by prior operators.

13.2 Metallurgical Studies by Hazen Research Inc., 1989

Hazen Research Inc. ("Hazen") tested mineralized material that contained both coarse free gold and gold intimately associated with pyrite and arsenopyrite. The purpose of the testing was to confirm a flow sheet that consisted of gravity, flotation and cyanidation treatments, and to provide data for design purposes.

The flow sheet consisted of gravity treating the primary grinding mill discharge to produce a high-grade gravity concentrate suitable for on-site smelting. The rougher gravity tailings were reground and treated to produce a cleaner float concentrate, which was combined with the cleaner gravity tailings. The combined products were then reground and cyanide leached to recover the contained precious metals.

Batch laboratory tests were used to determine optimum conditions for the gravity, flotation, and cyanidation unit operations. These conditions were then used in the operation of three mini-pilot plants investigating these three major operations. In addition, an 18 inch diameter SAG mill grind test was performed, with the results reported in a separate document.

According to Ronning and Prenn (2004), in their summary Hazen stated that "*Results confirm those obtained from previous test programs at Hazen and demonstrate an excellent response of the Lincoln gold ore to the proposed flow sheet*." Table 1 of the Hazen report showed an overall recovery of 94.6% for gold and 55.1% for silver. In their review of the 1989 Hazen testing, Behre Dolbear (2007) noted that the calculated head grade of the composite core sample was 0.245oz Au/ton and that the combined gravity and flotation recoveries averaged 98.9%; an average of 40.5% of the gold was recovered from the gravity concentrates using amalgamation (Table 13.1).

Table 13.1 Grind-size Based Gravity and Flotation Results from 1989 Hazen Research Testing
(Behre Dolbear, 2007)

Test 1929-	Grind		Amalgamation1	% Gold Distribution				Analyses, opt Au			
	Mesh			% Pass	Gravity Conc.	Flotation Conc.	Comb. Conc.	Gravity Conc.	Flotation Conc.	Flotation Feed	Flotation Tailings
59	48	72	28.4	28.6	69.8		98.4	705.3	1.77	0.161	0.004
60	65	80	50.9	50.9	48.5		99.4	3,218.1	1.089	0.143	0.002
61	100	80	54.1	54.4	45.1		99.5	1,220.4	1.177	0.166	0.002
62	150	80	40.5	42.2	57.5		99.7	677.0	0.942	0.142	0.001
63	200	80	22.3	22.7	73.8		96.5	2,741.7	0.933	0.134	0.007
64	270	73	46.8	47.1	52.6		99.7	1,786.2	0.846	0.134	0.001
Avg.			**40.5**	**41.0**	**57.9**		**98.9**	**1,724.8**	**1.126**	**0.147**	**0.003**
1Amalgamation of gravity concentrate (used to determine relative free gold quantity)											

Other work by Hazen in 1989 included determination of one rod mill and three ball mill Bond Work Indexes. The rod mill BWI was 11.4, while the ball mill indexes were in the range 12.4 to 12.9.

13.3 Metallurgical Studies by Interpro, 1991

Interpro conducted a series of gravity separation and froth flotation tests on about 144 pounds of mineralized material from the Lincoln mine. Their purpose was to recover the metallic fraction containing both gold and arsenic. The principal objective was to remove arsenic from the tailings, in order to produce tailings that could pass California standards (CAMWET) for underground tailings disposal.

Interpro found that leachates from separation tests contain less than 6 mg/l of arsenic. According to Interpro, "… *research indicates that the standard likely to be imposed would allow disposal of tailings material that produce less than 15 mg/l of arsenic in the CAMWET test*."

Overall gold recovery obtained by Interpro was more than 70%, and Interpro suggested that better recovery might be possible using a finer grind than they had. They recommended further testing using a finer grind.

13.4 Meridian Gold Royal Mountain King Industrial Process Plant Test, 1991

In September 1991, about 8,119 tons of material from the Sutter Gold Venture were milled at the Royal Mountain King mine, a mine controlled at the time by Meridian. This could have been a good test of the milling characteristics of the Sutter Gold material, but interpretation of the results was difficult due to an inability to accurately determine the original average grade of the material processed. Table 13.2, extracted from Table 1 of Hazlitt and Russell (1992), illustrates the problem.

Table 13.2 Comparison of Estimates from 1991 Mill Test Run
(Ronning and Prenn, 2004, taken from Hazlitt and Russell, 1992)

Item	Estimate 1	Estimate 2	Estimate 3	Estimate 4
Milled Tons	8,119	8,119	8,119	8,119
Gold Average Grade oz Au/ton	0.182	0.244	0.242	0.268
Number of Samples	72	72	63	952
Calculated Contained Ounces	1,479	1,983	1,962	2,176
Settlement in Ounces	1,404			
Recovery Based on 1404 ounces	94.9%	70.8%	71.6%	64.5%
Theoretical Ounces at 95% recovery	1,404	1,884	1,863	2,067
Estimate 1: Meridian data based on Royal Mountain King lab and mill, September 1991				
Estimate 2: Sutter Gold data based on Shasta Lab and Using Sept. 12 Royal Mountain King lab assays				
Estimate 3: Sutter Gold data based on Shasta Lab December 1991				
Estimate 4: Sutter Gold truck samples				

13.5 Brown and Root Braun Review of Metallurgical Test Results, 1992

In 1992, Wayne Henderson of Brown and Root Braun reviewed all of the metallurgical test reports done to that date and provided a summary of results, conclusions and recommendations for each one. An abridged summary of the Henderson (1992) report follows in Table 13.3.

Table 13.3 Summary of Metallurgical Test Results to 1992
(Ronning and Prenn, 2004)

Tester(s), Date:	Hazen Research Inc., September 1989
Description of Test	Prepare bulk composite from 762 pounds of broken core rejects from 21 drill hole cores representing 35 mineralized intervals. Prepare sample splits for composite analysis
Results, Conclusions, Recommendations	0.228 oz Au/ton predicted from individual core analyses 0.273 oz Au/ton weighted average of screen fractions Weighted averages from screen fractions should be more reliable estimate of gold heads than single assay sample.
Tester(s), Date:	Hazen Research Inc., September 1989
Description of Test	Prepare bulk composite from 130 pounds of material from 21 drill hole cores representing 35 mineralized intervals. Prepare sample splits for composite analysis
Results, Conclusions, Recommendations	0.240 oz Au/ton predicted from individual core analyses; 0.250 oz Au/ton weighted average of screen fractions 2,090 ppm arsenic;1.50% total sulfur, 1.38% as sulfide, 0.05% sulfate; 4.73% iron; 0.2 ppm mercury; other trace metals essentially nil
Tester(s), Date:	Hazen Research Inc., September 1989
Description of Test	Determine specific gravities of cores from 21 drill holes used for sample preparation
Results, Conclusions, Recommendations	S.G. ranged from 2.41 to 3.51; Weighted Mean Specific Gravity: 2.827; Arithmetic Mean Specific Gravity: 2.828
Tester(s), Date:	Hazen Research Inc., September 1989
Description of Test	Determine bond work indices for bulk core and reject composite samples
Results, Conclusions, Recommendations	Bond work indices range from 11.4 kWhrs/ton to 12.9 kWhrs/ton, and are in range of typical quartzitic ore
Tester(s), Date:	Hazen Research Inc., September 1989
Description of Test	Determine effects of grind size on gravity and flotation (batch tests). Determine free gold in concentrate by amalgamation. 21 drill hole cores representing 35 mineralized intervals
Results, Conclusions, Recommendations	Total recoveries ranged from 98.4% for a nominal grind of –48 to 99.7% for a nominal grind of –270 mesh.. Average of 98.8% of the gravity concentrate was “free or amalgamatable” gold.
Tester(s), Date:	Hazen Research Inc., September 1989
Description of Test	Demonstrate gravity recovery on commercial-scale gravity equipment. 21 drill hole cores, rejects from SAG mill tests
Results, Conclusions, Recommendations	A gravity-only recovery circuit could be designed to recover up to 85% of the gold values in Lincoln Mine ore.
Tester(s), Date:	Hazen Research Inc., September 1989
Description of Test	Demonstrate commercial type of flotation circuit using bulk composite from 21 drill holes representing 35 mineralized intervals
Results, Conclusions, Recommendations	The expected recovery range would be in the order to 93% to 95% of gold in commercial operation.
Tester(s), Date:	Hazen Research Inc., September 1989

Description of Test	Demonstrate extraction of gold from combined gravity tailings and flotation concentrate, using cyanidation
Results, Conclusions, Recommendations	To get acceptably high leach extractions, some regrind to minus 40 microns and a high-intensity cyanidation process is recommended. The removal of ultra-high grade free gold as an induction meltable concentrate prior to leaching will increase overall gold recovery.
Tester(s), Date:	Interpro Inc., December 1991
Description of Test	Define gold, sulfur and arsenic distributions by screen size fraction in head sample used for gravity and tails flotation tests. This material was crushed, not ground.
Results, Conclusions, Recommendations	The gold distribution followed that of the weight fraction in the screened intervals, except that some coarse gold was probably lost. Sulfur and arsenic more closely followed the weight fractions in each screen size interval.
Tester(s), Date:	Interpro Inc., December 1991
Description of Test	Define free gold fraction in gravity test feed screen size fractions
Results, Conclusions, Recommendations	Calculated head by weighted screen fractions analysis using amalgamation for free gold and fire assay of amalgamation tails produced a total gold content closer to that obtained from the bulk sample.
Tester(s), Date:	Interpro Inc., December 1991
Description of Test	Determine the applicability of flash flotation to recover gold from the coarse size fractions.
Results, Conclusions, Recommendations	Interpro concluded that flash flotation was not applicable to the coarser Lincoln material, but Henderson argued that Interpro's test apparatus did not work correctly.
Tester(s), Date:	Interpro Inc., December 1991
Description of Test	Determine the distribution of gold, sulfur and arsenic in gravity and flotation concentrates and tailings
Results, Conclusions, Recommendations	Flotation of the finer size fractions in combination with gravity beneficiation of the coarser size fractions provides the most efficient gold, arsenic and sulfide recovery into the concentrates and minimizes these components in the tailings.
Tester(s), Date:	Interpro Inc., December 1991
Description of Test	Determine the arsenic leachability from gravity and flotation tailings
Results, Conclusions, Recommendations	Interpro found that leachates from separation tests contain less than 6 mg/l of arsenic. According to Interpro, "… research indicates that the standard likely to be imposed would allow disposal of tailings material that produce less than 15 mg/l of arsenic in the CAMWET test."
Tester(s), Date:	Knight Piesold and Co. September 1989
Description of Test	Provide bleeding rate and settled dry density of tailings slurry deposited sub-aqueously. Provide estimate of water recovery as vertical seepage and as run-off. Used two tailings samples from hazen gravity and flotation tests.
Results, Conclusions, Recommendations	The slurries were suitable for rotational deposition
Tester(s), Date:	J.H. Bailey and Associates, April 1992
Description of Test	No test work; presented a critical project overview and proposed a gravity circuit flow sheet
Results, Conclusions, Recommendations	The proposed flow sheet was in accordance with the results of prior test work programs.

Tester(s), Date:	Pocock Industrial Inc. 1989
Description of Test	Determine the gravity sedimentation and vacuum filtration characteristics of rougher flotation tailings, reground gravity and flotation concentrates and CIL tailings, consistent with the use of a dewatered tailings deposition method.
Results, Conclusions, Recommendations	Partial de-watering of the tailings prior to disposal will work.
Tester(s), Date:	Sutter Gold Venture, October 1991
Description of Test	Lincoln Ore Mill Test at Royal Mountain King Mine
Results, Conclusions, Recommendations	see Section 13.4 and Table 13.2
Tester(s), Date:	Hazen Research Inc., January 1990
Description of Test	Determine if atmospheric, oxygen-assisted cyanidation presents any advantage for the Lincoln concentrate leaching
Results, Conclusions, Recommendations	Hazen concluded that there was little advantage for oxygenated cyanidation and that oxygenated pre-aeration did not have an advantage. Henderson disagreed and concluded that better, more extensive testing should be done.
Tester(s), Date:	CH_2M Hill, August 1989, October 1990
Description of Test	Determine the arsenic leachability of a mineralized mine muck sample and rougher flotation tailings
Results, Conclusions, Recommendations	If the permissible leachability limit is 15.0 mg/l, neither the mine muck nor the flotation tailings leach to a potentially environmentally hazardous level. Henderson recommended more extensive testing of gravity, CN, and other tailings samples..
Tester(s), Date:	Hazen Research Inc., August 1991
Description of Test	Determine the amount of recoverable gold by a combination of grinding, gravity concentration, sizing and retreatment of middling and slime fractions. Sample was 2,660 pounds of broken core rejects from previous testing.
Results, Conclusions, Recommendations	A concentrate recovering 83.5% of the gold can be obtained from a rougher and cleaner table gravity circuit. An efficient fine gold recovery system would be warranted.

13.6 Metallurgical Studies by Dawson Metallurgical Laboratories Inc., 1996

Dawson Metallurgical Laboratories Inc. ("Dawson") tested gravity concentration followed by flotation of the gravity tailings, using a 39kg sample of mineralized material crushed to minus ¼in, supplied by SGM. One of the objectives was to obtain a quantity of flotation concentrate to be tested by others. Table 13.4 gives the head analysis results.

The gravity concentrate obtained from two runs, each using 15kg of material crushed to minus 35 mesh, contained 1% gold, recovering about 30% of the gold contained in the original sample. Dawson suggested that a higher gold recovery might be obtained by grinding the mineralized material to a finer size.

Flotation of the gravity tailings produced a concentrate containing an additional 34% of the gold, for a combined gravity and flotation recovery of 64% of the gold. Behre Dolbear (2007) noted that this test represented the lowest gold recovery of any of the other bench-scale work and suggested the results are probably best explained by an inadequate reagent suite. This was higher grade than most of the samples tested by other laboratories (Behre Dolbear, 2007).

Table 13.4 Head Analysis Results from Dawson Metallurgical Laboratories
(Behre Dolbear, 2007)

Sample	Head	Au opt	% S	% As	% Fe
P-2378	Assay	0.588[1]	0.36	0.22	4.65
P-2378A	Assay	0.491[2]	0.38	0.22	4.57
P-2378	Calculated[3]	0.782	0.31	0.21	3.42

[1]Average of 4 assays ranging from 0.541 to 0.648 opt Au[2]Average of 2 assays ranging from 0.444 to 0.539 opt Au[3]Back calculated from the 30 gram test

Over 35% of the gold was lost to the flotation tailings, which Dawson attributed in part to gold encapsulation and significant oxidation of the sulfide mineralized material. Some loss was also attributed to non-floating gold.

13.7 Metallurgical Studies by Kappes, Cassiday & Associates, 1997

Behre Dolbear (2007) described testing by Kappes, Cassiday & Associates ("KCA"), who conducted pilot-scale continuous flotation tests on 479 pounds of gravity tailings (Table 13.5). The sample's head analysis was approximately 0.174oz Au/ton and 0.130oz Ag/ton. The pilot plant grind was about 82% passing 65 mesh. Over 86% of the gold contained in the head sample was recovered into the floatation concentrate.

Table 13.5 Kappes, Cassiday & Associates 1997 Pilot Flotation Test on Gravity Tailings
(As reported by Behre Dolbear, 2007)

KCA Sample No.	Weight Pounds	Weight %	Product Description	Grade opt Au	Grade opt Ag	Distribution % Au	Distribution % Ag
25450 A	9.436	1.98	Concentrate	6.591	1.67	86.3	40.0
25450 D	1.024	0.22	Cleaner[1]	1.814	0.46	2.7	1.2
25450 C	4.665	0.98	Rougher[2]	0.126	0.04	0.8	0.4
25450 B	432.767	91.02	Tailings	0.016	0.05	9.6	54.9
25450 F	27.558	5.80	Tailings[3]	0.016[4]	0.05[4]	0.6	3.5
Total	**475.45**	**100.00**				**100.0**	**100.0**
Calc. Head				0.151	0.08		
25450 E	4.189		Mill				

[1]Cleaner Cell Clean-out
[2]Rougher Cell Clean-out
[3]Tailings Clean-out
[4]Not assayed individually but given the grade of the bulk tailings

Behre Dolbear (2007) further reported that KCA conducted sodium cyanide leaching for 48 hours on four 500g samples of flotation concentrates. Reagent consumption ranged from 14.6 to 29.26lb/ton of cyanide at initial concentrations of 5g/l NaCN and 10g/l NaCN, respectively. Lime consumption was 6.0lb/ton. Recovery of gold from the concentrate was over 99%, although no residue assays were published.

13.8 Historic Specific Gravity Determination Testing

Ronning and Prenn of MDA (2004) reported that specific gravity tests were completed by Hazen for mineralized intervals in the Lincoln zone from 21 drill holes. These tests indicated an average specific gravity of 2.83 or a tonnage factor of 11.3 ft^3/ton. The test result of an average specific gravity of 2.83 is higher than the expected value for quartz of about 2.6. They suggested that additional specific gravity tests be undertaken.

Behre Dolbear (2007) reported that PAH used a tonnage factor of 12.5ft^3/ton for their resource estimate.

According to Behre Dolbear (2007), Payne and Grunewald used different factors varying from 13.0 to 11.2ft^3/ton, depending on the rock type, vein type, and mineralization. These factors were based on unpublished measured rock densities from projects elsewhere in northern California. According to Behre Dolbear (2007), to check these factors, SGM had specific gravity measurements performed on 10 core samples from the Keystone zone, one core sample from the 2006-2007 drilling, and one underground sample from the Comet deposit, and these measurements confirmed the adopted tonnage factors with one exception. The factor for replacement-type gold mineralization was changed from 11.2 to 11.0. Behre Dolbear (2007) reported that SGM has determined that 11.8ft^3/ton is an appropriate tonnage factor to use as an average for the entire Lincoln-Comet deposit. Behre Dolbear (2007) reported that the following tonnage factors, in cubic feet per ton, were used for Payne's August 2007 resource estimates:

- 12.0 – Vein quartz (greater than 50% vein quartz)
- 10.8 – Phyllonite (high-strain zones)
- 11.8 – Quartz stringer zones (10-50% vein quartz)
- 11.8 – Tensional veinlet arrays (10-50% vein quartz)
- 11.2 – Metavolcanic and metavolcaniclastic rocks (mafic to intermediate)
- 11.0 – Replacement-type gold mineralization (gray mineralization)
- 11.2 – Carbonate-altered metavolcanic and metavolcaniclastic rocks
- 13.0 – Metasedimentary rocks (slates, greywacke, etc.)
- 15.0 – Fault gouge and mineralized rubble.

Behre-Dolbear (2007) reported that the specific gravity of the "ore-grade" material was determined to be 2.82, but they did not provide details on who made that determination or any further details.

13.9 Recent Metallurgical Testing Commissioned by Sutter Gold Mining Inc.

McClelland Laboratories, Inc. ("McClelland") in Sparks, Nevada conducted a series of gravity/flotation tests to determine the optimum processing conditions for gold recovery of SGM's "ore-grade" samples (McPartland, 2009). Rougher tailings generated from a bulk sample at McClelland were sent to Golder Paste Technology and utilized for paste backfill testing (Golder Paste Technology Ltd., 2009). The results from McClelland's testing further served as a basis by which H. C. Osborne & Associates (2009) created a preliminary mill processing design.

A major issue identified by H. C. Osborne & Associates (2009), based on the metallurgical testing done on the project since 1983, is the extreme difficulty in obtaining a head grade analysis. No matter how carefully done, two splits of the same sample showed a variance of ± 25% in assays. The same report noted that flotation concentrates from the Lincoln Mine project contain high levels of arsenic, up to 5%, which may preclude sending them to a smelter or roaster unless the receiving plant can handle high arsenic tails.

13.10 Gold Gravity Recovery and Concentration Testing

The following information in Subsection 13.10 is taken from the May 2009 report from McClelland (McPartland, 2009).

McClelland conducted scoping-level gravity/flotation tests on a bulk "ore-grade" sample to confirm expected optimum processing conditions for recovery of gold from Sutter Gold ore. SGM collected a bulk sample of the Lincoln-Comet mineralized zones in 2008-2009. The sample was collected from SGM's development rock stockpile located at the mine site and consisted of six 55gal. steel drums – two identified as run of mine and four with sample crushed to a nominal 1/2in size.

McClelland also conducted batch and locked-cycle gravity/flotation tests on a drill core composite, using essentially the same optimized processing conditions to confirm the metallurgical response of a higher-grade drill-core sample and to determine the effects of scavenger concentrate recycle and cleaner tailings recycle during flotation. The sample consisted of three 5gal. plastic buckets of half split or sawn drill core that were reported to contain 25 drill core interval samples from holes DDH-0163 (1 interval), DDH-0164 (13 intervals), and DDH-0165 (11 intervals). The samples were stored in a freezer until testing to minimize sulfide mineral oxidation.

Head samples of both bulk "ore-grade" and drill core composite were submitted to American Assay for a metallic screen gold assay. Results are shown in Table 13.6.

Table 13.6 Head Assay Results and Head Grade Comparisons of Bulk "Ore Grade" and Core Composite Samples
(McPartland, 2009)

Determination Method	Head Grade, oz Au/ton ore	
	Bulk Ore	Core Composite
Metallic Screen Assay #1	0.199	0.220
Metallic Screen Assay #2	0.125	0.354
Metallic Screen Assay #3	0.098	
Metallic Screen Assay #4	0.148	
Metallic Screen Assay #5	0.173	
Metallic Screen Assay #6	0.195	
Calculated, Gravity/Flotation	0.240	0.513
Calculated, Gravity/Flotation	0.186	0.574
Average	0.171	0.415
Standard Deviation	0.045	0.160

McClelland reported that head grade standard deviations for the two samples were higher than normally expected, probably due to the presence of significant quantities of free-milling, particulate gold. Results from gravity concentration testing confirmed the presence of free-milling particulate gold in both samples. McClelland cautioned that care should be taken when evaluating grind size sensitivity data generated from the current testing program, because of the relatively large sample grade variation encountered during testing (Table 13.6). In particular, head grades for the bulk "ore grade" sample 100M and 150M gravity/flotation tests were 0.240 and 0.186oz Au/ton, respectively. Gravity/flotation gold recoveries can be expected to be sensitive to gold grade.

13.11 Bulk Sample Confirmatory Gravity and Flotation Testing

A series of two milling/gravity/flotation tests were conducted on the bulk "ore-grade" sample at feed sizes of 80%-100M and 150M.

Preliminary batch gravity/flotation testing showed that the Sutter Gold bulk "ore-grade" sample responded very well to milling/gravity/flotation treatment at 80%-100M and 150M feed sizes. Combined gold recovery obtained by grinding, whole ore gravity rougher/cleaner concentration and bulk sulfide flotation of the recombined gravity (cleaner and rougher) tailings was 97.9% at both grind sizes. The combined (gravity cleaner and flotation rougher) concentrate weight pull corresponding to that recovery was equivalent to 2.1% of the feed (whole ore) weight. The corresponding combined concentrate grades were equivalent to between 8.8 and 11.4oz Au/ton. Gravity cleaner concentrate grades were 72.9oz Au/ton (100M) and 49.7oz Au/ton (150M). Flotation cleaner concentrate grades produced from the gravity tailings were 4.0oz Au/ton (100M) and 5.3oz Au/ton (150M). Final tailings grades were 0.005oz Au/ton (100M) and 0.004oz Au/ton (150M). McClelland cautioned that the grade variations described in Table 13.6 may result in variations in recovery and that this may account, at least in part, for the higher gravity recovery from the 100M feed.

A cleaner flotation test was conducted on the bulk rougher concentrate produced while generating the flotation rougher tailings for paste backfill testing. Results from that test indicated that cleaner flotation was not particularly effective in significantly increasing the rougher concentrate grade. A recleaner concentrate grade of 5.0oz Au/ton was generated by cleaner flotation treatment of a flotation rougher concentrate with a grade of 2.7oz Au/ton. The concentrate upgrading came at the expense of significant loss of gold recovery (~42% of gold in the rougher concentrate).

13.12 Drill Core Composite Sample Gravity and Flotation Testing

A set of duplicate batch milling/gravity/flotation tests were conducted on the drill core composite at an 80%-150M feed size to confirm results obtained from the full-ore sample testing and to generate baseline data for comparison to locked-cycle flotation testing.

Results from duplicate batch milling/gravity/flotation tests conducted on a Sutter Gold drill core composite showed that the core composite responded very well to the optimized processing conditions at an 80%-150M feed size. Combined gold recovery obtained by grinding, whole ore gravity rougher/cleaner concentration and bulk sulfide flotation of the recombined gravity (cleaner and rougher) tailings averaged 99%. The combined (gravity cleaner and flotation rougher) concentrate weight pull corresponding to that recovery was equivalent to 3.2% (Avg.) of the feed (whole ore) weight. The corresponding combined concentrate grade was equivalent to 16.9oz Au/ton (Avg.). Gravity and flotation average cleaner concentrate grades were 1,030 and 5.0oz Au/ton, respectively. Final tailings grades were 0.002 to 0.003oz Au/ton.

Microscopic examination of the gravity cleaner concentrates showed free gold particles up to about 10M in size. McClelland noted that the presence of gold particles this coarse shows the importance of including gravity concentration in the circuit because gold particles that coarse are unlikely to be recovered by flotation.

Locked-cycle flotation testing conducted on gravity recombined tailings generated from the Sutter Gold drill core composite showed no significant detrimental effect to either scavenger concentrate recycle, or cleaner tailings recycle, to the rougher flotation feed. Final tail grades for both locked-cycle test series ranged from 0.002 to 0.004oz Au/ton, which were similar to those obtained from the batch tests. Recycle of a flotation scavenger concentrate to the rougher flotation feed appeared to be effective in significantly increasing the flotation rougher concentrate grade, and decreasing the flotation rougher concentrate pull weight. Recycle of a flotation cleaner tailings to the rougher flotation feed did not significantly improve cleaner flotation grades.

McClelland (McPartland, 2009) concluded that:

- *"The Sutter Gold ore samples responded very well to whole ore milling/gravity/concentration treatment, followed by bulk sulfide flotation treatment of the resulting gravity tailings, at 80%-100M and 150M feed sizes*
- *It should be possible, using this processing scheme, to produce a small volume, high grade gravity cleaner concentrate suitable for smelting, and a larger volume, lower grade flotation concentrate suitable for offsite shipment and processing.*
- *Expected combined gold recoveries by milling/ gravity/flotation treatment should be quite high.*
- *Recycle of a flotation scavenger concentrate to the rougher flotation feed may be effective in significantly increasing the flotation rougher concentrate grade, without increasing losses to flotation rougher tailings.*

- *Recycle of a flotation cleaner tailings (without regrind) to the rougher flotation feed may not be effective in significantly increasing the flotation cleaner concentrate grade.*"

13.13 Paste Backfill Testing

Bulk quantities of flotation tailings were used for backfill testing at Golder Paste Technology to be used as part of their conceptual study to assess the technical and economic viability of producing paste backfill at the Lincoln Mine project. Paste properties of the SGM tailings were determined to be appropriate for a paste backfill system (Golder Paste Technology Ltd., 2009). Paste costs significantly more than hydraulic sand backfill.

13.14 Summary

SGM provided approximately 23 metallurgical testing, analysis, processing, piloting and engineering reports and studies to Allihies Engineering Incorporated of Butte, Montana and Dr. Corby G Anderson, QP CEng FIMMM FIChemE, for a review. This task was undertaken in detail and resulted in an affirmation that previous work has been of a sufficient quality and quantity necessary to support a Preliminary Economic Assessment Technical Report in accordance with NI 43-101. Summary results of metallurgical testing are presented in Table 13.7.

Table 13.7 Metallurgical Property Summary for the Lincoln-Comet Resource Material Testing

	Source: Hazen, 1989		Source: McClelland, 2009 (McPartland, 2009)	
Criteria	Rod Mill	Ball Mill		
Bond Work Index	11.4	12.4-12.9		
Grind Size			P80-100 mesh	
Head Grade (oz Au/ton)			0.24	Calculated from concentrate and tails analysis due to head-grade sampling issue
Gravity Recovery (% Au)			82.1	Centrifugal concentrator followed by hand panning (due to sample size)
Gravity Concentrate Grade (oz Au/ton)			72.87	
Flotation Recovery (% Au)			15.8	Combined cleaner concentrate and tails
Flotation Concentrate Grade (oz Au/ton)			3.97	
Total Recovery (% Au)			97.9	
Concentration Ratio			1:48	Calculated as 2.1% of feed weight recovered

Further, after a review of the proposed gold mill design by Paul E. Danio & Associates, and a corroboration of this document with existing metallurgical studies to date, Allihies confirms that these proposed designs and economic estimates are now appropriate as a preliminary conceptual design and preliminary estimate based on the current data available. Allihies does not confirm or take responsibility for, or confirm, any past, current or future operations and any detailed designs. The mill designed by Danio and Associates was constructed on site during 2012.

14.0 MINERAL RESOURCE ESTIMATE

Section 14.2 describes MDA's 2011 mineral resource estimate for the Lincoln-Comet deposit (Tietz, et al, 2011) while Section 14.3 describes the current mineral resource estimate for the Keystone deposit.

MDA's project preview included an evaluation of the Payne (2008) project-wide Indicated and Inferred Resources for possible inclusion within the current resource tabulation. See Section 14.3.8 for a more detailed discussion of the Payne (2008) resource estimate.

No mineral reserves have been identified on the project and mineral resources that are not mineral reserves do not have demonstrated economic viability.

14.1 Introduction

Although MDA is not an expert with respect to any of the following aspects of the project, MDA is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Lincoln-Comet or Keystone mineral resources as of the date of this report.

The current Lincoln-Comet resource does abut, and is limited by, the existing property boundary in the Comet area of the deposit. Any subsequent changes to SGM's property position in this area, whether through acquisition or further boundary definition, could result in a material change to the current resource estimate.

MDA classifies resources in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories to be in compliance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (May 10, 2014) and therefore Canadian National Instrument 43-101. CIM mineral resource definitions are given below, with CIM's explanatory material shown in italics:

> **Mineral Resource**
>
> *Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.*
>
> A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.
>
> The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
>
> *Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.*

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Modifying Factors

> Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

MDA reports resources at cutoffs that are reasonable for deposits of this nature given anticipated mining methods and plant processing costs, while also considering economic conditions, because of the regulatory requirements that a resource exists "in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction ."

14.2 Lincoln-Comet Resource

The modeling and estimation of gold resources were done under the supervision of Paul Tietz and Steven Ristorcelli, qualified persons with respect to mineral resource estimation under NI 43-101. Mr. Tietz and Mr. Ristorcelli are independent of SGM by the definitions and criteria set forth in NI 43-101; there is no affiliation between either Mr. Tietz or Mr. Ristorcelli and SGM except that of an independent consultant/client relationship.

The drill and underground assay data used in the resource estimate have an effective date of September 2, 2009. The collar location database used in the resource has an effective date of February 15, 2010. The initial resource model and estimate, based on the February 15, 2010 data, were completed in May 2010, with a revised model and estimate completed December 14, 2010. The December model revision was required due to MDA's receipt of a revised land map indicating a minor change in the SGM-controlled property position. The resource reported in this section reflects the revised model and estimate reported in the 2011 PEA.

As discussed in Section 10.0, since completion of the Lincoln-Comet estimate, and up to the current report date, additional holes have been drilled at Lincoln-Comet. MDA reviewed all 26 of the surface holes completed in 2012, along with the majority of underground drilling and sampling, and concludes that this drilling substantially supports the 2011 estimate. Though the drilling and underground development did locally extend and expand the high-grade gold zones, this work did not change the resource in a material way. For this reason, the Lincoln-Comet resource estimate described in this section is still current.

14.2.1 Lincoln-Comet Resource Model Assay Database

The drill data used for the Lincoln-Comet resource estimate contains 753 underground channel samples, 87 surface and 107 underground core drill holes, and two RC drill holes. The resource database contains gold values for 7,343 sample intervals. All of the core and underground assay data were used in MDA's current resource estimate; the RC assay data were not used due to verification and sample-precision concerns.

The database also contains 563 arsenic analyses, all of which are from the SGM drilling.

In preparation for the 2011 resource estimate, MDA conducted a detailed audit of the drill-hole and underground sample data, which included reconstructing much of the assay database. The details of

MDA's audit, and the subsequent changes to the original SGM database, are provided in Section 12.2.1. As described in Section 12.2.1, the gold values used in the resource estimate are an average value of often multiple sample check re-assays. Due to the "nugget" character of the mineralization and apparent sub-sampling variability, the average value is considered to be more representative of the sample interval than the single initial assay value. The descriptive statistics on this resource assay database are provided in Table 14.1. The "0.000"oz Au/ton values noted in the table represent original "less than detection" assays converted and standardized in the database to a zero gold value.

The underground channel samples account for approximately 10% of the gold assays within the resource database. The significantly higher mean and median values for the underground samples, as compared to the drill-hole data, reflect the concentrated location of underground sampling along the major veins within the high-grade center of the deposit. Although there are some concerns over sample reliability, the underground sample data provide significant spatial and grade control within the deposit and are deemed appropriate for use in estimating and classifying the current resource.

The project coordinates are truncated California State Plane – Zone 2 coordinates using the NAD 27 datum.

Table 14.1 Lincoln-Comet Resource Assay Database

	# Samples	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Northing	7,330					69799	73096	ft
Easting	7,330					40330	42465	ft
Elevation	7,330					220	1489	ft
Sample length	7,330	3.0	3.8	1.5	0.4	0.3	12.5	ft
All Gold data	7,330	0.002	0.072	0.425	5.899	0.000	17.018	oz Au/ton
UG sample length	753	2.5	3.8	2.0	0.5	0.3	12.0	ft
UG gold data	753	0.095	0.454	1.164	2.565	0.000	17.018	oz Au/ton
DH sample length	6,577	3.0	3.8	1.4	0.4	0.4	12.5	ft
DH gold data	6,577	0.000	0.033	0.214	6.391	0.000	12.075	oz Au/ton

14.2.2 Geologic Background

Gold mineralization within the Lincoln Mine project is characterized by sheared, quartz-ankerite veins, containing free gold and 1-2% accessory sulfides, hosted within greenstone metavolcanic rocks. The veins are emplaced within through-going structural zones considered to be extensional structures related to the dominant, district-wide, northwest-trending faults which bound the eastern and western sides of the metavolcanic rocks.

The gold-quartz veins branch and anastomose along the 3,000ft length of the Lincoln-Comet resource, with the strongest gold mineralization often localized within distinct dilation zones along the veins or at structural/vein intersections. Horizontal continuity of Lincoln-Comet geology and grade is greater than vertical continuity, which is in contrast to most of the historic deposits within the district, though there is no information below about 800ft from the surface; deeper exploration could show different relative continuity of the mineralized veins.

The primary gold-bearing veins of the Comet and Lincoln zones trend N30°W and generally dip steeply west at an average of 70 degrees. Within the higher-grade portions of the Lincoln and Comet zones, the west-dipping veins often terminate against shallow, east-dipping fault/vein structures which serve as structural traps for mineralization. Gold-bearing veins are generally between 1ft and 4ft in thickness, though a composite thickness of up to 40ft results from the intersection of four veins in the upper portions of the Lincoln zone.

A pervasive fault overprint is common, and gold mineralization appears to coincide with the late faulting. Mineralized quartz veins are often bounded by fault slip planes that generally define one or both vein walls, and sheared, ribboned vein-quartz exceeding 1ft in width is commonly associated with the higher-grade mineralization. Where fault shearing has removed the quartz veining, gold mineralization occurs within narrow (<2in) shear planes that can be traced within the main structures. If the fault overprint is missing, or the fault trends out of the quartz in to the wallrock, gold grades within the quartz vein are often weak and zones of barren quartz can occur. Where the faulting occurs on one quartz vein wall, gold grades will weaken away from the fault plane.

The gold has a strong nugget character, being highly erratic in grade both on a sample scale and along strike within the individual veins. Gold grades of >1oz Au/ton can quickly transition to <0.1oz Au/ton over just a few feet along strike, while duplicate underground sampling has shown consistent assay differences of over 100%.

14.2.3 Density

There are a total of 33 density measurements on various lithologies from within the Lincoln Mine project area, though only two samples are from within the Lincoln-Comet resource area and have known locations. Ten of the density samples are from core holes in the Keystone area, which are outside of the current Lincoln-Comet resource area, while 21 density measurements are from unspecified Lincoln-area drill core presumably within the current resource model. The latter density samples were submitted in 1989 for metallurgical testing, and there is no record of the specific drill hole IDs or footage intervals for these 21 samples. Previous authors have combined these data with density data from other Mother Lode deposits in determining the density value(s) to be used in their respective studies. Section 13.8 provides a summary of the limited historic density testing and the various density values used within previous technical reports.

Some previous authors have used, or have recommended, single deposit-wide, density values ranging from 11.3 ft^3/ton to 12.5 ft^3/ton. Conversely other authors have sub-divided the data and used different density factors varying from 10.8 to 15.0ft^3/ton, depending on the rock type, vein type, and mineralization. Though the data are very limited, the more quartz-dominant samples usually associated with higher-grade mineralization have density values of around 11.5ft^3/ton.

MDA is using just one density value (12.0 ft^3/ton) within the Lincoln-Comet resource due to the scarcity of data and the difficulty in correctly estimating density within highly variable mineralized structures. This value might be somewhat high but will lend some conservatism to the resource estimate.

There has been no additional density testing conducted in preparation for the current resource estimate, and MDA believes that additional density testing is warranted. The testing should be based on a statistically valid number of samples from within the major rock units, especially within those hosting significant mineralization. The samples selected should also provide sufficient spatial coverage to adequately characterize the full deposit.

14.2.4 Geology Model - Structures and Veins

A geologic/structural cross-sectional model of the Lincoln-Comet deposit was created by Mark Payne and Bill Mitchell in 2009. The model is based on 57 cross sections spaced 50ft apart along a N30°W axis. The cross sections are oriented perpendicular to the general strike of the deposit.

All significant structures and associated quartz veins were modeled, resulting in a total of 38 unique mineralized veins within the Lincoln-Comet resource area. Many of the veins have a limited strike and/or dip extent, but some veins, such as veins 42, 50, and 51, extend for much of the full length of the resource area. The veins and surrounding structural zones pinch and swell, with the smaller veins often occurring as branches off the main veins. The veins are dominantly steeply west-dipping, though there are some veins (e.g., the 37 vein) with shallow west dips. In general, vein widths range from 1 to 4ft, though vein thickness often increases to a maximum of 20ft at the top of the west-dipping veins where they intersect, and often terminate, at east-dipping vein structures. Examples of shallow east-dipping veins are veins 20, 23, and 61 in the Comet zone and vein 9 in the Lincoln zone.

Although gold mineralization occurs within all 38 modeled veins, the majority of the mineralization is hosted within five veins; these are the through-going 40, 42, and 50 veins along with veins 6 and 43, which are localized within the Lincoln and Comet zones, respectively. Mineralization along all of the individual veins is highly variable and significant portions of many of the veins are low-grade and likely sub-economic.

The geologic model also included the metavolcanic/slate fault contacts along the eastern and western sides of the metavolcanic graben. The great majority of the Lincoln and Comet veins occur within the metavolcanic rocks, and the west-dipping structures all terminate at depth against the West fault contact. As modeled by SGM, some portions of the upper vein elevations of vein 42 and the down-dip extension of the east-dipping vein 9 extend through the eastern graben fault into the slate.

14.2.5 Gold Mineral Domain Model

MDA used the geologic cross-sectional model as a base and guide for the gold mineral model. The underground workings were also plotted on cross-section to guide the gold model. Quantile plots of gold were made to help define the natural populations of metal grades to be modeled on the cross sections. The quantile plots, along with additional statistical analyses, indicated that the gold mineralization can be modeled using three mineral domains. The low-grade gold domain (domain 100) is characterized by a range of grades of ~0.01oz Au/ton to ~0.07oz Au/ton and generally represents mineralization associated with weak veining and/or shearing either in the wallrock outside the primary vein or within the structures at depth or along strike away from the center of the deposit. The mid-grade gold domain (domain 200) is characterized by a range of grades of ~0.07oz Au/ton to ~0.25oz Au/ton and generally represents gold mineralization associated with increased shearing and/or sporadic coarse gold deposition within or along the immediate boundaries of the mineralized veins. The high-grade gold domain (domain 300) is defined by grades generally exceeding ~0.25oz Au/ton that are associated with increased shearing and coarse gold deposition within the high-grade core of the mineralized veins.

Color-coded assays corresponding to population breaks indicated by the quantile plots, along with the geologic cross-sectional interpretation, were plotted on cross sections and were used in the creation of the gold mineral domains. After discussions with SGM, MDA considered each vein a unique entity for sample coding and estimation purposes, so unique mineral domains were created for each vein. For example, the low-grade domain within vein 42 was coded as domain 142; the low-grade domain in vein 50 was coded as domain 150. In the same way, the high-grade domain in vein 42 is coded as domain 342. The mineral domains as modeled and drawn on the cross sections are not strict "grade shells" but are created using geologic information for defining orientation, geometry, continuity, and contacts in conjunction with the grades. Typical cross sections of the geology and gold domains for the Lincoln and Comet zones are shown in Figure 14.1 and Figure 14.2, respectively. Note the location of the SGM property boundary (shown as a vertical blue line) on the Comet zone cross-section (Figure 14.2)

Using the cross-sectional interpretations as a framework, level plans of the gold domains were created at a 10ft-spacing. The 10ft-spaced level plans were 3-D rectified to fit the drill and underground sample data, and Surpac mining software was used to code domain percentages into the block model.

The 3-D solid of the underground workings was used to code the blocks with a partial percentage volume of open space ("void"). Where a percentage of a block is coded as void, and one or more gold domains are present, the domain(s) percentage in the block is reduced by the volume percent of void. The highest grade domain percentage is initially reduced first followed by the next lowest gold domain present in the block. The assumption is that previous development followed, and removed, the highest grade portions of the vein. If the block is 100% void, then all mineralization is removed from the block.

Figure 14.1 Gold Domain Model for Lincoln Zone Section 950 North

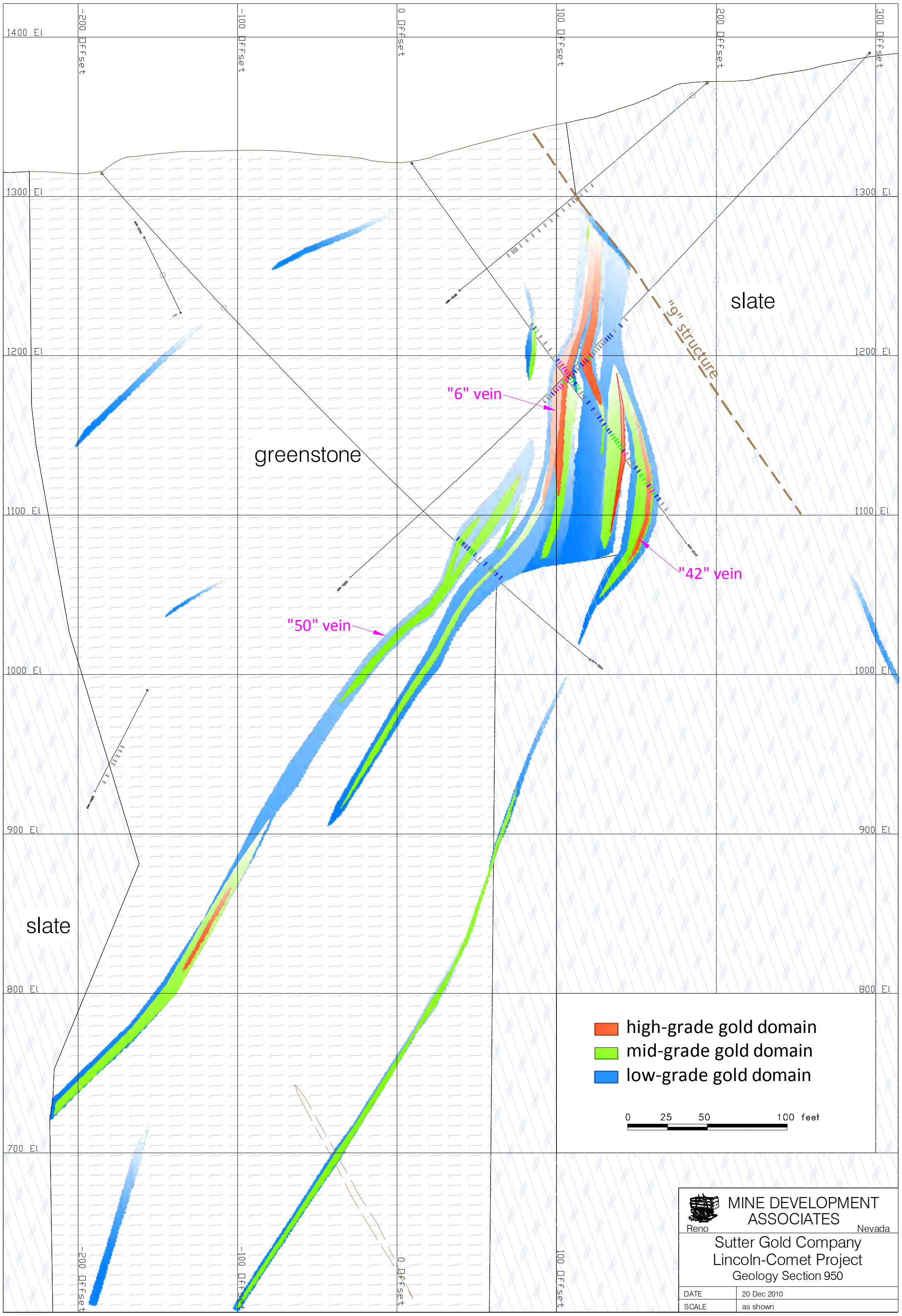
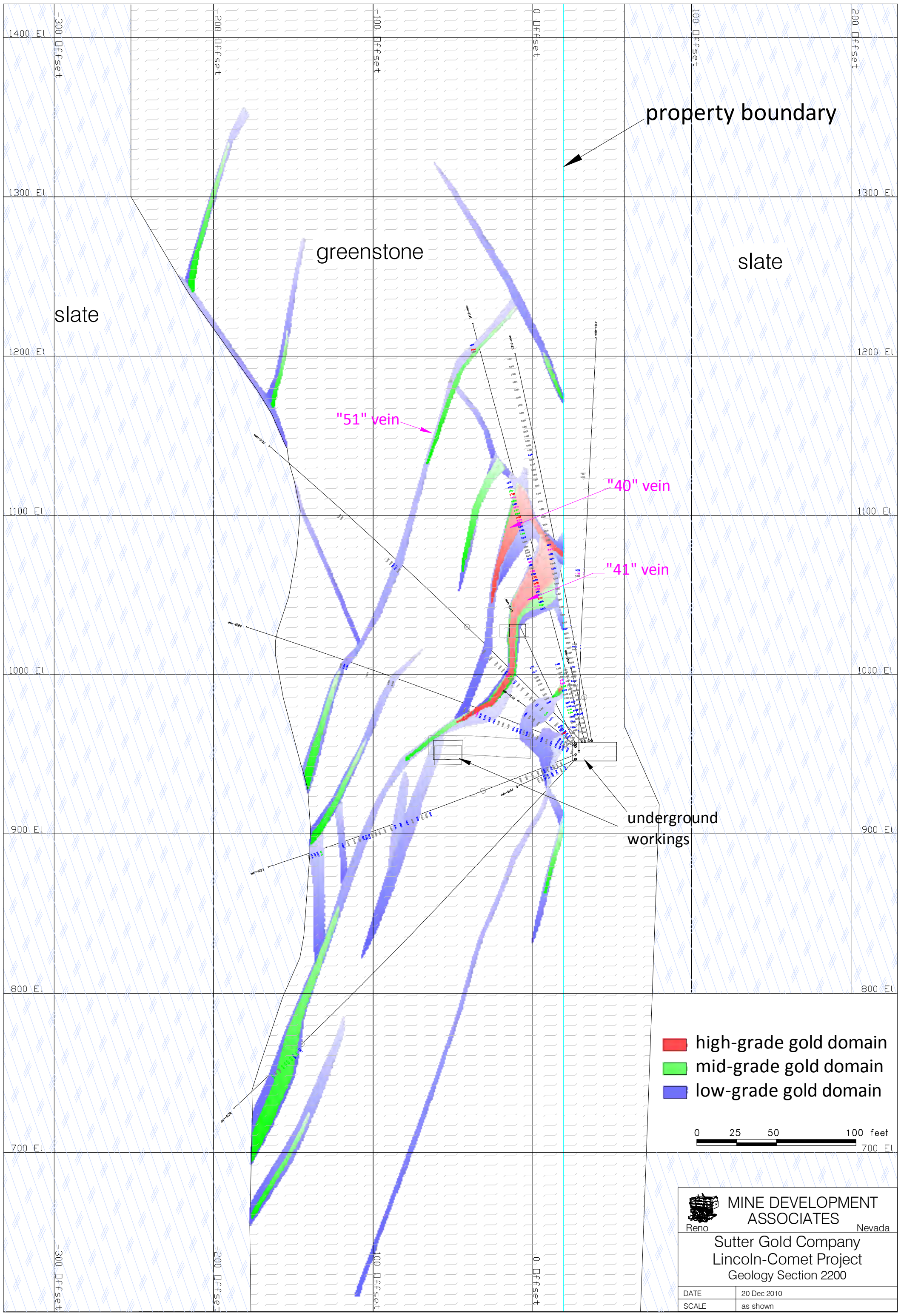


Figure 14.2 Gold Domain Model for Comet Zone Section 2200 North



14.2.6 Sample Coding and Capping

The cross-sectional gold mineral domains were used to code the drill assays and many of the underground samples. Due to the close spacing of the underground samples and projection issues arising from cross-sectional coding, it was necessary to manually code many of the underground samples to their respective domains.

Table 14.2 presents the assay descriptive statistics for the combined veins within each of the low-, mid-, and high-grade domains: domains 100, 200 and 300, respectively. Examples of the assay statistics for the mid- and high-grade domains within individual veins are presented in Table 14.3. The veins shown in Table 14.3 are the more significant veins (numbered 6, 40, 42, 43, and 50) within the Lincoln-Comet resource, each containing greater than 10,000 ounces combined Indicated and Inferred gold resource.

Table 14.2 Assay Descriptive Sample Statistics by Gold Mineral Domains – All Veins

Gold Domain 100 to 161 capping to 0.250 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	1,649	2.5	3.1	1.2	0.383	0.3	8.5	ft
Au	1,649	0.021	0.028	0.031	1.117	0.000	0.520	oz Au/ton
Au_cap	1,649	0.021	0.028	0.027	0.985	0.000	0.250	oz Au/ton
Au_dmn	1,649					100	161	

Gold Domain 200 to 261 variable capping to 2.500 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	817	2.0	3.1	1.5	0.497	0.3	12.0	ft
Au	817	0.112	0.145	0.264	1.821	0.000	12.075	oz Au/ton
Au_cap	817	0.112	0.135	0.122	0.901	0.000	2.500	oz Au/ton
Au_dmn	817					200	261	

Gold Domain 300 to 351 variable capping to 6.000 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	441	2.0	3.3	1.7	0.507	0.3	9.0	ft
Au	441	0.529	1.054	1.569	1.489	0.000	17.018	oz Au/ton
Au_cap	441	0.529	1.009	1.319	1.307	0.000	11.19*	oz Au/ton
Au_dmn	441					302	351	

* uncapped value in vein 40

Table 14.3 Assay Descriptive Sample Statistics for Mid- and High-Grade Gold Mineral Domains – Significant Veins Only

Gold Domain 206 — No Cap

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	20	2.5	2.8	0.9	0.311	1.3	4.5	ft
Au	20	0.149	0.178	0.105	0.588	0.024	0.441	oz Au/ton
Au_cap	20	0.149	0.178	0.105	0.588	0.024	0.441	oz Au/ton

Gold Domain 306 — capping to 3.000 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	14	2.0	2.1	0.2	0.104	2.0	2.5	ft
Au	14	1.185	1.254	1.223	0.975	0.179	5.061	oz Au/ton
Au_cap	14	1.185	1.114	0.820	0.736	0.179	3.000	oz Au/ton

Gold Domain 240 — No Cap

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	104	2.0	2.7	1.3	0.467	0.3	7.0	ft
Au	104	0.094	0.103	0.065	0.629	0.007	0.395	oz Au/ton
Au_cap	104	0.094	0.103	0.065	0.629	0.007	0.395	oz Au/ton

Gold Domain 340 — No Cap

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	180	2.2	3.5	1.7	0.483	0.5	8.0	ft
Au	180	0.610	1.243	1.681	1.352	0.022	11.190	oz Au/ton
Au_cap	180	0.610	1.243	1.681	1.352	0.022	11.190	oz Au/ton

Gold Domain 242 — capping to 0.500 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	199	2.5	3.8	2.1	0.557	0.5	12.0	ft
Au	199	0.110	0.131	0.139	1.058	0.000	1.900	oz Au/ton
Au_cap	199	0.110	0.124	0.084	0.673	0.000	0.500	oz Au/ton

Gold Domain 342 — capping to 5.000 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	102	3.0	4.0	1.9	0.470	1.0	9.0	ft
Au	102	0.470	0.920	1.689	1.837	0.055	17.018	oz Au/ton
Au_cap	102	0.470	0.810	0.794	0.980	0.055	5.000	oz Au/ton

Gold Domain 243 — capping to 0.500 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	60	2.5	3.0	1.0	0.330	0.8	5.0	ft
Au	60	0.116	0.125	0.096	0.767	0.000	0.600	oz Au/ton
Au_cap	60	0.116	0.124	0.091	0.730	0.000	0.500	oz Au/ton

Gold Domain 343 — capping to 6.000 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	45	2.0	2.5	0.9	0.373	0.5	4.0	ft
Au	45	0.651	1.246	1.803	1.447	0.073	9.007	oz Au/ton
Au_cap	45	0.651	1.180	1.555	1.318	0.073	6.000	oz Au/ton

Gold Domain 250 — No Cap

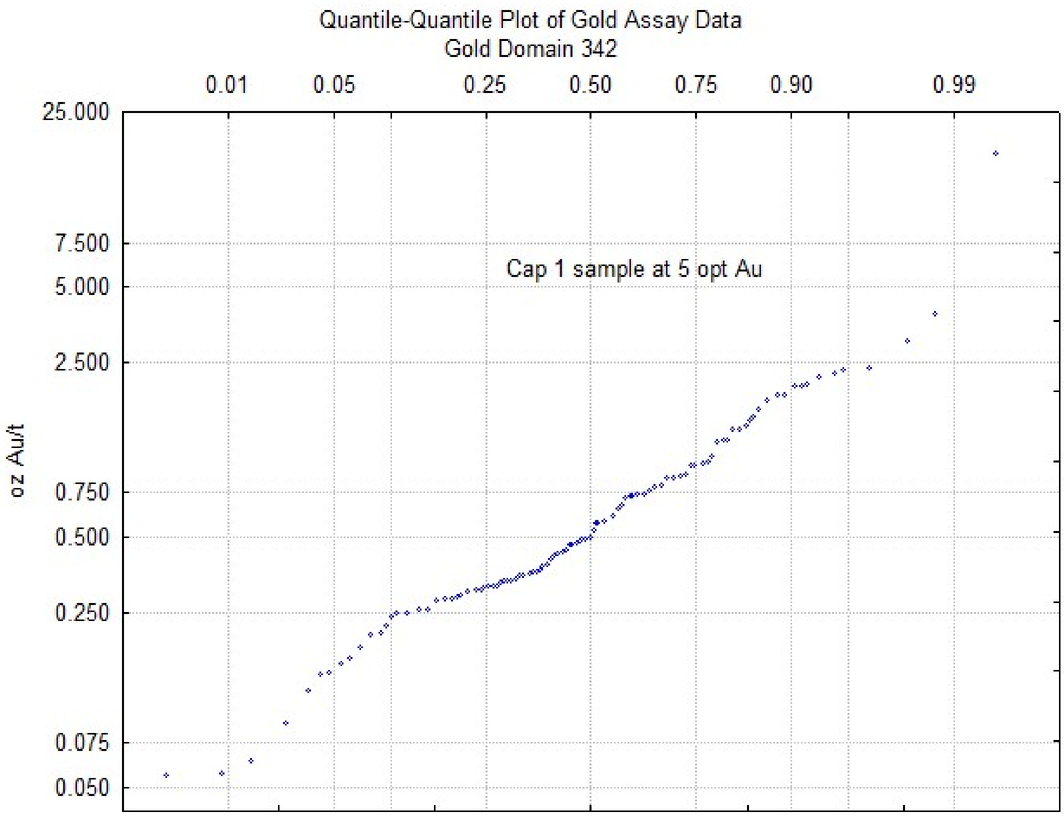
	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	85	2.0	2.6	0.9	0.334	0.7	5.6	ft
Au	85	0.111	0.126	0.091	0.722	0.000	0.973	oz Au/ton
Au_cap	85	0.111	0.126	0.091	0.722	0.000	0.973	oz Au/ton

Gold Domain 350 — capping to 1.200 oz Au/ton

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	40	2.0	2.5	0.7	0.272	1.0	4.5	ft
Au	40	0.344	0.471	0.358	0.760	0.044	1.864	oz Au/ton
Au_cap	40	0.344	0.457	0.313	0.685	0.044	1.200	oz Au/ton

Included in the statistical analyses of the assay data are cumulative probability assay population plots for the combined domains, as well as the individual plots for the major veins. These plots are used to identify anomalous higher-grade samples which fall outside the general assay population and which are potentially subject to grade capping. As an example of one of the plots, the vein 42 mid-grade domain, is shown in Figure 14.3. All of the probability plots are included in Appendix A.

Figure 14.3 Cumulative Probability Plot for Vein 42 Mid-grade Domain



After reviewing all of the statistical data, the anomalous samples that were considered candidates for capping were viewed onscreen to determine their spatial relationship within their respective domains. Specifically the samples were evaluated in relationship to the adjacent samples sharing the same domain coding. They were also checked as to whether the existing domain modeling has already volumetrically restricted the influence of the high-grade sample alleviating the need for further capping.

After completing the statistical and model analyses of the anomalous samples, MDA decided to cap 20 samples. All of the low-grade domains were evaluated together and a total of six samples were capped at 0.25oz Au/ton. The mid- and high-grade domains were evaluated on an individual basis, and a total of 14 samples were capped in these domains. Capping levels within the mid- and high-grade domains range from 0.4oz Au/ton in domain 226 (vein 26; mid-grade domain) to 6.0oz Au/ton in domain 343 (vein 43; high-grade domain). The resulting assay database used in the estimate has a maximum gold value of 11.19oz Au/ton. This high value is spatially associated with fourteen other underground samples having gold grades greater than 5.0oz Au/ton that occur within a localized mineralized horizon in vein 40. None of these assays were capped. The effect of the grade capping on the population statistics for each of the domains is shown in the "Au_cap" rows in Table 14.2 and Table 14.3. Grade capping produces an average 7 percent and 4 percent decrease in mean grade in the mid- and high-grade domains, respectively (Table 14.2). A comparison of the gold grade determined for the cross-sectional polygonal model using both uncapped and capped samples indicates a 6 percent decrease in average deposit gold grade as a result of grade capping.

14.2.7 Compositing

Once the individual gold samples were capped, they were down-hole composited into maximum 5ft composites honoring all mineral domain contacts. Table 14.4 presents the composite descriptive statistics for the low-, mid-, and high-grade domains within all veins. The composite statistics for the mid- and high-grade domains for the significant veins are presented in Table 14.5. No minimum length restrictions were imposed on the composites, and length-weighted composites were used in the estimation. The narrow width of many of the domains results in composites that are on average approximately 3ft in length for the mid- and high-grade domains (as indicated in Table 14.4). The complete statistics for each of the individual gold domains is included in Appendix B.

Table 14.4 Composite Descriptive Statistics by Gold Mineral Domains – All Veins

Low-Grade Composites (weighted - domain 100s)

Item	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	1238	4.00	3.62	1.48	0.41	0.10	5.00	ft
Au_Grade	1238	0.022	0.028	0.023	0.818	0	0.25	oz Au/ton
Au_dmn	1238					100	161	

Mid-Grade Composites (weighted - domain 200s)

Item	Count	Median	Mean	Std. Dev.	CV	Min.	Max.	Units
Length	648	3.00	3.23	1.47	0.46	0.30	5.00	ft
Au_Grade	648	0.116	0.136	0.117	0.86	0	2.5	oz Au/ton
Au_dmn						200	261	

High-Grade Composites (weighted - domain 300s)

Item	Count	Median	Mean	Std. Dev.	CV	Min.	Max.	Units
Length	390	3.00	2.95	1.46	0.49	0.20	5.00	ft
Au_Grade	390	0.61	1.01	1.208	1.196	0	11.19	oz Au/ton
Au_dmn						302	351	

Table 14.5 Composite Descriptive Sample Statistics for Mid- and High-Grade Gold Mineral Domains – Significant Veins Only

Composites Domain 206

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	15	3.43	3.70	1.28	0.37	1.30	5.00	ft
Au_Grade	15	0.178	0.142	0.081	0.454	0.075	0.355	oz Au/ton

Composites Domain 306

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	8	3.69	4.00	1.36	0.37	2.00	5.00	ft
Au_Grade	8	1.114	1.036	0.521	0.467	0.318	1.723	oz Au/ton

Composites Domain 240

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	88	2.71	2.50	1.25	0.46	0.30	5.00	ft
Au_Grade	88	0.103	0.096	0.056	0.537	0.01	0.239	oz Au/ton

Composites Domain 340

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	169	2.95	2.80	1.44	0.49	0.50	5.00	ft
Au_Grade	169	1.243	0.803	1.499	1.206	0.022	11.19	oz Au/ton

Composites Domain 242

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	172	3.43	3.60	1.51	0.44	0.50	5.00	ft
Au_Grade	172	0.124	0.112	0.071	0.568	0.004	0.467	oz Au/ton

Composites Domain 342

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	99	3.33	3.50	1.53	0.46	0.20	5.00	ft
Au_Grade	99	0.81	0.573	0.736	0.908	0.12	5	oz Au/ton

Composites Domain 243

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	53	3.03	2.50	1.44	0.48	0.80	5.00	ft
Au_Grade	53	0.124	0.115	0.083	0.671	0.002	0.5	oz Au/ton

Composites Domain 343

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	44	2.09	1.90	1.12	0.53	0.50	5.00	ft
Au_Grade	44	1.18	0.602	1.555	1.318	0.073	6	oz Au/ton

Composites Domain 250

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	61	3.24	3.00	1.33	0.41	1.00	5.00	ft
Au_Grade	61	0.127	0.111	0.061	0.485	0	0.451	oz Au/ton

Composites Domain 350

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	30	3.18	3.00	1.22	0.38	1.00	5.00	ft
Au_Grade	30	0.471	0.31	0.328	0.696	0.18	1.864	oz Au/ton

14.2.8 Resource Model and Estimation

The model used 10ft by 10ft by 1ft-wide blocks with the long dimensions oriented N30°W and vertical. The block dimensions were chosen to minimize dilution for underground mining of a deposit of this kind.

Following compositing and the previously described statistical analyses of those composites, correlograms were constructed in multiple directions for all domains together as well as for many of the major veins. Due to the relatively small number of samples, the individual vein correlograms returned poor, not readily understandable, results. The correlograms for the combined mid- and high-grade domains indicated a maximum distance of grade continuity, both along strike and down-dip, of 50ft with almost all variability within the first 10ft. The estimation criteria were, in part, defined by these correlograms and, in part, by attempting to honor understood geologic controls and distributions. Those estimation parameters are given in Table 14.6. All gold domains have the same estimation parameters, which include a 50ft first pass, a second 250ft pass, and a final pass that filled the respective domains. All searches were isotropic, though the individual vein domains spatially controlled the estimation, which resulted in very planar search ellipses oriented along the general strike and dip of the veins. Estimation within each mineral domain used only those composites coded to that respective domain. Inverse-distance estimation was chosen as the base case, while estimates were also made by nearest neighbor and Kriging. The latter two were used as checks on the given estimate.

Table 14.6 Estimation Parameters

Description	Parameter
All Gold Domains	
Inverse distance power	2
Rotation/Dip/Tilt (all variograms and searches)	150º / 0º / 70º
First Pass Samples: minimum/maximum/maximum per hole*	3 / 15 / 3
First Pass Search (ft): major/semimajor/minor	50 / isotropic*
Second Pass Samples: minimum/maximum/maximum per hole*	2 / 15 / 3
Second Pass Search (ft): major/semimajor/minor	250 / isotropic*
Third Pass Samples: minimum/maximum/maximum per hole*	1 / 15 / 3
Third Pass Search (ft): major/semimajor/minor	Fill domain / isotropic*

*All underground samples considered as one drill hole so maximum of three underground samples per estimation pass.

14.2.9 Lincoln-Comet Mineral Resources

Resource classification used distance to the nearest sample, number of samples, geologic confidence, and mineral domain continuity. While the estimation included the low-grade domains, MDA did not include the low-grade (<0.07oz Au/ton) estimated blocks in the reported resource or the low-grade composite data in the classification criteria due to the erratic and likely sub-economic nature of the mineralization.

The criteria for resource classification are given in Table 14.7. The samples used for the classification criteria stated below are independent of the modeled domains. There are only Indicated and Inferred resources within the Lincoln-Comet deposit. There are no Measured resources associated with the

Lincoln-Comet deposit due to a) a scarcity of density measurements, b) significant mineral variability leading to uncertainty in grade estimation, and c) some spatial uncertainty in the geologic model. None of these issues deters from the overall confidence in the global project resource, but they do detract from confidence in some of the accuracy which MDA believes is required for Measured. Indicated resources are spatially associated with underground development and/or tight-spaced drill information. All mineralized material not classified as Indicated is Inferred.

Table 14.7 Criteria for Lincoln-Comet Resource Classification

Measured	
There is no Measured material within the Lincoln-Comet resource	
Indicated	
Minimum no. of samples /minimum no. of holes / maximum distance (ft)	3 / 1 / 50
and	
a 2nd hole must be within 75ft	
or	
Resource block is within "Indicated" solid that surrounds underground development	
Inferred	
All material not classified above but lying within the modeled mineralized domains is Inferred	

Because of the requirement that the resource exists "in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction," MDA is reporting the resource at a 0.12oz Au/ton cutoff gold grade that is reasonable for deposits of this nature and for the expected mining conditions and methods. MDA considered metal prices, recovery, and economics to derive the reported cutoff. Although preliminary in nature, MDA believes there is sufficient information to make a reasonable estimate of a projected economic cutoff that should not be materially different, under similar economic situations, after obtaining more information.

A tabulation by classification and cutoffs of the Lincoln-Comet resource is presented in Table 14.8, while the reported resource summary is in Table 14.9. The stated resource is undiluted and is based on 10ft by 10ft by 1ft-wide blocks. The block dimensions were chosen to provide spatial definition and minimize dilution for underground mining a deposit of this kind. The undiluted resource includes just the mid- and high-grade domain-coded blocks. Figure 14.4 and Figure 14.5 present cross-section examples of the gold block models for the Lincoln and Comet zones, respectively.

No mineral reserves have been identified on the project and mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.8 Lincoln-Comet Gold Resources

Indicated Material				Inferred Material			
Au Cutoff (oz Au/ton)	**Tons**	**Grade** (oz Au/ton)	**oz Au**	**Au Cutoff** (oz Au/ton)	**Tons**	**Grade** (oz Au/ton)	**oz Au**
0.07	209,000	0.318	67,000	0.07	714,000	0.209	149,000
0.1	179,000	0.358	64,000	0.1	619,000	0.227	141,000
0.11	168,000	0.374	63,000	0.11	559,000	0.24	134,000
0.12	**152,000**	**0.401**	**61,000**	**0.12**	**506,000**	**0.254**	**128,000**
0.13	135,000	0.436	59,000	0.13	449,000	0.27	121,000
0.14	119,000	0.478	57,000	0.14	388,000	0.291	113,000
0.15	107,000	0.513	55,000	0.15	322,000	0.321	104,000
0.16	98,000	0.547	54,000	0.16	264,000	0.358	95,000
0.17	91,000	0.578	52,000	0.17	227,000	0.39	88,000
0.18	85,000	0.607	51,000	0.18	206,000	0.412	85,000
0.19	79,000	0.638	50,000	0.19	183,000	0.44	81,000
0.2	74,000	0.666	49,000	0.2	166,000	0.465	77,000
0.22	68,000	0.71	48,000	0.22	143,000	0.508	72,000
0.24	63,000	0.743	47,000	0.24	131,000	0.533	70,000
0.26	60,000	0.769	46,000	0.26	117,000	0.566	66,000
0.28	57,000	0.796	45,000	0.28	110,000	0.584	64,000
0.3	54,000	0.827	45,000	0.3	104,000	0.602	63,000
0.4	40,000	0.987	40,000	0.4	72,000	0.719	51,000
0.5	33,000	1.1	37,000	0.5	53,000	0.811	43,000

Table 14.9 Lincoln-Comet Reported Gold Resources

Lincoln-Comet Reported Resource				
Classification	**Au Cutoff** (oz Au/ton)	**Tons**	**Grade** (oz Au/ton)	**oz Au**
Indicated	0.12	152,000	0.401	61,000
Inferred	0.12	506,000	0.254	128,000

Figure 14.4 Gold Block Model for Lincoln Zone Section 950 North

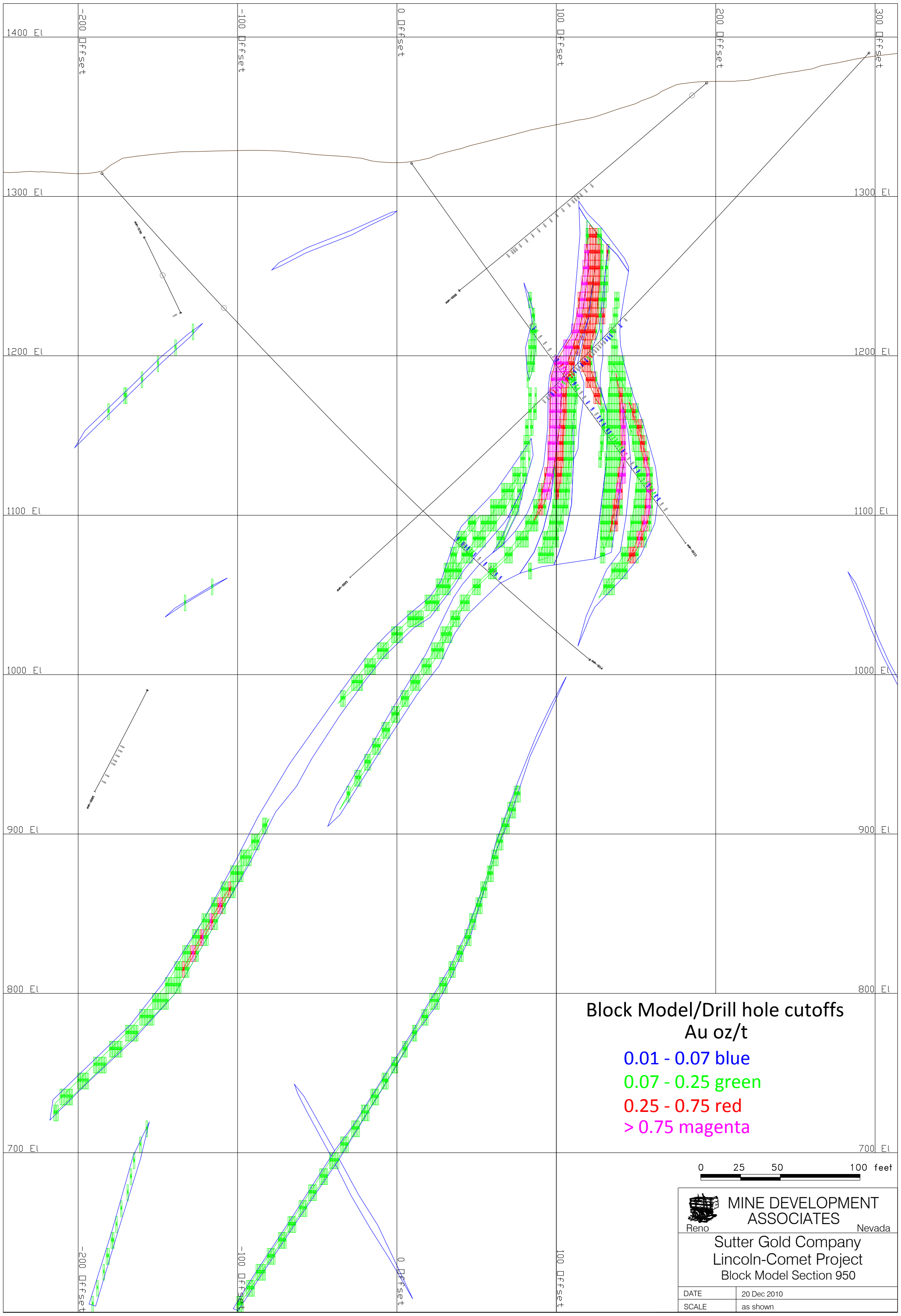
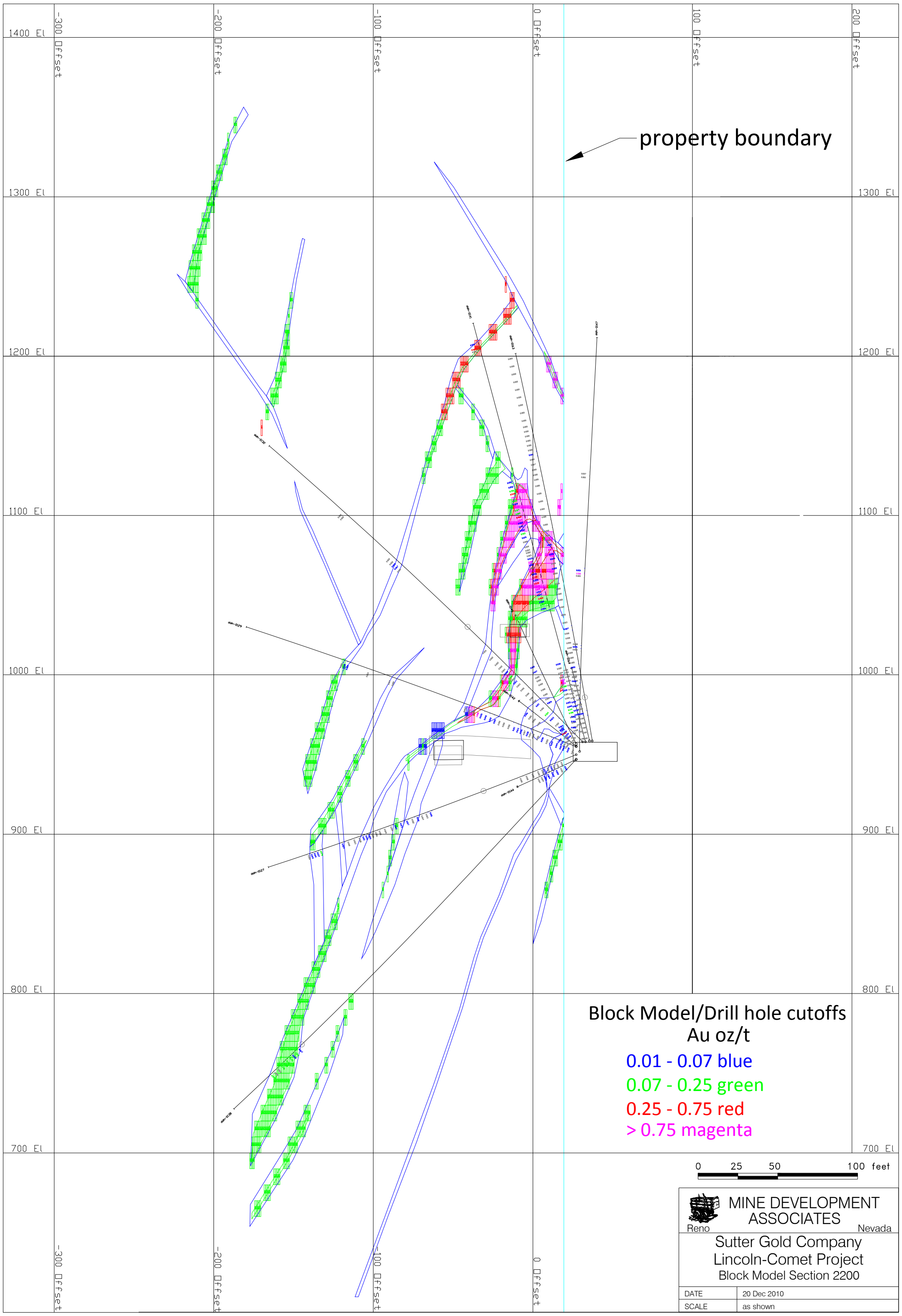


Figure 14.5 Gold Block Model for Comet Zone Section 2200 North



14.2.10 Lincoln-Comet Resource Model Validation

Various visual checks made on the Lincoln-Comet resource model included:

- Cross sections of the block model with the mineral domains, drill-hole assays and geology, topography, sample coding, and block grades with classification were plotted and reviewed for reasonableness; and

- Block-model information, such as coding, number of samples, and classification were checked on the computer by domain and lithology.

The resource model was further validated by conducting a number of statistical and analytical comparisons of the block volumes and grades against assay and composite data, cross-sectional and level plan polygonal volumes, and polygonal, nearest-neighbor and Kriging estimation models.

14.2.10.1 Cross-section and Block Model Volumes

The block model volumes were compared against the initial cross-sectional polygons and then against the 10ft-spaced level plans (Table 14.10). There is only a minor volume decrease from cross-section to block model with no appreciable change in volume between the level plans and the block model.

Table 14.10 Volume Comparisons

	Volume (cuft)			diff. (%)	
Domain	Cross-Section	Level Plan	Block Model	Section-BM	Plan-BM
200	10,071,495	9,957,965	9,957,820	-1.13%	0.00%
300	1,909,530	1,863,211	1,863,148	-2.43%	0.00%
Total:	11,981,025	11,821,175	11,820,968	-1.34%	0.00%

14.2.10.2 Grade Comparison between Polygonal Model and Block Model

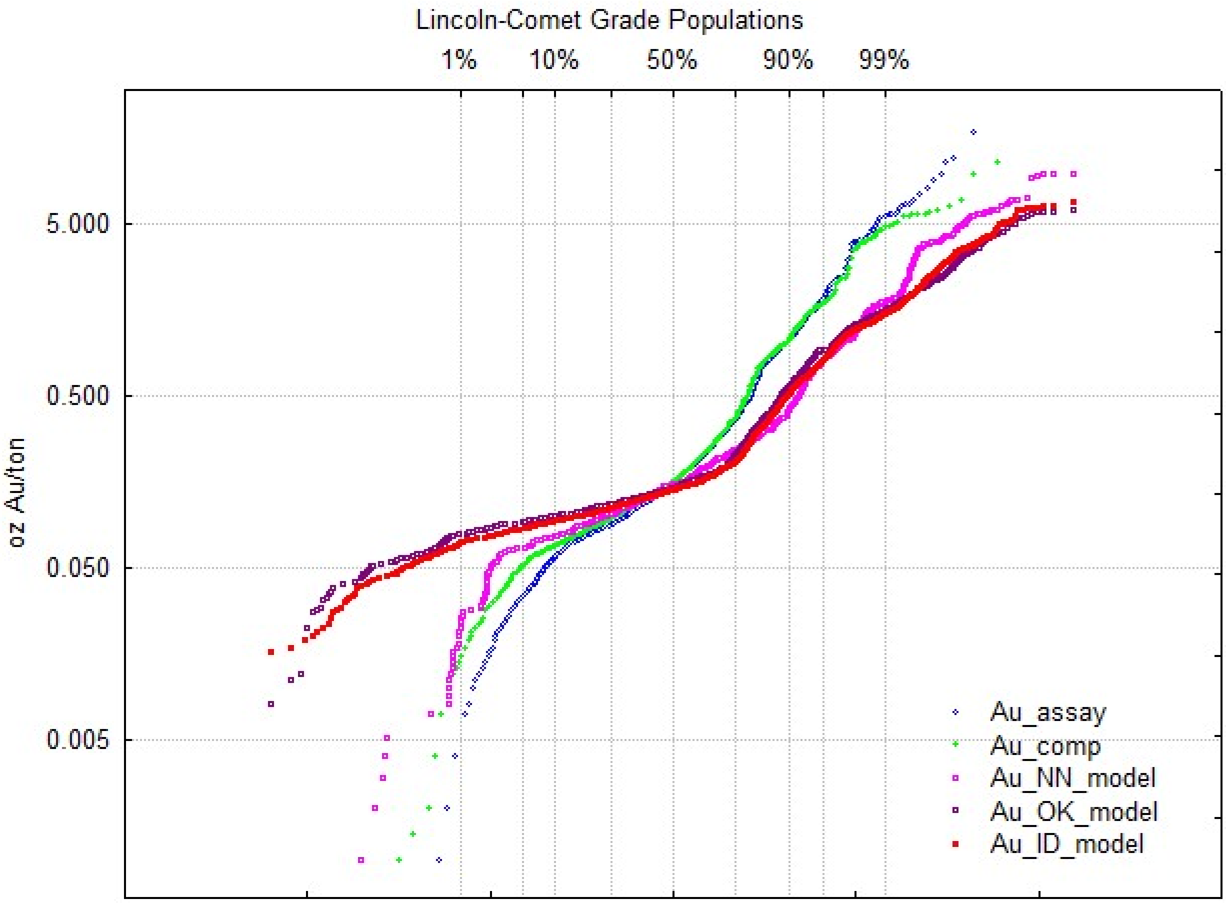
The block model average grade was compared against a polygonal model gold grade, the latter created using the original modeled cross-section domains. To best simulate the polygonal model, all blocks coded to a mineral domain were included and no cutoff grade was used to limit the block tons. The resulting block model average grade is 0.24oz Au/ton, while the polygonal model average grade is 0.256oz Au/ton.

14.2.10.3 Assay-Composite-Estimation Comparison

Nearest-neighbor and Kriging estimation models were run as a check on the Inverse Distance estimation. The gold-grade populations for these estimates, along with the grade populations for the capped assays and composites, were graphically compared as shown in Figure 14.6. The graph is a log normal probability plot showing the gold-grade value in the vertical axis and the grade population distributions in the horizontal axis. As indicated on the graph, the population of composite and assay values have a close correlation, with the expected divergence at the extreme low- and high-grade portions of the populations. Similarly, the grade populations of the three model estimates have a close correlation, with also the expected divergence at the extreme grade ranges. All populations cross at the

50-percent mark, indicating that no appreciable grade shift or bias has been introduced during the compositing or model estimation procedures.

Figure 14.6 Grade Population Comparisons



After reviewing all of the validation results, it is deemed that the resource estimate is reasonable, honors the geology, and is supported by the geologic model**.**

14.2.11 Lincoln-Comet Discussion, Qualifications, Risk, Upside, and Recommendations

For the Lincoln-Comet vein-gold deposit, the most important characteristics that impact the resource estimate are the strongly anastomosing nature of the narrow, gold-bearing veins and the significant mineral variability within the veins. The total gold resource of 189,000 Indicated and Inferred gold ounces is contained in over 30 distinct veins, many of which branch off the main through-going vein structures.

Typical of other Mother Lode district high-grade gold systems, there is significant mineral variability within the deposit on a sub-sample scale to a mining scale. This "nugget" character results in uncertainty in grade estimation both in the resource model and also in mine planning and reconciliation. Away from the underground development, there is spatial uncertainty in the geologic model due to the more widely spaced drill data and the highly variable, branching nature of the vein system.

There are no Measured resources within the Lincoln-Comet deposit, and just 30% of the resource is classified as Indicated. The lack of Measured and limited amount of Indicated material are a result of the spatial and grade estimation uncertainties inherent within the high-grade, coarse-gold deposit. Additional drilling within the currently defined deposit, especially away from the underground development, could materially change the existing resource with the discovery of either localized, high-grade mineralization within known veins or new veins branching off the main structures. There is potential for expanding the Inferred resource by drilling both down-dip and along strike to the northwest and southeast. Increasing the Indicated resource, though, would likely entail further underground development and tightly-spaced drilling.

An issue that affects the resource model and estimate is that the property boundary impinges on the resource along the northeast boundary of the Comet zone. The existing resource reflects the current property boundary; any changes to this boundary would have a material effect on the resource model and estimate.

14.3 Keystone Resources

The modeling and estimation of Keystone gold resources were done under the supervision of Paul Tietz, a qualified person with respect to mineral resource estimation under NI 43-101. Mr. Tietz is independent of SGM by the definitions and criteria set forth in NI 43-101; there is no affiliation between Mr. Tietz and SGM except that of an independent consultant/client relationship.

The drill assay data used in the Keystone resource estimate was initially verified by MDA in 2010. To the best of MDA's knowledge, there has been no additional work, either drilling or underground, on the Keystone deposit since 2010. MDA reviewed the project data again in 2015 and accordingly, the drill data has an effective date of June 15, 2015. The resource model and estimate, based on the June 15, 2010 data, was completed June 22, 2015.

For this current Keystone resource estimate, MDA first reviewed the data verification and validation procedures that had been conducted as part of MDA's assessment of the project data in 2011. Cross-sections spaced 100ft apart were then created and drill assays plotted down-hole. Also plotted on the drill trace were the designated vein intervals ("K5, "K13, etc) as determined by SGM and as reported in Payne (2008). Grade domain cross-sectional polygons for the K5 and K13 veins were created and drill assays were coded by the vein polygons. Resource estimation using cross-sectional polygonal model was chosen for Keystone due to the limited drill data and the expected Inferred-only classification. Using a cut-off gold grade of 0.12oz Au/ton for the individual gold polygons, an undiluted Inferred-only polygonal resource was calculated for a portion of the Keystone deposit area.

14.3.1 Database

The drill data used for the Keystone resource estimate contains 23 surface core drill holes and seven RC drill holes. The Keystone resource database contains gold values for 1,471 sample intervals. All of the core assay data were used in MDA's current resource estimate; the RC assay data were not used due to verification and sample-precision concerns.

The project coordinates are truncated California State Plane – Zone 2 coordinates using the NAD 27 datum.

14.3.2 Geology Background

Mother Lode-style Gold mineralization within the Keystone deposit is localized within two north-northwest-trending structural zones: the West Contact zone on the west and the Medean zone on the east. The mineralization style is similar to the Lincoln-Comet mineralization in that the gold-bearing veins, usually 1 to 4ft in thickness, occur as fissure veins within the structural zones. A pervasive fault overprint is common, and gold mineralization appears to coincide with the late faulting. Mineralized quartz veins are often bounded by fault slip planes that generally define one or both vein walls, and sheared, ribboned vein-quartz exceeding 1ft in width is commonly associated with the higher-grade mineralization.

The Keystone veins trend N30°W though in contrast to the Lincoln-Comet veins, which dip primarily west, the Keystone veins dip west at 50 degrees to 60 degrees in the West Contact zone and 60 degrees to 70 degrees in the Medean zone.

The “K5” structure/vein is the dominant vein in the West Contact zone. As logged by SGM geologists, the K5 vein has been intercepted in 15 drill holes. In three of these holes, workings (4ft to 6ft voids in the drill core) have been encountered which are believed to be historic development associated with the South Spring Hill mine. Drill spacing on the vein is relatively wide at 200ft to 300ft. Weak sub-grade mineralization occurs in holes within the southeast down-dip portion of the structure/vein. Smaller structures/veins (“K22 through “K26”) occur within the footwall and hanging wall of the K5 structure.

The “K13” structure/vein is the footwall vein of the Medean zone and appears to have been exploited by the Medean mine and northern extension of the Talisman mine. As logged by SGM geologists, the K13 vein has been encountered in 18 drill holes though most of these intercepts are just weakly mineralized and were not included in the MDA resource model. The drilling is more closely spaced on the K13 vein as compared to the K5 vein, though still considered relatively wide at about 200ft. The K13 vein is the primary footwall Medean zone structure while a number of smaller structures (“K16” through “K20”) occur within the hanging wall, up to 200ft to the east of the K13 vein.

14.3.3 Coded Assay Data

The low-grade assay population represents the weakly mineralized wallrock, or low-grade portions of the structure/veins, that are likely sub-economic with a limited chance of eventual economic extraction. These samples values are all well below the resource cut-off grade (using the same 0.12oz Au/ton cut-off gold grade as at Lincoln-Comet) and therefore are not included within the undiluted polygonal resource estimate discussed in Section 14.3.6. The polygonal resource is based on the mid-grade domains only.

Table 14.11 shows the statistics for the mid-grade, domain-coded assays which are used in the polygonal resource estimate for each vein.

Table 14.11 Keystone ResourceAssay Statistics - K5 and K13 Vein

	# holes	# samples	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Sample Length	15	33	3.1	3.1			1.5	4.2	ft
K5 vein	7	16	0.262	0.159	0.235	0.9	0.081	0.9	oz Au/ton
K13 vein	8	17	0.164	0.144	0.127	0.77	0.014	0.508	oz Au/ton

14.3.4 Density

A tonnage factor of 12cuft/ton is used for the Keystone mineralization. This is the same tonnage factor used in the Lincoln-Comet resource. See Section 14.2.3 for the discussion on the Lincoln Mine Project density data.

14.3.5 Keystone Resource Estimate

The Keystone resource is based on an undiluted polygonal cross-sectional model using only the mid-grade gold polygons. In order to localize the estimate, the cross-sectional mid-grade polygons were broken into sub-polygons localized around each drill hole. These local polygons, which extended about 100ft from the drill intercept, were assigned the grade of the coded drill intercept. Away from the drill data, the mid-grade polygons which had no coded drill assays within them were assigned the average grade of all drill intercepts of that specific vein.

MDA reviewed the assigned grade for each polygon and those polygons with grades below the 0.12oz Au/ton cut-off were removed from the resource tabulation. Due to the wide drill spacing, and lack of modern underground sampling data, the Keystone resource is restricted to an Inferred classification. Table 14.11 shows the tons, gold grade, and gold ounces within each vein along with the total Keystone resource.

Table 14.12 Keystone Inferred Resources

vein	Tons	grade oz Au/ton	oz Au
K5	301,000	0.261	79,000
K13	98,000	0.189	18,000
total	399,000	0.243	97,000

14.3.6 Keystone Resource Discussion, Qualifications, Risk, Upside, and Recommendations

As is typical of other Mother Lode district high-grade gold systems, there is significant mineral variability within the Keystone deposit on a sub-sample scale to a mining scale. This “nugget” character results in uncertainty in grade estimation in the resource model.

There are no Measured or Indicated resources within the Keystone deposit. The lack of Measured and Indicated material are a result of the spatial and grade estimation uncertainties due to the limited, widely spaced drill data and also the inherent grade uncertainties within high-grade, coarse-gold deposits. Additional drilling within the currently defined deposit could materially change the existing resource with the discovery of either localized, high-grade mineralization within the known veins, or new veins branching off the main structures.

Tightly spaced drilling could result in an upgrade in classification. As in the Lincoln-Comet, some underground development is advised to better characterize the local grade variability along the veins. An Indicated classification would also warrant the construction of a three dimensional block model and grade estimate to better characterize the local grade variability and vein location.

There is potential for expanding the Inferred resource by drilling both down-dip and along strike to the northwest and southeast.

An issue that affects the Keystone resource model and estimate is that the current resource is spatially related to historical underground development. Historical maps and reports, along with the incomplete 3D drawings available to MDA, show historical mine access and development on multiple levels in close proximity to the current resource. Voids were encountered within the modeled vein intercepts in a few of the "K5" drill holes. There is the possibility that some of the current resource in both veins has been mined out.

The Keystone resource is undiluted and some dilution is expected depending on proposed mining methods. The affect on the resource is unknown but it is possible that some portions of the veins would no longer be considered economic under certain circumstances.

14.3.7 MDA's Evaluation of the Payne (2008) Resource

MDA's project evaluation included a review of the Payne (2008) project-wide Indicated and Inferred Resources for potential inclusion within the current resource tabulation. The Lincoln-Comet resources reported by Payne (2008) had been previously superseded by MDA's 2011 resource estimate and therefore were not included in this current review. The review of the Payne drilling-related Keystone resources has resulted in the current Keystone resource reported in Section 14.3.

MDA's evaluation consisted of reviewing and verifying the original source data, along with the modeling and estimation methodology used by Payne. MDA focused on the drilling-related Indicated and Inferred resources within the Keystone area, and Payne's Inferred Resources estimated using historical data from underground mines within the Lincoln Mine property. Some of the latter included the Keystone, the Central Eureka area, which includes the Empire mine, and the Lincoln-Wildman-Mahoney mines.

14.3.8 2008 Historic Underground Mine Resources

The Inferred Resources estimated from historic mine data (designated "Inferred Resources B" by Payne) total about 294,000 ounces of gold. These resources are based on 17 references including correspondence, consultants' reports, company annual and monthly reports, and government reports that date from 1876 to 1939. The information provided in these reports is limited to general tons and grade estimates within remaining "ore blocks" in 20 separate levels within four historic mine areas; the Keystone, Lincoln Consolidated, Wildman & Mahoney, and Central Eureka mines. As reported by Payne, there is a lack of specific underground sample data or information regarding methods used to estimate volumes and grades. The various historic workings are all inaccessible so there is no ability to verify any of the historical data.

Though there is definite potential to develop mineral resources within these historic mine areas, albeit at a significant exploration cost, MDA does not consider the mineralization within these historic underground areas as valid mineral resources under current NI 43-101 or CIM guidelines. These mineralized areas are therefore removed from the tabulation of Lincoln Mine project resources.

14.3.9 2008 Keystone Drilling-Related Resources

Payne (2008) stated resources for seven distinct veins intercepted by modern (post-1983) drilling in the Keystone area. Indicated Resources totaled about 35,000 ounces of gold from five veins (designated the “K5”, “K13”, “K18”, “K19”, and “K23” veins) while Inferred Resources totaled about 111,000 ounces gold from the same five veins, plus two smaller veins (designated the “K17” and “K25” veins). The “K5”, historically mined as the West Contact vein, is the primary mineralized vein accounting for 63 percent of the Indicated gold ounces and 82 percent of the Inferred gold ounces.

According to the 2006 and 2008 Technical Reports authored by Payne, the resources derived from modern exploration drilling were estimated by manual methods from vertical longitudinal projections constructed for six individual gold-quartz vein structures (the “K5” and “K23” veins were modeled along the same longitudinal projection). A 3ft minimum horizontal thickness was used and wallrock dilution was added if the mineralized drill intercept was less than 3ft in horizontal width. The final individual block boundaries are hard boundaries, primarily polygonal in morphology. A resource cutoff grade of 0.140oz Au/ton was applied to each individual resource block and, unless restricted by geology, the block boundary was usually projected halfway to the adjacent sub-grade drill intercept. The resource estimates were undiluted.

Resource classification was based on confidence in geologic interpretation as determined primarily by distance from closest drill intercept. Where unrestricted by adjacent holes, the polygonal projection of Indicated Resource blocks loosely adheres to a 100ft by 50ft- radius ellipse with the long dimension being sub-horizontal. The Inferred blocks usually represent projected extensions (usually another 100ft along strike of mineral trend) of mineralization into areas lacking in drill data.

MDA’s review of the 2008 Keystone drilling resources showed that only the “K5” and “K13” resource estimates used more than two drill intercepts for grade estimation. The other five veins, all but one located hanging wall to the “K5” or “K13” veins, were often defined by erratic sub-grade intercepts in which spatial continuity was not well defined. An internal SGM review of the Keystone property by Zahony (2012) stated:

“…there are really two main fissure veins systems with perhaps a weaker third system. The main high-grade fissure vein, mined and explored extensively in the past is the Contact Vein…A second major fissure vein system is the Medean vein which is low grade but wider.”

The Contact Vein refered to by Zahony (2012) is Payne’s “K5” vein, while the Medean vein correlation is not clear but is assumed to be the “K13” vein. The latter assumption is based on the “K13” location and increased continuity between drill holes. Due to their erratic mineralization, and small potential resource, the five minor veins were not included in MDA’s current resource as reported in Section 14.3.

The Keystone drilling is at an approximate 200ft spacing along the primary mineralized veins. Under the assumption that the strong spatial and grade variability observed within the more drill-defined Lincoln-Comet system occurs also within the Keystone vein systems, MDA believes that the 2008 Indicated Resource classification is used too liberally for the Keystone resource. Using the same classification guidelines as being used for the Lincoln-Comet resource, all of the drill defined Keystone resource would be classified as Inferred only.

15.0 MINERAL RESERVE ESTIMATE

No estimates of Mineral Reserves were made for this report.

16.0 MINING METHODS

A preliminary economic assessment was completed for the project based on mining the Lincoln-Comet deposit by underground methods in 2011. MDA has updated this study with work completed between 2012 and 2014. The only change to the mining methods section of the 2011 study was to remove about 3,000 feet of development from Tables 16.4 and 16.5 that have been completed. About 3,100 tons of low grade material has also been stockpiled close to the existing mill. The scope of work for this study includes the analysis and selection of an appropriate mining method and production rate. Multiple factors contribute to the selection of a mining method. These include the deposit geometry, the strength of the host rock and vein, the depth of the deposit, the infrastructure and historical mining in the district, economies of scale, and existing permit limitations. The mining rate has a direct proportional relationship to the mining method, and as such, affects and is affected by the mining method selection.

Extensive development of the Lincoln Mine project has been completed, and many permits for mining have already been obtained.

16.1 Deposit Geometry

The Lincoln-Comet deposit consists of a network of veins that are variable in width, strike, and dip. In general, the veins are considered sub-vertical and steeply dipping with notable flatter exceptions (e.g., 37 vein). The mining method must be flexible to accommodate the variability of the vein structures, provide for minimum dilution even when the vein widths decrease, and allow for maximum mechanical advantage (i.e. reduction of labor). The general vein geology is described in detail in Sections 7 and 8 of this report.

The general strike of the deposit is thirty (30) degrees west of north. The vertical extent of the deposit considered in this study is from 600ft to 1500ft above mean sea level. Due to the narrow nature of the deposit and the depth at which it extends to, underground mining is the only option considered for the extraction of this deposit. This is congruent with the permitted mining activity for the site.

The deposit contains a number of instances where veins intersect. Some of the material near the intersection of veins will experience higher dilution when mined close to the intersection, and some may need to be left in place due to geometry and dilution issues.

16.2 Geotechnical Characteristics

Geotechnical studies for the deposit were completed by Watters in 1988 and Golder Associates in 1989. Quantitative measurements of geotechnical conditions in and adjacent to the mining areas are limited to observational and laboratory testing of core samples from previous drilling projects.

Cores tested for unconfined compressive strength ("UCS") on nine samples ranged from a low of 5,478 psi to a high of 36,053 psi, averaging 17,083 psi (Watters, 1988). Golder Associates (1989) identified, tested, and quantified three major rock type categories present in drill core, prior to underground development of the Stringbean Alley decline ("SBA decline"). These rock types and their associated UCS, rock quality designation ("RQD"), rock mass rating ("RMR"), and tunneling quality index ("Q") are summarized in Table 16.1.

Table 16.1 Summarized Rock Strength Data

ROCK TYPE	UCS	RQD	RMR	Q
Augite Porphry	22,000	81	59	40
Interbedded Tuffaceous Greenstone and Slate	15,000	56	60	28
Altered Metavolcanics and Metavolcaniclastics	16,908	69	54	35

The existing workings of the Lincoln Mine project (the SBA decline and associated development) have stood well for the 20 years of their existence. Steel sets, reinforcing mesh, and shotcrete support the SBA decline portal for a length of approximately 80ft, supporting the near-surface weathered ground mass. With the exception of the decline portal, little support beyond spot placement of friction bolts and steel mats has proved to be necessary. The SBA decline was supported upon development with friction bolts and wire mesh which are largely oxidized and considered ineffective at this time, although generally unnecessary. With the exception of local faults and zones of structural weakness from joint sets and other fractures, the stability of underground workings is expected to be good, with little rock support needed for the short duration the ground is open in the stopes. Currently, SGM performs maintenance and scaling of the workings as necessary, with minimal sloughing or slaking occurring.

Areas of blocky and slabby ground are observed within the current mine openings. These areas have high joint cohesion; some are spot bolted; the majority are unsupported. These slabs are not expected to be a common occurrence in the small openings of the production areas; development openings, which are larger in span, may need spot bolting and very occasionally wire mesh. Areas which cross slate/metavolcanic contact zones may need additional support.

The need for ground support during development and production mining phases is expected to be light. Test mining that has taken place to date indicates that little support is needed during short mining cycles. With proper stope cycling, maximum projected time for unsupported ground in the stopes will be days.

Ground support in the stopes will entail spot bolting with 4-ft split-set bolts and standard plates, with additional wire mesh or mats installed where ground is fractured or highly jointed. In some places, the stope width may limit ground support options. The bolt length utilized will be determined by the stope width and orientation of the slab needing support. Spot bolts will be installed at the discretion of the miners or by order of geologists or engineers. Table 16.2 shows the typical ground support required by each heading.

Table 16.2 Typical Ground Support By Heading

Typical Ground Control by Heading			
Opening Type	**Support Type**	**Bolt Spacing**	**Notes**
12x15 Decline	Spot Bolting, strap and/or wire mesh where needed	3.5 /ft	May require additional bolting at level intersections, noses and at fault intersections
8x8 Level Drift	Spot Bolting, strap and /or wire mesh where needed	1.75/ft	May require additional bolting at dump pockets and noses.
Scram	Occasional Spot Bolting	0.25/ft	Very short stand-up time
Raise	Spot Bolting, strap where needed	0.67/ft	Includes non-support bolting installed for temporary lagging, etc.
Backstope Mining	Spot Bolting, strap where needed; hydraulic sand backfill	0.25/ft	Very occasional bolting anticipated

With increasing mining depth, higher ground pressure is anticipated, which will require an additional future review of the ground support program. MDA recommends that a ground support plan be designed by a geotechnical engineer.

16.3 Backfill

This study assumes that hydraulic sand backfill will be employed in the stopes to serve as the sill upon which to slush and drill from, as well as to serve a role in ground support as the stope is mined. Paste backfilling has also been considered but has been eliminated due to its high cost.

If necessary, the tailings will be sized via cyclone at or near the tailing thickener. Slimes will be disposed of along with excess tails at the Surface Fill Unit (“SFU”). Thickeners will hold approximately 210 tons of tailings surge capacity. Tailings can also be dry stacked after disk filtering for temporary handling prior to loading into trucks and transported to the SFU. Backfill sand will be gravity fed through a borehole from the surface to the 1300-ft level and distributed from this level by gravity.

The miners in each stope will construct a backfill wall of mine timber and burlap to allow for decant at each end raise. Due to the small quantity of each pour, and top surface, no additional drainage materials are anticipated. The final one foot of each pour may have cement added at approximately 5-10%, creating a hard surface to work upon for the following cut.

MDA recommends that further testing of tailings material characteristics be completed prior to final backfill engineering.

16.4 Selection of Mining Methods

The deposit is a narrow vein deposit that will require an effective grade control program and efficient mining of the narrow mineralized veins. As such, the project requires a unique approach to combine modern rubber-tired equipment and decline haulage with traditional overhand narrow-vein stoping methods.

The mining method will need to have the ability to mine narrow veins that are highly variable geometrically (irregular width, dip, and strike). Grade control will be very important as in many places the veins are present, but only a portion of the vein will carry economic grades. A selective, rather than bulk mining method will be required. The possibilities include shrinkage stoping, long-hole (or blast hole) stoping, cut-and-fill stoping, and other variations of those methods.

Shrinkage stoping may cause too much dilution of the ore-grade material, based on historic observations of stope stability and performance. Additionally, the vein irregularity makes shrinkage stoping geometrically prohibitive.

Long-hole stoping also suffers from dilution issues in such irregular veins. In very narrow vein deposits, resuing can be used; however, in this deposit, the grade definition is often not apparent enough to justify or even allow this somewhat costly method.

Breast stoping was considered but discarded as the primary mining method for the following reasons:

- It requires approximately double the number of working faces and associated development to maintain similar production rates to cut-and-fill stoping.
- Moving slushers, set-up and breakdown time, and blast fume ventilation are greater than back stoping, resulting in lower efficiency.
- It does not decrease dilution except in areas where the vein is changing strike/dip within the stope cut.
- It does not facilitate stope connection for escape, ventilation, etc. except during backfilling.

Previous studies (for example, Russell and Hazlitt, 1992; Armbrust, 1994; Smith, 1997; Behre-Dolbear, 2007) have all identified some form of cut-and-fill stoping as the method of choice for this deposit. Typically, previous studies were based upon significantly (100% or more) wider vein widths allowing for more mechanization in the stopes.

Cut-and-fill stoping with an average minimum mining width of 3ft is the preferred method as it allows for a high degree of flexibility, excellent recovery, and low dilution. If the veins display areas of greater width or regularity, breast stoping could be applied locally, which would provide an economic benefit. This should be studied and evaluated during mining. Should conditions warrant, any of the mentioned methods can be applied. Cut-and-fill utilizing mill tailings for the fill is the selected method for this evaluation and for planning.

16.5 Mining Rate

The production rate was chosen based on the mining method and the desire to maintain a minimum of a five-year operation. Preliminary economic estimates indicated the cutoff grade would be around 0.22oz

Au/ton. Table 16.3 shows the material available for mining based on a 0.22oz Au/ton cutoff grade, 20% dilution, 20,000 tons of included below-cutoff material, and a 10% material loss.

Table 16.3 Material Available for Mining

Item	000's Tons	oz Au/t
Undiluted Resource*	210.3	0.573
Dilution	42.1	0.000
Include waste**	19.6	0.200
Total diluted resource	272.0	0.457
Ore loss (10%)	27.2	0.457
Total diluted resource available for mining	244.8	0.457
* The undiluted resource contains mostly inferred materials		
** The included waste is based on 25% of the material above a 0.18 oz Au/t		
At 350 days per year a five year operation rate is 150 tpd.		

The 150 tpd rate will require about eight stopes to be mined per day with about 13 being active. This may prove to be a little aggressive, but MDA believes that it is achievable.

16.6 Mine Development

The Lincoln-Comet deposit is currently accessed via the SBA decline. The SBA decline and existing workings are shown in Figure 16.1 (plan view) and Figure 16.2 (long-section view). A more detailed plan view of the 900 level development is shown in Figure 16.3. Existing crosscuts, located at approximate 200ft intervals, are numbered from the portal to the bottom of the decline. These crosscuts are described as bearing either east or west of the SBA decline. For example, the 8th crosscut in the decline, on the west side of the decline is labeled SBA8W (see Figure 16.3). A drift beginning in a crosscut would then be labeled as the first drift in the cross-cut and described by its bearing. As example, the drift to the north in SBA8W is labeled 8W1N. Some of the pre-existing sublevels have colloquial labels such as "Miner's Hall" or "Larry's Stope".

To the extent possible, development will be in vein material in order to keep development in waste rock to a necessary minimum. The preliminary design provides that 20,300 tons of “ore grade” material are produced by development. Total waste tonnage produced is 89,800 tons. Underground development will utilize the existing underground workings to the extent possible.

SGM mine development in 2012 through 2014 included construction of the 1,200 and 1,300 level portals and secondary access along with development drifting on the 900, 1,000, and 1,100 levels.

Figure 16.1 Existing Underground Development – Plan View

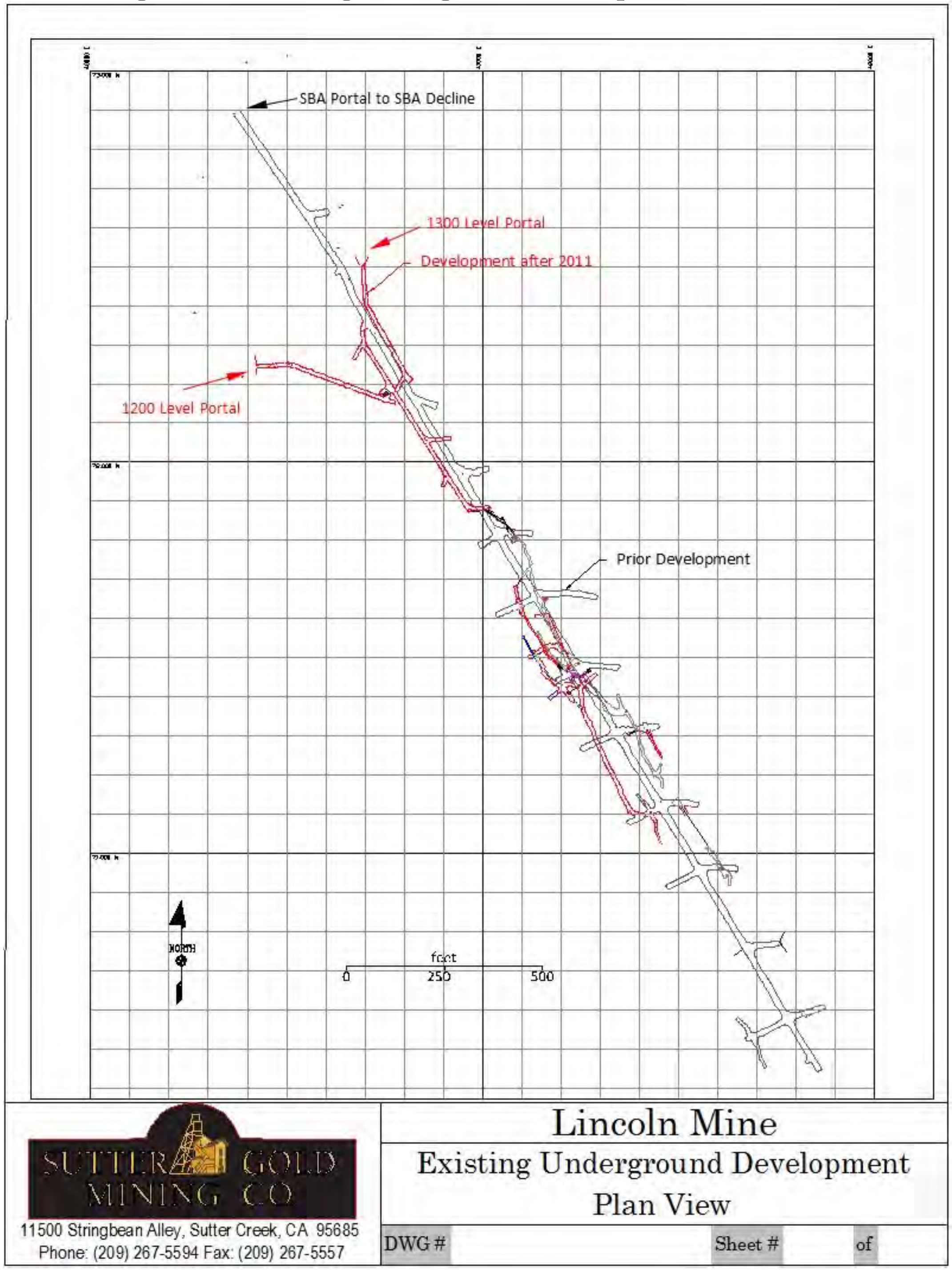


Figure 16.2 Existing Underground Development – Long Section looking N60W

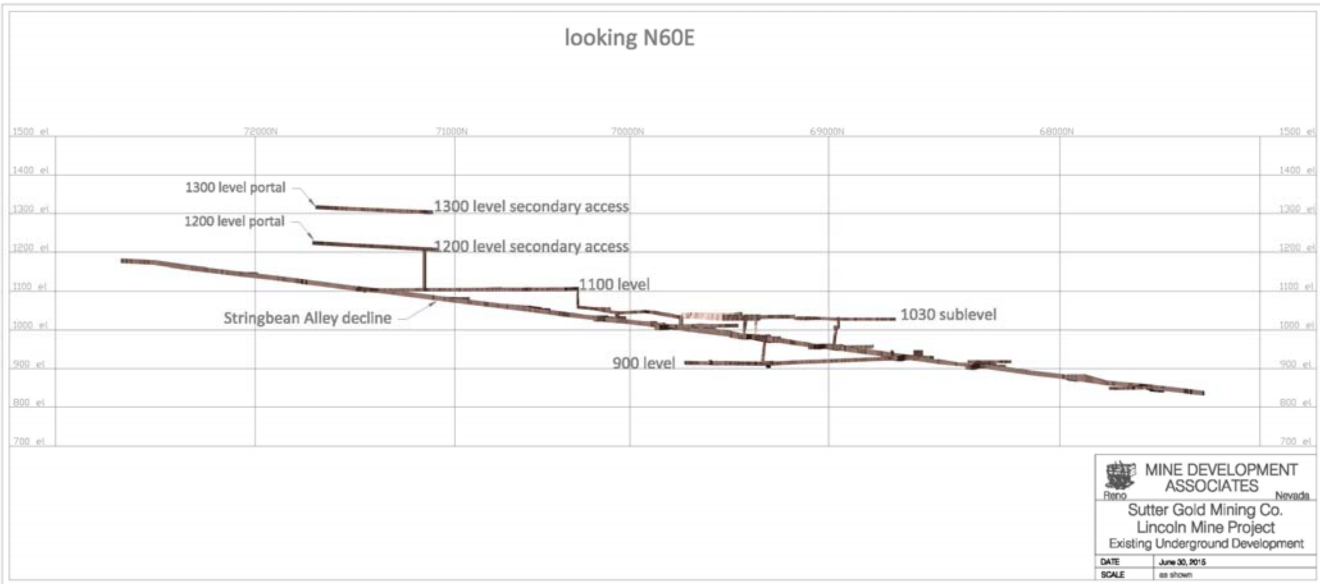
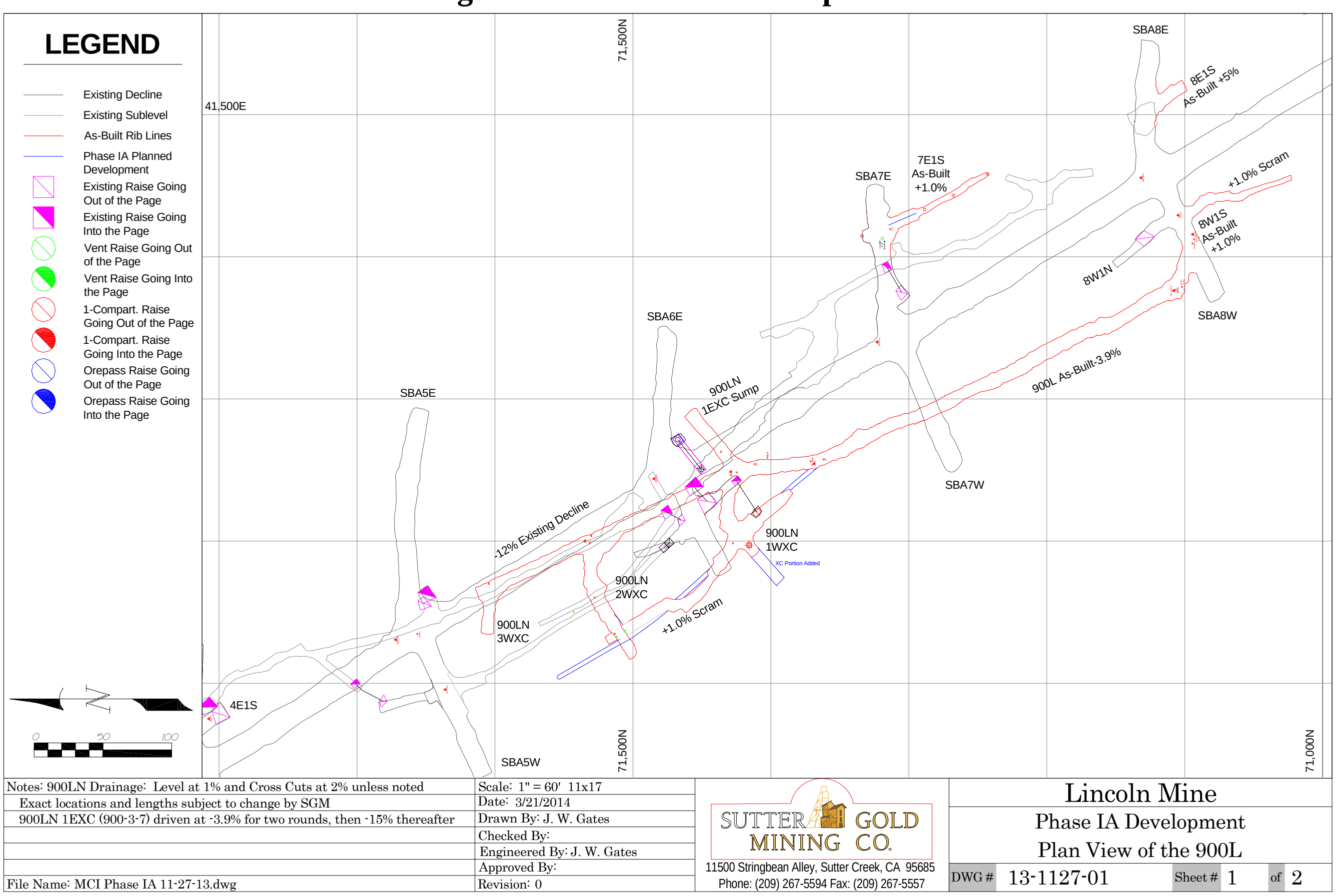


Figure 16.3 900 Level Development



16.7 Mine Level Development

Horizontal levels are planned on regular 100-ft spacing for access to the stope areas. Each level, consisting of one main drift (more if geometry warrants) driven from the SBA decline and its existing cross-cuts, will allow for access to multiple veins and stope panels, while providing near-horizontal transport of 'ore grade" material from stopes to main haulage ore-passes. Each vein and its stopes will be accessed via crosscuts from the main level drift.

Each level has been designed in such a way as to minimize distance from the centerline of the drift to each draw point while providing access to the multiple veins. The lowest level of the mine is designated the 700 level; the highest level of the mine is designated the 1300 level. Levels are referenced to the nearest 100-ft increment of elevation above mean sea level.

Approximately 2,200ft of development drifting has been completed in 2012 through 2014 by SGM on the 900, 1,000, and 1,100 levels.

16.8 SBA Decline Haulage and Transport

Haulage during development will be accomplished utilizing LHDs to muck material from the working faces initially to waiting trucks in the SBA decline. As development progresses, LHDs will deliver muck to the dump pockets located on each level above controlled loading chutes accessible by trucks from the SBA decline. A 15 to 22-ton haul truck will self-load from each dump pocket and transport the waste, or ore, to the surface via the SBA decline. Material to be processed in the mill will be delivered to the coarse-ore bins located at the rear of the mill building.

16.9 1,200 Level Decline, 1,300 Level Decline, and Ventilation Raise

The SBA decline will continue as a 12-ft by 15-ft decline at the same gradient. Crosscuts driven from the SBA decline will match the existing crosscut dimensions, allowing for haul truck access to the area immediately adjacent to the SBA decline.

Secondary access is currently provided by a second decline (named the 1,200 level decline; previously named "North Star" in the 2011 technical report) driven from the southeastern corner of the process plant pad (1210 elevation). This decline, driven at 8ft by 8ft, is just under 600 linear feet at a slight negative gradient to the 1200 level. An approximately 130-ft vertical raise connects the 1,100 level off the SBA decline to the 1,200 level decline at the 1,100L 2WXC location (Figure 16.1). The 1,200 level decline provides exhaust ventilation and secondary entry/egress to the mine workings, as well as functioning as an early production portal for the stopes above the 1200 level.

Additional access is provided by a 200ft-long, 8ft by 8ft decline driven from the 1310 elevation near the tailings dewatering plant to the 1300 level in the 51 vein. This decline (named the 1,300 level decline; previously named "Wabash" in the 2011 technical report) provides another escapeway, as well as an ingress point for the sand backfill.

An exhaust and escape raise from the 1200 level, through the 50 vein, to the surface near a SGM owned house on the Lincoln-Comet mine property, will also be developed later in the life of the mine.

16.10 Drifts

Level drifts will be sized at a nominal 8-ft by 8-ft dimension. Drift mining can be accomplished either by use of mechanized drill jumbo or jackleg drill and LHD. Larger drift sizes may be considered if they lead to more efficient movement of materials, as long as WRP permit limitations can be met.

16.10.1 Structural/Vein Model and Gold Mineral Domains

Cross-sections looking N30W and spaced at 100ft intervals were created across the Keystone deposit area. Drill assay data along with the designated vein intervals were plotted on the cross-sections.

After reviewing the 2008 model, and then evaluating the current cross-sections, MDA determined that the minor footwall and hanging wall veins would not be included in the current resource due to the limited drill intercepts and uncertainty in structure/vein continuity. Accordingly, the current resource model is based on just the K5 and K13 veins.

The Keystone drill assays were color-coded based on the population breaks seen in the general assay population: low-grade 0.01oz Au/ton to 0.07oz Au/ton, mid-grade (0.07oz Au/ton to 0.25oz Au/ton), and high-grade (>0.25oz Au/ton). Using the labeled K5 and K13 vein intervals as a guide, low- and mid-grade gold mineral domain cross-sectional polygons were created based on the drill assay populations. Due to the limited number of high-grade samples, a unique high-grade domain was not created and those high-grade samples are included within the mid-grade domain.

The gold polygons were limited in their elevation extent, both up-dip and down-dip, by either existing drill data or by geologic constraints (structural intersections, etc.) as modeled by Payne in 2008. Within some holes, where there is a sharp contact between the mineralized vein and the weakly altered wallrock, the low- and mid-grade domain boundaries are at the same drill sample footage.

The current K5 mineralized vein, as interpreted in the cross-sectional model, has an approximate 1,000ft strike length and an 800ft down-dip extent. The vein thickness that contributes to the current polygonal resource ranges from 3ft to 8ft.

The K13 mineralized vein, as interpreted in the cross-sectional model, has an approximate 800ft strike length and a 600ft down-dip extent. The vein thickness that contributes to the current polygonal resource ranges from 3ft to 6ft.

16.11 Pre-Production Development

Pre-production development includes only the development necessary to place the mine at a production capacity. The development necessary for SGM to safely access and mine the deposit will include the extension of the existing SBA Decline by 391ft, 125ft of additional 12ft by 15ft cross cuts, the excavation of 3,469ft of 8ft by 8ft level access and dump pockets, 1,536ft of raise mining, and 579ft of scram drift mining to develop 13 stope panels. These footage figures are exclusive of the development work completed by SGM in 2012 through 2014.

This development is the minimum to start the operation. MDA recommends that a test mining period be included in the pre-production development period as planning for the project proceeds for the purpose of completing a detailed evaluation of the stope panels required to achieve production.

16.12 Scram Drift

A scram or "slusher" drift is mined at the bottom of each stope panel, connecting the ends of the stope panels to the accesses or draw points and raises. The scram drift dimensions will be a minimum of 3ft wide (i.e. stope width) and 8ft tall. The scram will be mined entirely in mineralized material with few exceptions noted in the pre-production development accounting. Following conventional drilling, blasting, and ventilation, miners will muck the mineralized material to the draw point using electric or air powered slushers; the “ore grade” material will then be transported by LHD. Scram cuts are considered the base cut of a stope panel.

16.13 Stope Raise

Each typical stope panel will require one raise to be developed from the lower to upper level of the stope panel to provide for access, ventilation, and utilities. Panels that are combined along strike can share raises and multiple accesses. For this study it was assumed that each panel would require one raise in development. The raise will be mined in “ore grade” material when possible, with a finished opening dimension of stope width (typically 3ft wide) by 8ft long (the longest dimension aligned with the strike of the vein).

Ore raises will be mined by miners working from a timbered landing. Miners will set timbers on one side of the raise for legal access – ladders with 30ft landings – while leaving the other side open to act as a muck chute and equipment chute as the raise progresses. The top of the muck chute will be covered when miners are working at the top of the raise. Utilities placed will be either temporary or permanent, depending on the location in the raise.

Following completion to the upper level, the raise will contain timber that will be removed and replaced as needed when mined through. The raise and landings will act as additional access, and the equipment chute can be used to load and remove equipment from the top.

During excavation, the raise muck will be mucked from the draw point by an LHD or slushed to a draw point where an LHD can load, and then will be transported to the nearest dump pocket to feed truck haulage.

Once the raise(s) and scram drift are in place, the stope panel is considered developed and ready for production mining.

16.14 Mine Development Schedule

16.14.1 Pre-production Mine Development

Pre-production development will develop the underground infrastructure for eight stope panels, the minimum number of stope panels necessary to support production at 150 tpd, plus an additional five panels to be available. A total of 13 stope panels will be developed. Pre-production development must allow for future production development to maintain the total number of stope panels necessary. Total pre-production development footage is listed in Table 16.4 and is calculated to occur during the 9 month pre-production period.

Table 16.4 Pre-production Development

Item	Drift Size	Length (ft)
Main Decline	12x15	391
Level Decline	8x8	409.5
Level	8x8	2105
Level X-Cuts	8x8	954.5
Decline X-Cuts	12x15	125
Scram in Ore	3x8	516.5
Orepass in Ore	3x8	397
Manway in Ore	3x8	791
Orepass in Waste	3x8	227
Manway in Waste	3x8	121
Scram in Waste	3x8	62

16.14.2 Production Mine Development

The development required during the production period is shown in Table 16.5. All development during years 2 through 6 is included in the mine operating cost estimate.

Table 16.5 Production and Development Schedule

Item	Year 1*	2	3	4	5	6	Totals
DEVELOPMENT							
Ramp (Feet)	516	0	0	0	0	0	516
Level (Feet)	3,469	2,281	2,281	2,281	2,281	0	12,593
Raise - WASTE (Feet)	350	441				0	791
Scram - WASTE (Feet)	62	0				0	62
Totals (Feet)	**4,397**	**2,722**	**2,281**	**2,281**	**2,281**	**0**	**13,961**
PRODUCTION							
Raise - ORE (Tons)	2,000	1,672	1,672	1,672	1,672	975	9,664
Scram- ORE (Tons	1,500	1,665	1,665	1,665	1,665	971	9,130
Stope Mining ORE (Tons)	3,750	49,164	49,164	49,164	49,164	24,458	224,864
Totals (Tons)	**7,250**	**52,501**	**52,501**	**52,501**	**52,501**	**26,404**	**243,658**
*Year 1 all to pre-production							

16.15 Mine Production

To meet the mill demand of 150 tons per day, at seven days per week, the mine will need to have at least eight stopes in production at any given time. At least five to eight more stope panels should be available to produce as contingency stope panels.

Production mining utilizing the overhand back stoping, cut-and-fill mining method will take place in nominal 100ft by 100ft by 3ft panels. Each panel will be developed by at least one raise and one scram drift ("slusher drift") in the vein on each level.

Stope panels will be located utilizing the most practical access point from each main drift. Defined by stope access (man-way or muck pass raise and crosscut), each stope panel will be designed for flow-through ventilation and multiple ingress/egress, when practical. Vein geometry will dictate the placement of raises and crosscuts.

16.15.1 Stope Panels

Stope nomenclature will be by vein number, vertical level, and panel number. Veins are numbered as described in Section 7.5. The panel number is the number of the stope panel, in that particular vein, as numbered from north to south along the N30W deposit strike. Identified stope panels are listed in Table 16.6 and shown in Figure 16.4 through Figure 16.8. Note that the stope panel names have a letter inserted for the level (A=1300, B=1200, etc.).

Table 16.6 Stope Panels

Level	Panel	Level	Panel	Level	Panel	Level	Panel	Level	Panel	Level	Panel	Level	Panel
700L (G Panels - 3)		800L (F Panels - 8)		900L (E Panels - 13)		1000L (D Panels - 33)		1100L (C Panels - 38)		1200L (B Panels - 22)		1300L (A Panels - 2)	
700L	42G1	800L	42F1	900L	4.00E+02	1000L	42D1	1100L	51C1	1200L	51B1	1300L	51A1
700L	42G2	800L	42F2	900L	4.00E+03	1000L	42D2	1100L	51C2	1200L	51B2	1300L	51A2
700L	42G3	800L	50F1	900L	4.00E+04	1000L	40D1	1100L	42C1	1200L	51B3		
		800L	51F1	900L	4.00E+05	1000L	40D2	1100L	42C2	1200L	42B1		
		800L	50F2	900L	4.00E+06	1000L	43D1	1100L	42C3	1200L	51B4		
		800L	50F3	900L	4.20E+02	1000L	43D2	1100L	42C4	1200L	28B1		
		800L	50F4	900L	4.20E+03	1000L	40D3, 41D1	1100L	51C3	1200L	42B2		
		800L	50F5	900L	42E3, 6E1	1000L	40D4, 41D2	1100L	28C1	1200L	40B1		
				900L	21E1, 23E1	1000L	40D5	1100L	51C4	1200L	43B1		
				900L	5.00E+02	1000L	40D6	1100L	28C2	1200L	51B5		
				900L	5.00E+03	1000L	42D3	1100L	40C1	1200L	28B2		
				900L	5.00E+04	1000L	42D4	1100L	40C2	1200L	28B3		
				900L	5.00E+05	1000L	23D1, 30D1	1100L	43C1	1200L	40B2		
						1000L	23D2	1100L	40C3, 7C1	1200L	43B2		
						1000L	42D5	1100L	43C2	1200L	51B6		
						1000L	51D1	1100L	40C4	1200L	51B7		
						1000L	51D2	1100L	51C5	1200L	51B8		
						1000L	50D1	1100L	51C6	1200L	50B1		
						1000L	51D3	1100L	28C3	1200L	50B2		
						1000L	51D4	1100L	28C4	1200L	6B1		
						1000L	50D2	1100L	45C1	1200L	6B2		
						1000L	50D3	1100L	20C1	1200L	2B1, 3B1		
						1000L	6D1	1100L	51C7	1200L	2B2, 3B2		
						1000L	6D2	1100L	5C1	1200L	9B1, 42B3		
						1000L	5D1	1100L	6C1				
						1000L	5D2	1100L	50C1				
						1000L	50D4	1100L	6C2				
						1000L	50D5	1100L	50C2				
						1000L	6D3	1100L	50C3				
						1000L	42D6	1100L	6C3				
						1000L	42D7	1100L	6C4				
						1000L	42D8	1100L	3C1				
								1100L	3C2				
								1100L	42C6				
								1100L	42C7				
								1100L	42C8, 9C1				
								1100L	3C3				
								1100L	42C9				

Since this is a preliminary economic assessment and includes Inferred materials, a detailed plan of stope panel sequencing was not completed.

Figure 16.4 The Number 2, 5, 20, & 40 Vein Stope Panels (Elevation Looking N60E)

Figure 16.5 The Number 3, 41, & 51 Vein Stope Panels (Elevation Looking N60E)

Figure 16.6 The Number 6, 7, 23, & 30 Vein Stope Panels (Elevation Looking N60E)

Figure 16.7 The Number 9, 20, 43, & 50 Vein Stope Panels (Elevation Looking N60E)

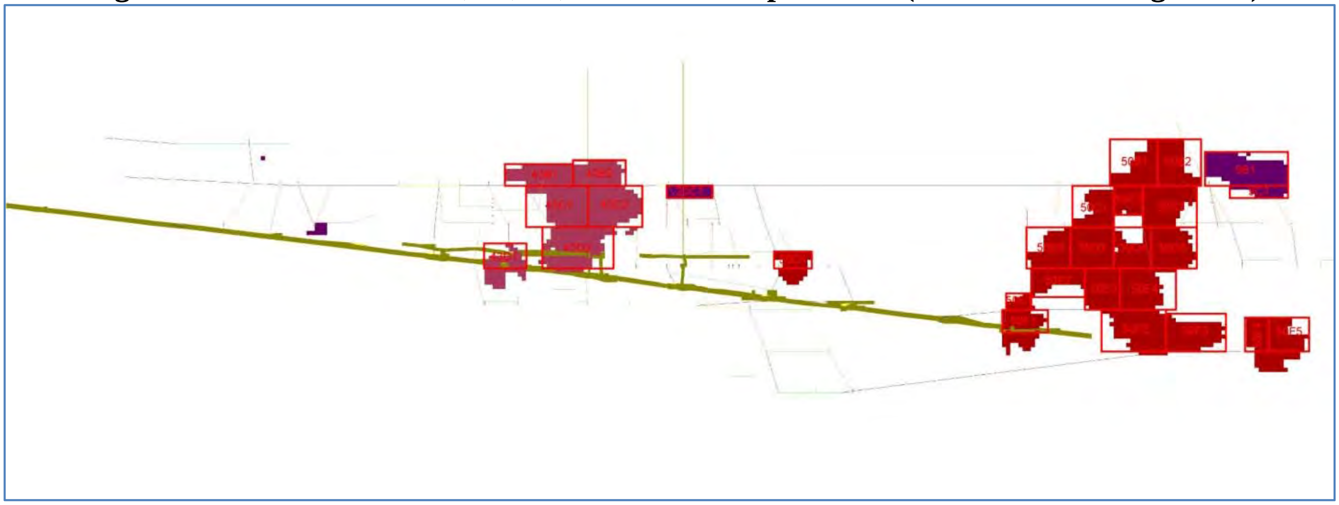
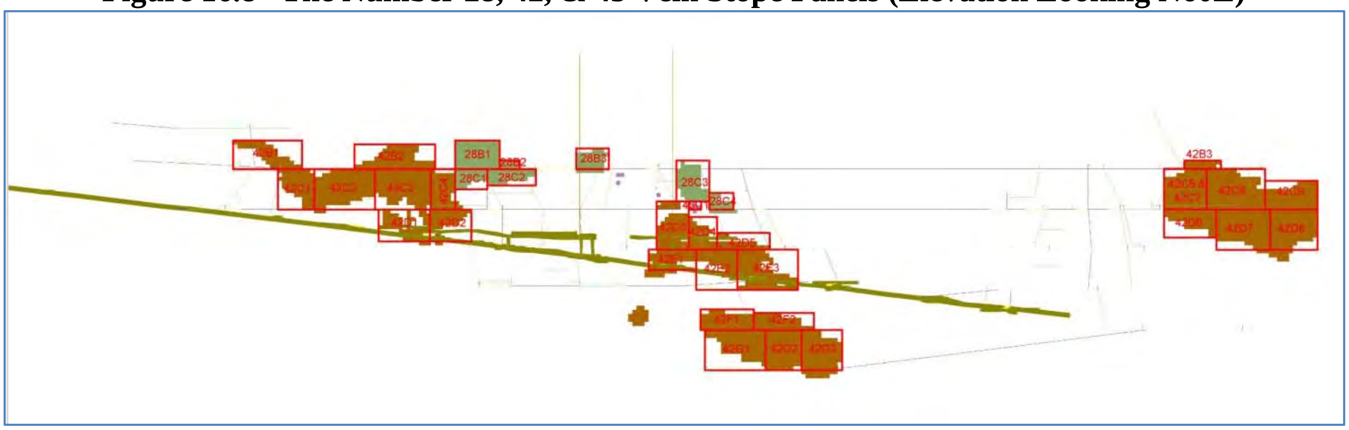


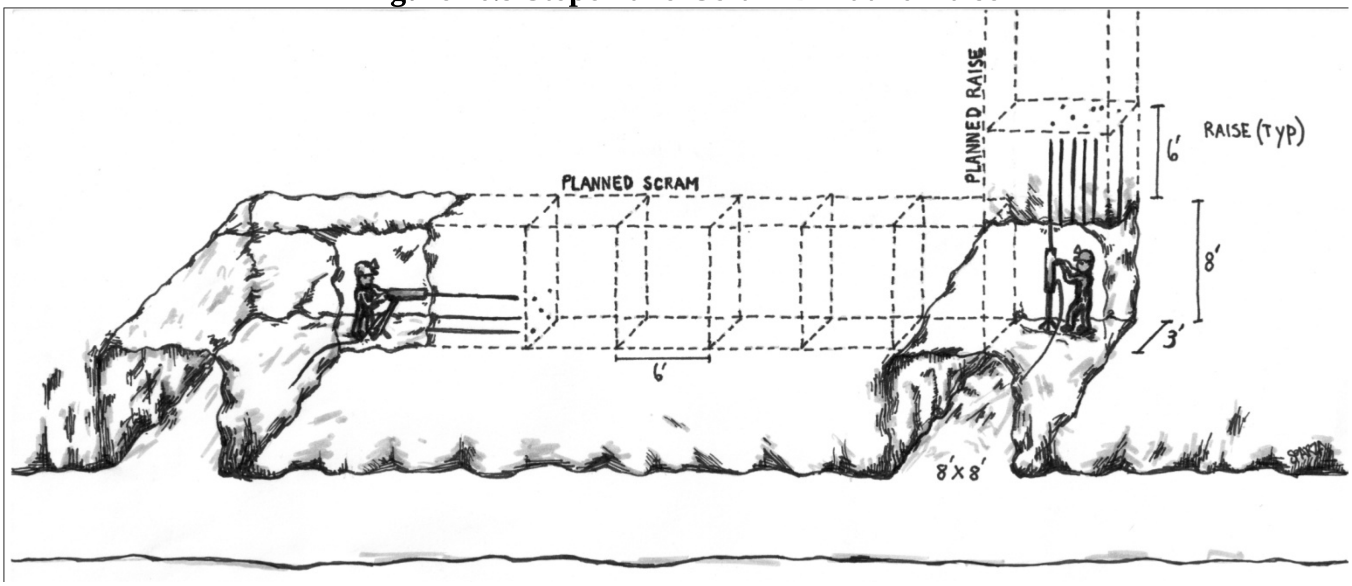
Figure 16.8 The Number 28, 42, & 45 Vein Stope Panels (Elevation Looking N60E)



16.15.2 Stope Panel Cycle

A typical stope panel (100ft in length by 100ft in height) is mined out in 17 cuts at eight days each, or 136 days. For the average stope panel, 24 days are necessary for scram and raise development. A stope will be considered ready for production when there is ventilation through the panel, entry and egress through a minimum of two openings, and at least one muck bay available. Figure 16.9 shows the stope panel scram drift.

Figure 16.9 Stope Panel Scram Drift and Raise



Back cuts will be mined overhand to 6ft height for the length of the stope panel. The cut outline will be identified by the geologist prior to production drilling. It will be necessary to sample each cut so that the mineralization can be identified as the mining proceeds. The drill pattern will call for tightly spaced holes, with a concentrated effort to minimize dilution. Figure 16.10 and Figure 16.11 illustrate the first and second cuts in the stope panel, respectively.

Figure 16.10 First Back Cut in Stope Panel

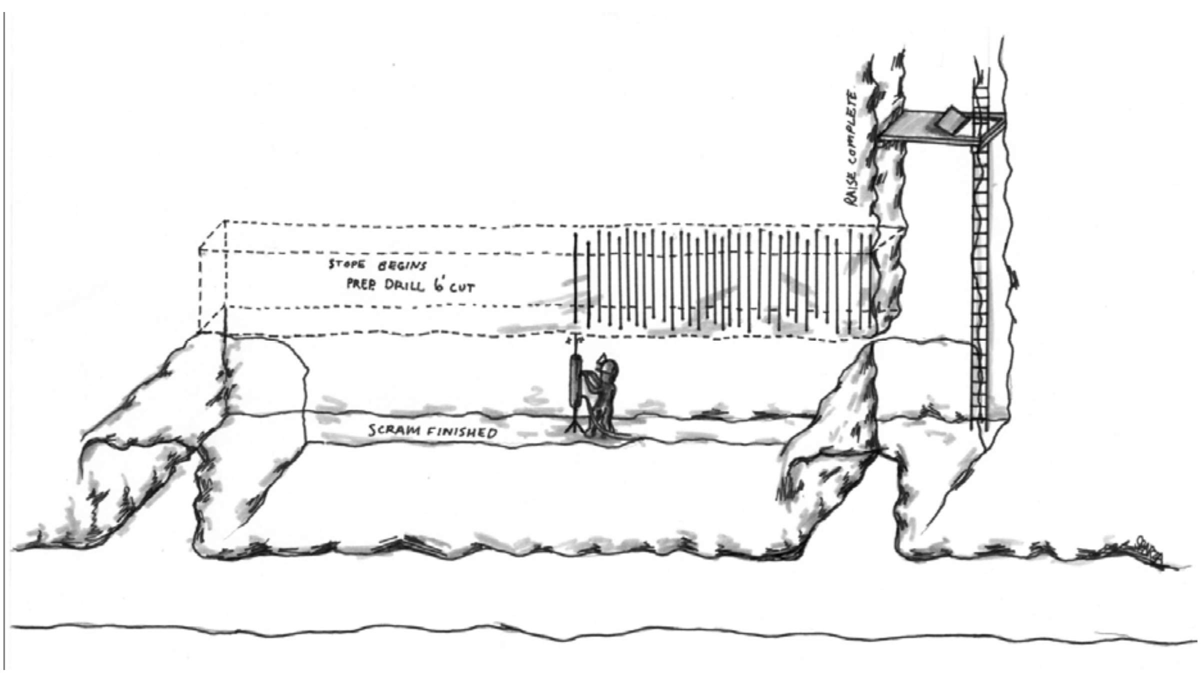
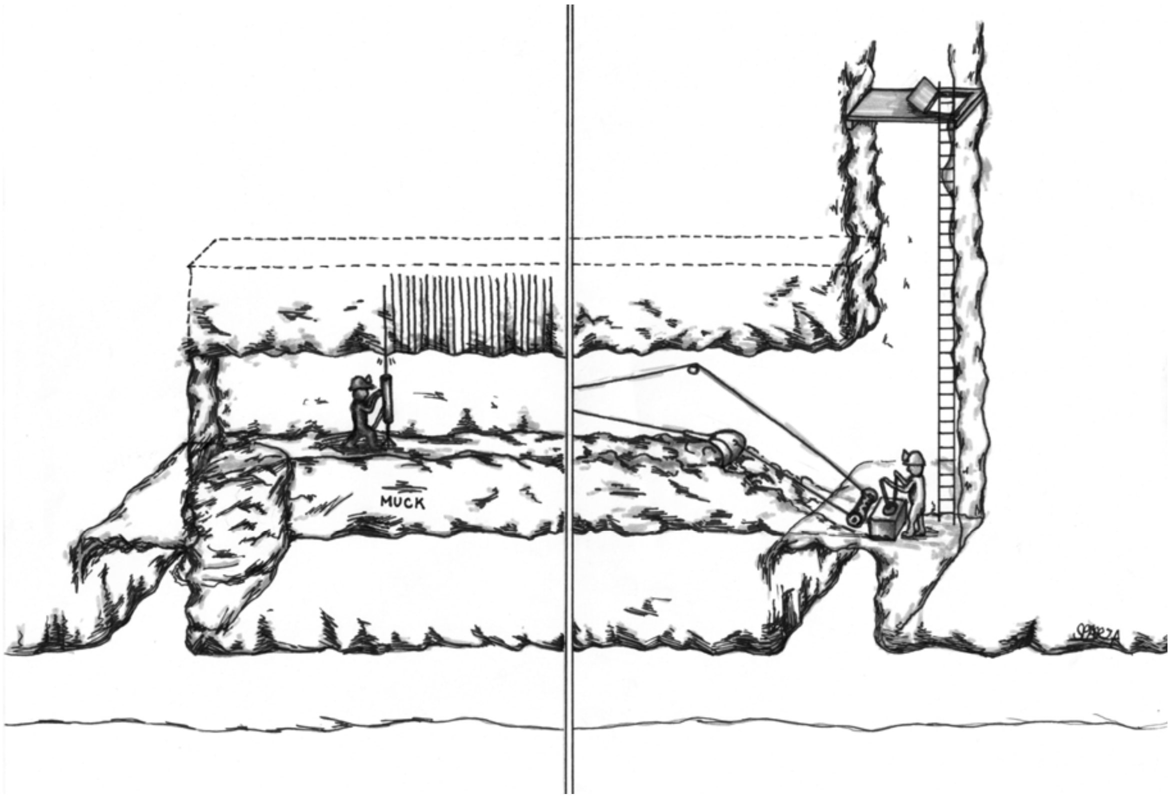
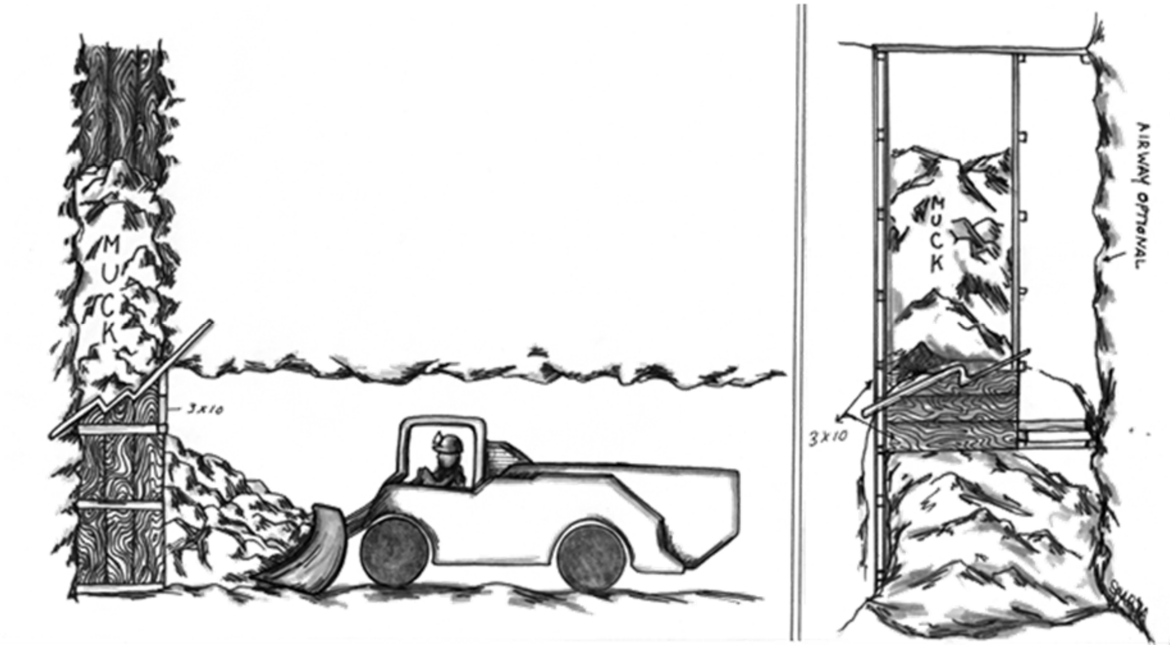


Figure 16.11 Second Back Cut in Stope Panel



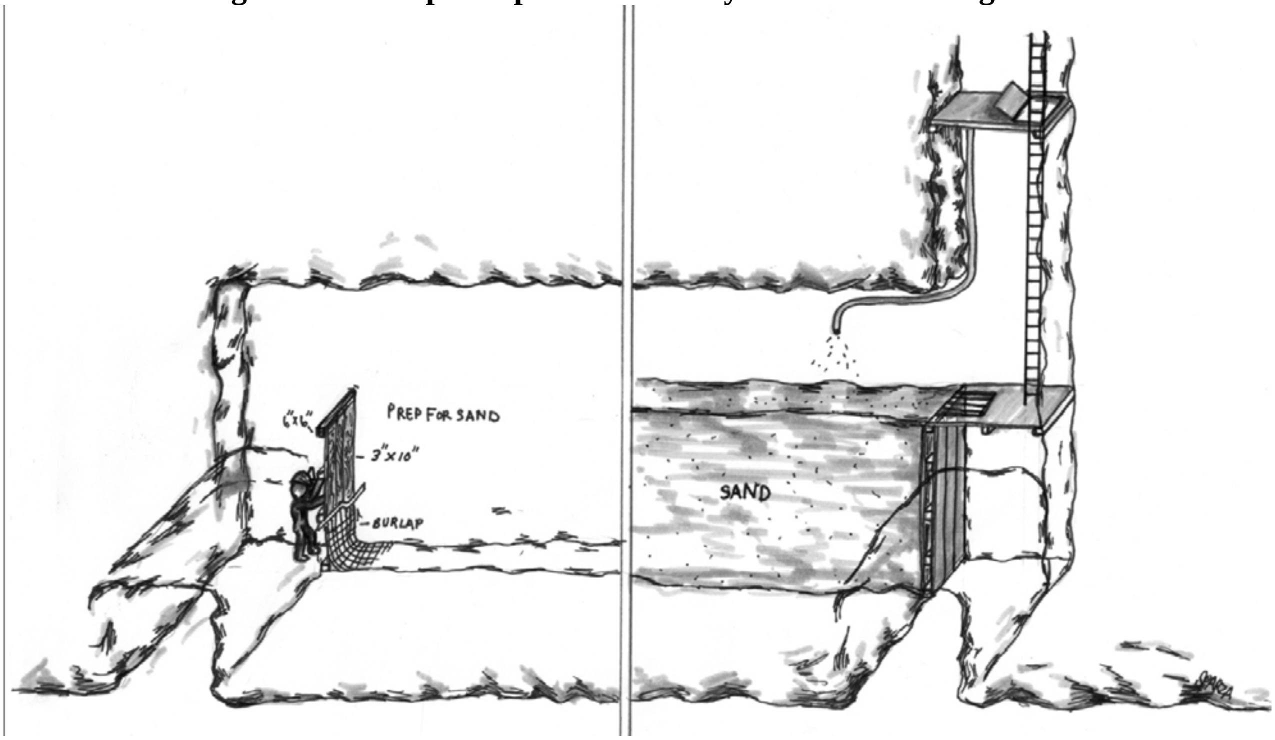
Following the blast, the stope will be scaled from the raise to the opposing end of the panel, and the slusher blocks set. The “ore grade” material will be slushed using either an electric or pneumatic double-drum slusher to a muck pass, sized at that point with a 6in grizzly. The “ore grade” material should typically break smaller than the grizzly size. “Ore grade” material that is oversize will be broken manually if necessary. Once slushed to the raise, the “ore grade” material will then travel by gravity to the level below, where it will be handled by a 2.5-yard rubber-tired LHD. The chutes in the stope panels to the draw point will be designed in such a way as to allow for free flow of material in a safe manner as the LHD removes the material. With the drawpoints located on the central haulage level, the LHD will take the “ore grade” material from each stope panel to either a loading bay in the SBA decline or, preferably, to a loading chute on the level that delivers to a cut-out lower in the decline where trucks can be filled from a pneumatically controlled chute. Underground trucks will convey the material from the chutes to the mill on the surface. Typical stope rounds will break about 150 tons. Figure 16.12 illustrates loading from the stope panel drawpoint.

Figure 16.12 LHD Drawpoint Below Muckpass



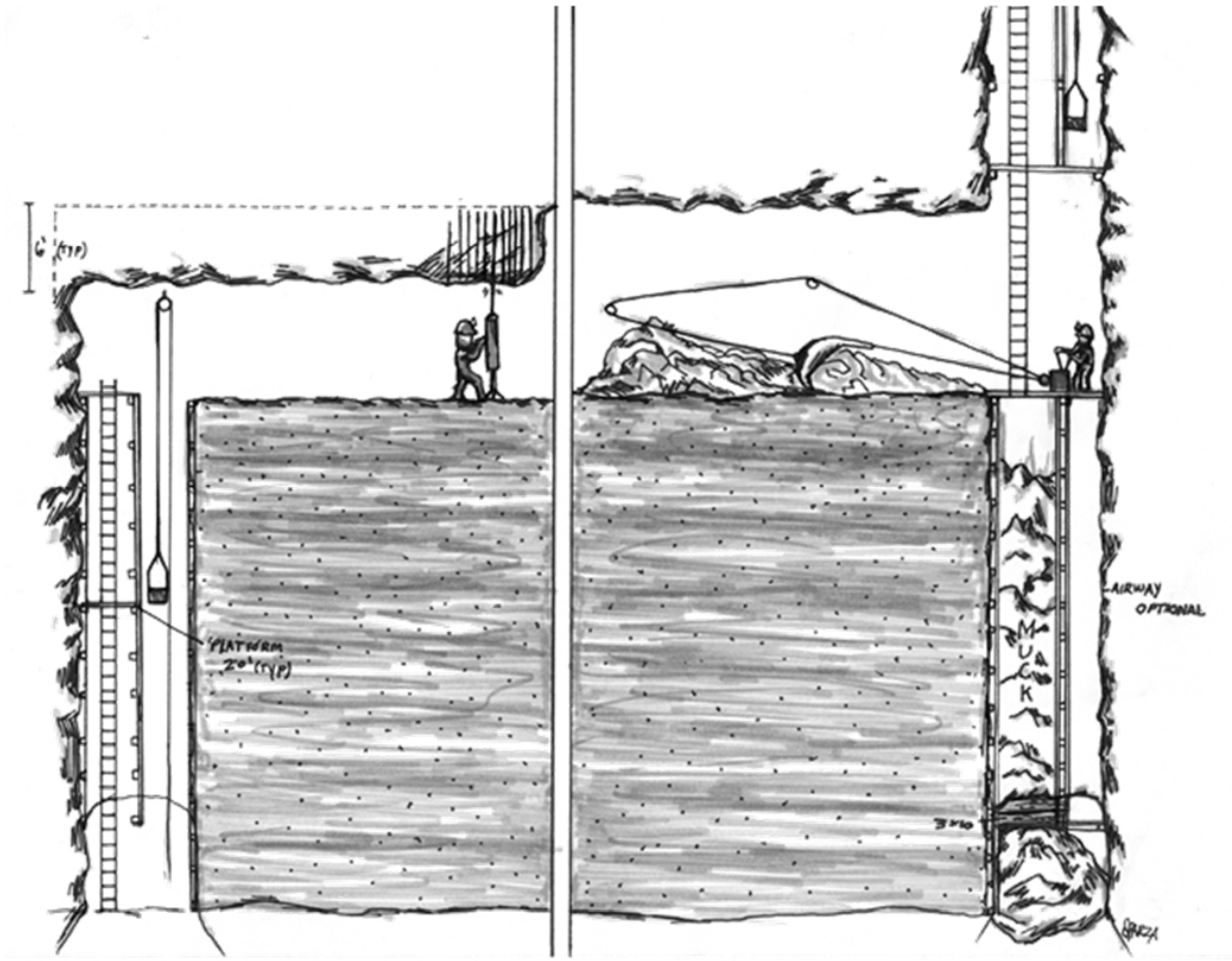
Upon removal of the "ore grade" material from the stope, the crew will prepare the stope for backfill. Generally, the stope is prepared to accept backfill by timbering the chute and manway to a height of 6ft (per lift), facing the timber with burlap, and sealing each end. Hydraulic sand backfill is then placed in the stope to the depth of the previous cut, maintaining approximately 8ft of clearance from the sill to back for stoper drilling; the top 1ft of backfill may be cemented The stope fill will be allowed to decant for a planned three days. One shift is needed to prepare the stope for the next cut, which begins the mining cycle again. Figure 16.13 shows the stope panel preparation for hydraulic backfill.

Figure 16.13 Stope Preparation and Hydraulic Backfilling



Each panel is mined 100ft vertically to the level above, or to its economic limit. Panels that connect vertically to other stope panels will be designed to minimize duplicate raises (orepasses and manways) and other development. Adjacent stope panels will be sequenced to allow for maximum efficiency in mining, ventilation, escape, and material transport. Figure 16.14 shows the sixth cut in a stope panel.

Figure 16.14 Stope Panel 6th Cut - During Production



Total production of 150 tpd is expected to require eight active stope panels. Considering unexpected delays and inefficiencies, 13 actual stope panels should be available to deliver "ore grade" material to the mill at any one time. Table 16.7 shows the typical eight-day stope panel cycle.

Table 16.7 Stope Panel Cycle

Per 100'	Day 1		Day 2		Day 3		Day 4		Day 5		Day 6		Day 7		Day 8	
Shift	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Prep for Drill																
Drill Back Round																
Load/Blast/Bar & Prep																
Slush					75	75										
Lift & Prep for Backfill																
Backfill																
Backfill Cure																

16.15.3 Stope Sequencing

Sequencing of the mining of stope panels will be crucial to the efficient use of ventilation and equipment and for haulage of the material. Complete development and mining sequencing was not completed for this study. MDA recommends that a detailed development and mining schedule be completed. This schedule should be designed to provide a constant mill feed, optimize the equipment and labor, minimize ventilation requirements and re-handling of material, and maximize efficiency of the mining operation. Scheduling should be completed for a monthly plan over the life of the mine, with further detail recommended for the first two years of development and production.

16.16 Mining Equipment

SGM owns some underground mining equipment in a variety of conditions but will need to purchase most of the equipment to complete the development and production phases of the mine. For this preliminary economic assessment, all mobile equipment necessary for this project will be assumed to be purchased. Due to the small production size and mix of mechanized and labor-intensive non-mechanized mining, the quantity of equipment required is not great. Additionally, used equipment may be available at significant savings over new prices; however, MDA would recommend mostly new equipment for the main pieces of mining equipment. Table 16.8 summarizes the mining equipment assumed for the mine development and production.

Table 16.8 Underground Mining Equipment

Item	Totals
4 cy LHD	1
2.5 cy LHD	4
UG Truck 22 ton	2
Drill Jumbo - Single Boom	1
Booster Fan, 40 HP	1
Stope/Heading Fan	4
Slushers, 15HP	8
Scrapers, 36" wide	8
Jackleg, complete	15
Stopers, complete	15
Air Tuggers	8
Sump Pumps	1
Heading Pumps	4
Explosive Loaders	15
Safety & Consume	30
Cap Lamps	100
UG Flammables Storage	10
Pickup Trucks	2
Blast Monitoring	1
Misc.	1

16.17 Mine Services

16.17.1 Dewatering

Water will be directed by drainage ditches to the lowest point of the mine (the bottom of the SBA decline), where it naturally accumulates. Current inflow of water to the underground workings is minimal at approximately 2 gallons per minute ("gpm").

Dewatering of the mine will be accomplished by pumping water from a sump located at the bottom of the SBA decline, as is current practice. Water will be pumped via an existing electric powered, 13 hp sump pump to the decant area and water storage at 5WXC. There, an identical pump delivers water to the recently reconstructed water treatment plant, located underground at 1EXC.

Water will be pumped for underground use in drilling, etc., to the mill for use in processing, or to the permitted spray field for land application.

16.17.2 Compressed Air & Water Supply

Compressed air (120 psi) will be supplied by parallel piston compressors located on the surface adjacent to the mill. The compressor(s) will be housed in a sound-insulated compressor building. The compressor(s) will be sized according to the chart below. The compressor system should include an in-line air dryer; however, an air cooling unit should not be necessary. A single receiver of at least 1000 gallon capacity will be installed to supplement the receiver capacity of the air lines.

Water utilized in the mine will be supplied by 4in high-density polyethylene ("HDPE") from the dewatering system.

HDPE and grooved-end, schedule 80 (schedule 40 minimum) steel pipe with Victaulic-type fittings will be utilized. HDPE will be run in lengths of greater than 200ft. Steel be will run in short lengths, typically on production levels and in development where the fittings and pipe can be re-used. HDPE pipe is not run in these areas because HDPE is not efficiently installed when run in short lengths, and is less re-usable than steel pipe. Compressed air and water will be supplied as follows:

- Surface: 6in air, 4in water
  - HDPE run from compressor house to portal
  - Mill is supplied with 2in or less pipe
- Decline: 6in air, 4in water
  - HDPE utilized. Existing steel pipe needs replacement.
  - Valves/drops located at each level
- Production level: 4in air, 2in water
  - Grooved steel pipe with Victaulic-type fitting
  - Valves/drops located at each dump pocket
- Scram/Raise/Stope: Combination of steel pit and temporary flexible hose
  - Utilities run as the stope progresses vertically
  - 2in Steel pipe for both water and air run to the stopes.

Table 16.9 shows the estimated compressed air requirements.

Table 16.9 Compressed Air Requirements

Item	# Units	Multiplie	CFM/unit	Total
Jackleg Drill	8	5.16	191	986
Stoper Drill	8	5.16	159	820
Air Tugger	4	n/a	125	500
Subtotal				2306
Leakage/Contingency	25%			577
Totals				**2883**

16.17.3 Mine Electrical Distribution

The mine will require electrical power underground for the main ventilation fans, electric slushers and possibly electric tuggers (small hoists), auxiliary fans, pumps, and other uses.

Electricity is currently provided to the site by overhead, high-tension distribution lines at 13kV. A transformer reduces voltage to 4,160V. Electric power is provided to the underground workings at a 300 kVA skid-type transformer/substation located at the SBA decline portal. American Mustang SOOW 2/4 AWG cable delivers electricity at 480v from the skid transformer to the upper 50hp main mine ventilation fan, the underground water treatment plant, the pump at 5WXC, and a 480V/120V transformer which supplies lighting for the current tour operations. Electricity, at 4160V is also cabled to another 300kVA skid-type transformer/substation at 7WXC. The skid transformer at 7WXC supplies power for the lower 75hp main ventilation fan and the main dewatering pump (at the lowest portion of the decline).

For the proposed mining operations, additional cable and skid-type transformer/substations will be purchased. Distribution substations will be located on each mine level.

16.17.4 Underground Communications

Underground communications will be installed in all active working areas of the mine, consisting of a modern Leaky Feeder Radio (“LFR”) system. LFR systems allow communication via standard hand-held radio to line-of-sight cables hung on the ribs or roof of the mine. The system is boosted every 1,000ft by an amplifier. The system is carried to the surface, so that surface personnel can communicate easily and efficiently with underground personnel.

In addition, the mine will have hard-wired pager phones where practical and appropriate.

16.17.5 Ventilation

The amount of fresh air required to be delivered to the mine is a function of the amount of fresh air required to dilute the diesel particulate matter (“DPM”) being produced by the operation of diesel powered equipment underground, to evacuate other dust and particulate matter, and to supply the required air for human activity underground. Current MSHA regulations establish airflow quantity requirements based upon equipment engine models. To prepare this preliminary assessment, an average was based on likely engine models found by researching currently available used mining equipment in the size needed for this project. The ventilation requirements will need further study as the project proceeds.

The mine is currently ventilated by existing 50hp and 75hp axial vane fans, which function in series inside the 3ft-diameter steel ventilation duct. This duct hangs along the length of the SBA decline. These fans, when operated together, draw approximately 30,000 cfm down the SBA decline and exhaust air out through the 3ft-diameter duct. These fans are reversible, allowing for the reversing of the air-flow direction.

The calculated minimum amount of air to be delivered underground is about 70,000 cfm based on the equipment planned for the operation. Main mine fans will be located in the new North Star decline, which will access the 1200 level of the mine workings. These 75hp fans, mounted in a bulkhead in a run-around, will exhaust the entire mine workings through various raise connections. Air doors will be used to prevent air from circulating at this run-around. The main fans are expected to produce an additional 55,000 cfm at minimum. MDA suggests that ventilation professionals review the ventilation plans for the operation.

The auxiliary ventilation system consists of auxiliary fans and/or stoppings that pull or direct the fresh air out of the main mine air stream and into stopes, scrams, raises, or other development headings. One additional fan location has been identified during simulation. An auxiliary fan providing 10,000 cfm will be located at the northern terminus of the 900 Level, drawing air from the SBA decline, through 900-7-4 manway raise, onto the 900 level.

Stopes will be provided with flow-through ventilation. Temporary or permanent stoppings with regulators as needed will be installed at the terminus of each raise on the level above. By restricting the volume of the opening, the airflow through each stope will be regulated.

Auxiliary fans will be utilized to pull fresh air to each working face during development of levels, scrams, and raises. These fans will blow fresh air into ventilation tubing, which will be split off each level to the respective scram or raise in development.

17.0 RECOVERY METHODS

Allihies Engineering Incorporated ("Allihies") provided a comprehensive review of the metallurgical testing and submitted a report entitled "*A professional review of metallurgical testing analysis, processing, pilot and engineering studies on the Sutter Creek Gold Mine*" ("the Allihies Report") (Anderson, 2010) that is incorporated in Section 13 of this report.

In 2011, Allihies conducted a review of the proposed conceptual gold mill design done by Paul E. Danio & Associates and compared this with existing metallurgical studies to date. Allihies confirmed that the Paul E. Danio & Associates proposed conceptual preliminary design and the related preliminary economic estimates, focused largely on the specific samples provided to McClelland Laboratories, are now appropriate as a preliminary conceptual design and preliminary estimate based on the current data available. Allihies does not confirm or take responsibility for, or confirm, any past, current or future operations and any detailed designs. The proposed mill of the 2011 technical report was constructed, but after processing about 1,000 tons of low grade material it was determined that modifications to the existing mill would be required to achieve production goals. The suggested revisions are noted in section 17.6.

17.1 Metallurgical Testing Summary

Allihies provided a summary of the available information in its report (quoting Behre-Dolbear, 2007):

> "*Since 1989, samples from the Sutter Creek Gold Mine have been investigated at Hazen Research, Dawson Laboratories, and Kappes Cassidy, Inc. Both bench scale and pilot plant scale testing were conducted. The metallurgical response, as measured at each laboratory, was similar. Gravity recoveries, at relatively coarse grinds, typically ranged from 23 to 54 percent. In the most extreme example, gravity recoveries as high as 90 percent were achieved. Typical flotation recoveries (based on overall plant feed) contributed an additional 45 to 74 percent gold recovery. Overall recoveries were generally greater than 90 percent.*"

Allihies (quoting McPartland, 2009) summarized the metallurgy as follows:

> "*The Sutter Gold ore samples responded well to whole ore milling/gravity/concentration treatment, followed by bulk sulfide flotation treatment of the resulting gravity tailings, at 80%-100M and 150M feed sizes.*
>
> *It should be possible, using this processing scheme, to produce a small volume, high grade gravity cleaner concentrate suitable for smelting, and a larger volume, lower grade flotation concentrate suitable for offsite shipment and processing.*
>
> *Expected combined gold recoveries by milling/gravity/flotation treatment should be quite high.*
>
> *Recycle of a flotation scavenger concentrate to the rougher flotation feed may be effective in significantly increasing the flotation rougher concentrate grade, without increasing losses to flotation rougher tailings.*
>
> *Recycle of a flotation cleaner tailings (without regrind) to the rougher flotation feed may not be effective in significantly increasing the flotation cleaner concentrate grade.*"

17.2 Flowsheet Development

Based on the review of the available reports and conclusions contained therein, a mill flowsheet has been developed for a 210 tons per day (150 tons per day equivalent at seven days per week) gravity and flotation mill. Gravity concentration is required in the mill circuit as explained by McClelland (McPartland, 2009):

> *"Microscopic examination of the gravity cleaner concentrates produced by hand-panning revealed the presence of liberated (free milling) gold particles of up to approximately 10M in size. The particles had rough, irregular surfaces, and appeared to have been flattened out during grinding. The presence of liberated gold particles indicates good potential for producing a smeltable gravity gold concentrate from the Sutter Gold ore. The presence of gold particles as coarse as 10M in size indicates the importance of including gravity concentration in the circuit, as gold particles that coarse are unlikely to be recovered by flotation."*

17.2.1 Gravity and Flotation Flowsheet

The existing mill flowsheet is shown in Figure 17.1, while Figure 17.2 and Figure 17.3 are layouts of the mill. The mine trucks will deliver "ore grade" material to drive-over truck dump bins of approximately 400 tons live capacity. The use of drive-over bins will reduce noise at the mill site, which is important due to the close proximity of the towns of Sutter Creek and Amador City. This design also allows for the covered storage of two to three days of mine output and addresses or eliminates storm-water management issues. Feed and transfer belts will then deliver the run-of-mine material at 100% minus 6in to a single deck vibrating screen to remove minus 0.5in material ahead of crushing in a 12in by 24 to 36in jaw crusher set at 1in discharge (gape). Tons per Hour ("TPH"), as part of a mill audit conducted for SGM, has suggested adding a secondary crusher to the plant prior to screening the product. The screening and crushing section will be completely enclosed and fitted with a water mist dust suppression system to eliminate airborne particulates. The discharge of the jaw crusher will be transported to a 400-500-ton, live-capacity, crushed "ore" bin by covered conveyor belts, protruding from the side of the building on the northern side, further reducing storm water and dust control issues. The conveyor belt will be 80ft long in the first run at no greater than a 17° angle, transferring to a 90ft belt, also at no greater than a 17° angle.

A variable speed belt feeder will supply the minus 0.5in crushed material from the bottom of the crushed "ore" bin to a 5ft-diameter by 10ft-long rod mill. This mill, in a closed circuit, will discharge to an 8-mesh trash screen.

The ground material will flow by gravity to a centrifugal concentrator (which will have a 10 mesh feed screen) for the production of a rougher gold concentrate and rougher gravity tailing. The rougher gravity tailing will discharge to a sump to be pumped to a hydrocyclone. The oversize (>100 mesh) from the cyclone will flow to the rod mill, closing the grinding circuit. The overflow from the cyclone, at 80 percent passing 100 mesh, will flow to the rougher flotation cells.

After flotation, the final tailings will be pumped to a 50 to 60ft-diameter thickener for the reclamation of process water and for the production of mine hydraulic sand backfill. The thickener will provide for approximately one day storage of thickened tailings ready for direct placement as fill underground, or alternatively, thickened tailings will be filtered into cake for storage under cover for later use as fill, or permanently impounded in the dry tailings Surface Fill Unit.

Figure 17.1 Mill Process Flowsheet

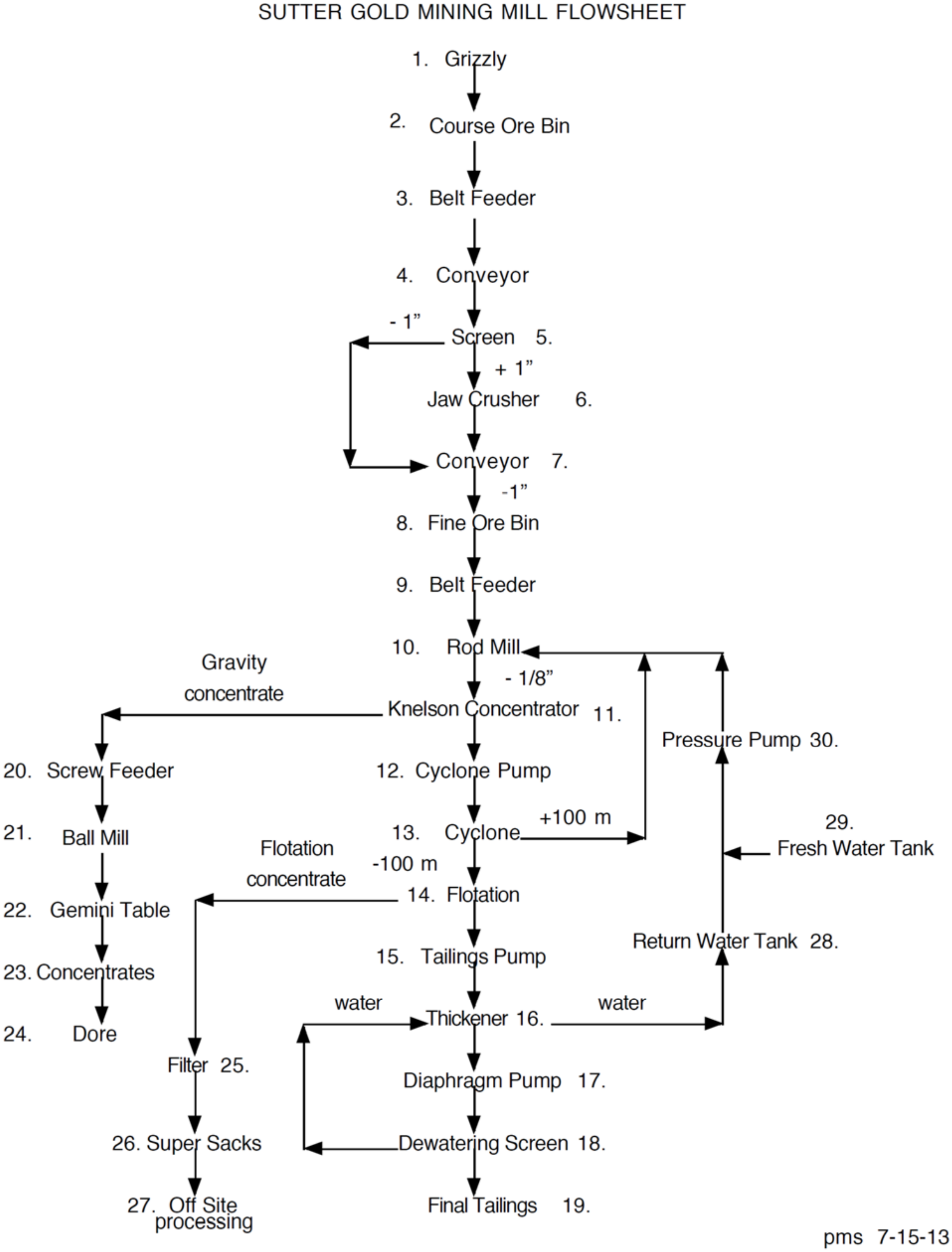


Figure 17.2 Mill Building – Elevations

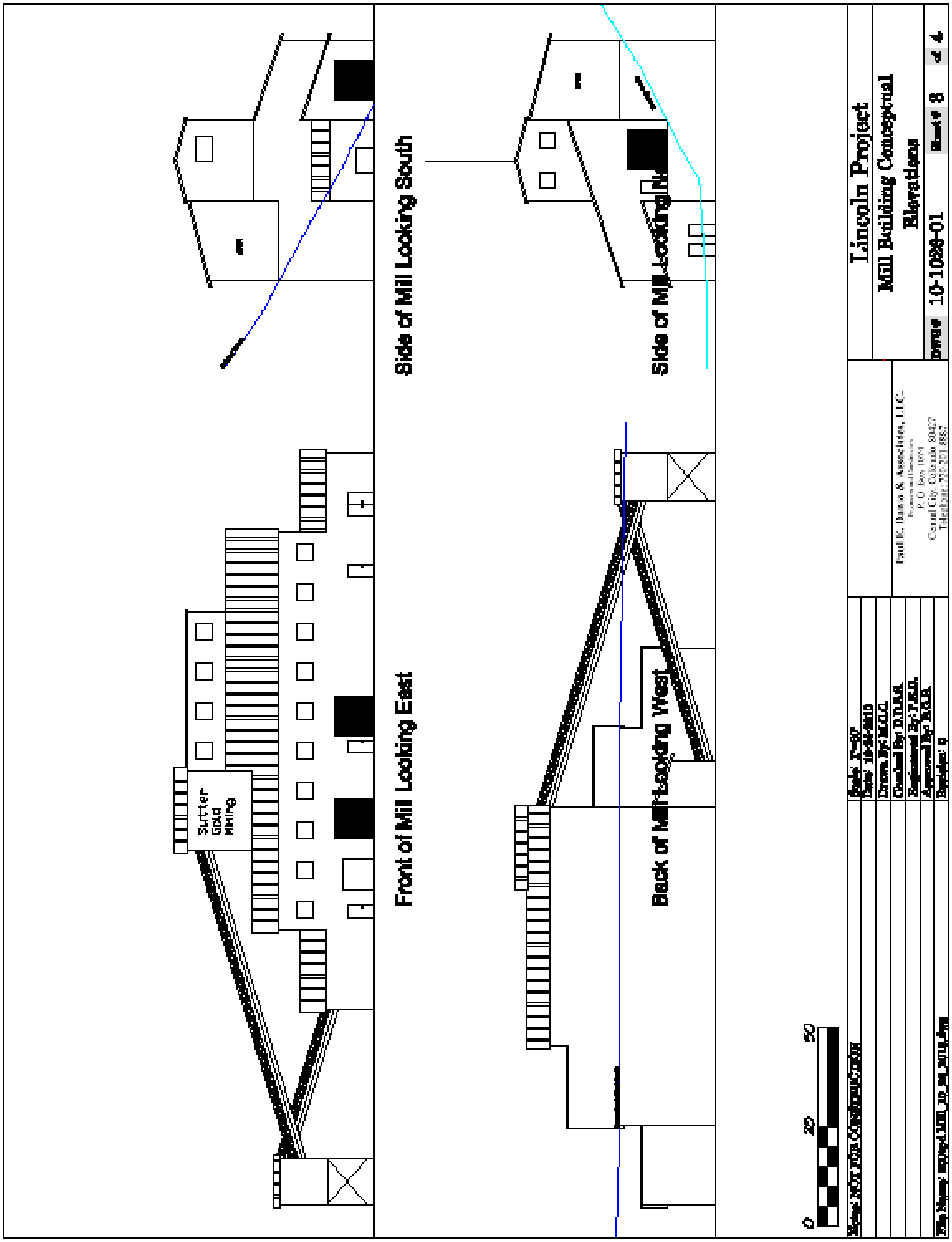
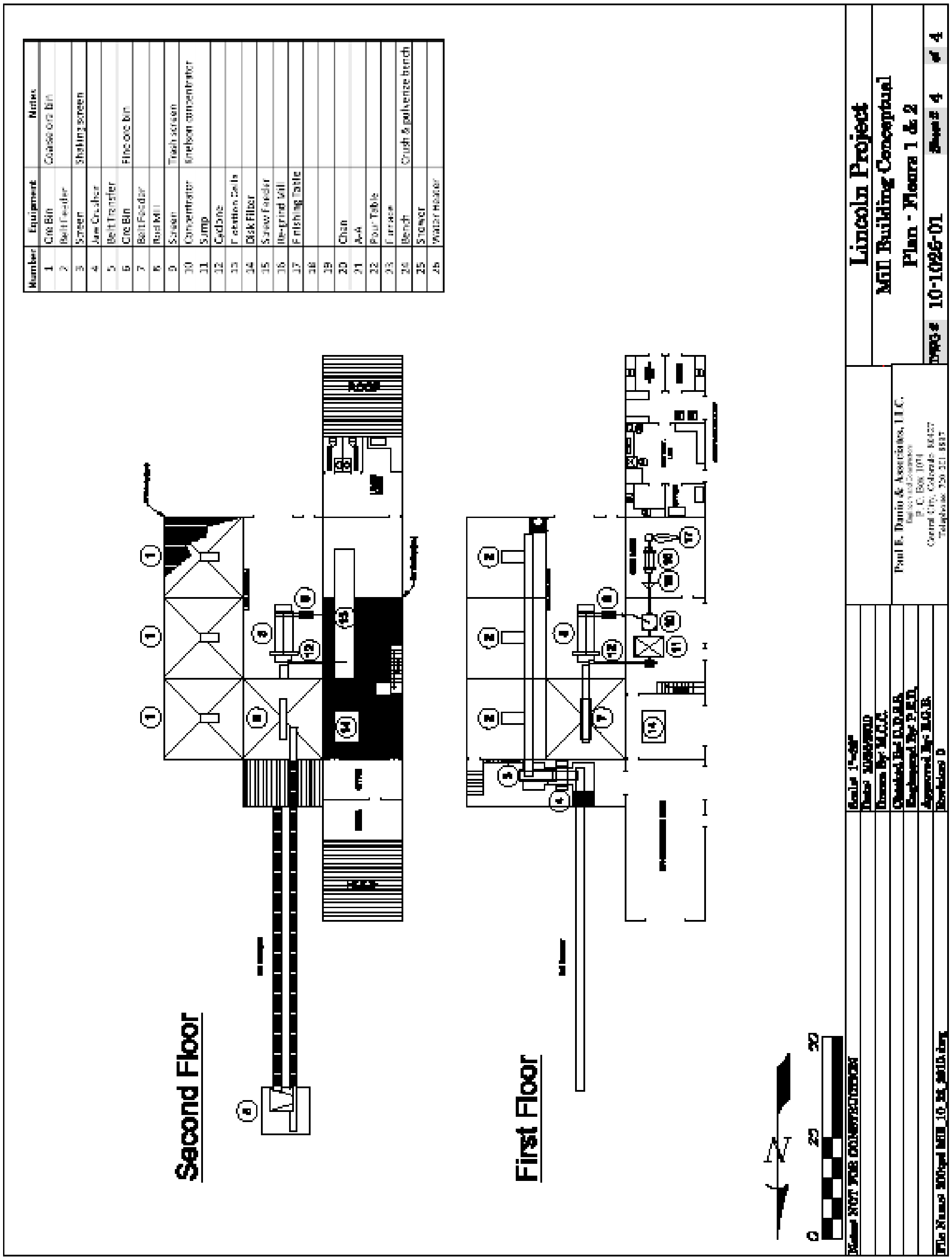


Figure 17.3 Mill Building - Floor Plans



17.3 Precious Metal Recovery

The rougher gravity and flotation concentrates produced will contain in excess of 90 percent of the precious metal contained in the whole ore. Specifically, tests performed by McClelland showed that gravity concentration combined with flotation recovered 97.9 percent of the gold contained in the Lincoln-Comet mineralized material (McPartland, 2009, pg.1). Each circuit is described in further detail in the appropriate section below. For the purpose of economic analysis, 96% recovery from processing is assumed, with 70% of total mill head gold recovered in the gravity circuit and 26% recovered in flotation. Concordant with, but conservative from McClelland's work, the criteria used for the PEA are as follows: 370:1 rougher gravity circuit concentration ratio and 54:1 flotation concentration ratio.

17.3.1 Gravity Circuit

The rougher concentrates from the Knelson concentrator will discharge to a dewatering screw feeder and be fed to a small (24in diameter by 36in long) regrind mill. The discharge from this mill will feed a finishing table for the production of a high-grade final gold concentrate. It is expected that the mill will produce approximately 10 pounds of gravity final product per day. This high-grade concentrate is amenable to direct smelting and will be processed onsite in the planned metallurgical laboratory into doré bars for sale to a refinery. The tailing product from the finishing table will be returned to the main grinding circuit sump for recycling or, if grade is sufficient, to the flotation concentrate.

McClelland showed that between 72.4 and 82.1 percent of the gold in the whole ore was recovered by gravity concentration (McPartland, 2009). The gold recovered was as coarse as 10M. For the purpose of economic analysis, 70% of gold reports to the gravity circuit concentrate.

17.3.2 Flotation Circuit

The flotation circuit will consist of roughing and one-stage cleaner flotation. The flotation cells will be Denver #21 Sub-A or equivalent cells, purchased on the used market. The circuit is designed to recover fine gold that the gravity circuit misses and to eliminate arsenic in the final tailings to a level below that consistent with the California Solid Waste Requirements for Class B solid wastes. Cleaner tailings may be returned to the grinding circuit for improved separation of middlings particles. This is not shown on the flowsheet, although it should be considered during initial testing as the mill is commissioned.

The Lincoln-Comet ores contain arsenic as arsenopyrite, with minor amounts of pyrite consisting of approximately 2.1 percent or less of the total material (McPartland, 2009). The pyrite will be floated with amyl-xanthate after activation with copper sulfate at a ph of 7 to 8. Additionally, a collector (Aerofloat 208 or 3477 or equivalent) will be used to optimize free gold recovery. Pine oil will be used as the frother. The copper sulfate should be added directly to the rod mill along with any pH modification, as necessary. The xanthate should also be added to the rod mill, eliminating the need for a conditioner tank. The frother and Aero promoter should be added directly to the feed to the rougher flotation circuit. Tailings from the flotation assayed 377 ppm arsenic at Kappes, Cassiday & Associates in 1997, which is below the 500 ppm limit for Class B solid waste imposed by the State of California. This needs to be monitored closely, and alternative plans need to be developed should the arsenic levels exceed the limit.

Flotation concentrate will be transferred directly to a 4ft, two-disk rotary disk filter, which will dry the flotation concentrate to approximately 12% moisture (water weight). Thickening of the product prior to drying is considered unnecessary at this plant size. This product will discharge from the disk filter into sack-type containers, where it will be stored on-site until at least one truckload has been accumulated. The approximately 22-ton shipment will be loaded and transported for further processing by a second party. The flotation concentrate will be processed by the second party for a fee of 15% of the gold in the concentrate.

17.3.3 Gravity Concentrate Grade

Gravity concentrates produced by centrifugally enhanced gravity concentration varied between 49.738 and 72.874oz Au/ton for a Sutter Gold Bulk "Ore Grade" Sample (H. C. Osborne and Associates, 2009; McPartland, 2009 included herein by reference). A confirmatory test on drill cores showed final cleaner gravity concentrates of 863.351 – 1195.858oz Au/ton (McPartland, 2009). As stated in this report, the grade of concentrates produced in the proposed mill will vary from these values depending on the head grade mined:

> "*It should be noted that gold recoveries obtained during laboratory gravity concentration testing using centrifugal concentrators are normally significantly higher than would be expected by gravity concentration in a commercial processing plant. The recoveries presented here do not include any discounts for possible gold losses during subsequent concentrate processing for gold recovery*."

17.4 Mill Production Schedule

The mill will operate 24 hours per day, five planned days per week basis. This allows for expansion of mine production at increased gold prices and as with modern milling practice, will not suffer from historic problems of gold loss with mill shutdown and start-up. This schedule also allows for ample maintenance time.

Nominally, the mill will run Monday through Friday, with weekends reserved for scheduled maintenance when needed. Plant capacity is designed to accommodate a minimum of 150 tpd on a seven day week, or 210 tpd per five day week.

17.5 Products

Two products will be created by the proposed mine. As indicated by the 2009 metallurgical testing (McPartland, 2009), the greater proportion of gold will be recovered in the gravity circuit, estimated at 84% (H. C. Osborne and Associates, 2009). This product will be extremely high gold grade, and will be smelted on site to produce a doré. The remaining recoverable gold will be recovered in the flotation circuit and report to the bulk flotation concentrate.

17.5.1 Doré

Bullion, in the form of doré, will be sold to a refiner. Many refiners of this type of product are active and operating in the United States. This product will be generated at less than one pound per day and can be shipped via U.S. Postal Service or secure private courier. Estimated payment for this product is conservatively included in the Preliminary Economic Analysis at 99.75% of contained gold.

Johnson-Matthey was contacted regarding the potential sale of this product. The following information was provided. For gold doré, as would be produced by SGM's Lincoln Mine project, the treatment charge is $1.00 per ounce, with a minimum charge of $1,000.00 ($1 per ounce per thousand-ounce lot). Payment of gold content is 99.75% of spot price (London PM) on the day of settlement, usually about 15 business days from receipt of product.

17.5.2 Flotation Concentrate

Flotation concentrate will be dried, bagged, and transported via truck to either a pressure-oxidation treatment facility in Nevada or a pyrometallurgical smelter elsewhere.

Newmont Mining Corp. indicated interest in purchasing the flotation concentrate, FOB the Twin Creeks facility, at approximately 85% of the contained gold at the 30 day LME PM spot price (personal communication, 2010). For the purpose of economic assessment, payment of 85% of contained gold is used.

17.6 2012-2014 Mill Construction and Operation

SGM constructed a 210 ton per day processing mill on the Lincoln Mine property and completed about 3,300ft of underground development during the period of late 2012 through early 2014. A total of about $22 million was spent on the project during this period. Approximately 3,100 tons of low grade material have been stockpiled near the mill. About 1,000 tons of low-grade material was processed in the mill during 2013-2014, but the rod mill produced excessive fines and the rate of processing material through the grinding circuit was much lower than expected. Gold recovery was very low due to the excessive fines. SGM has developed plans to mitigate these problems which are incorporated in this PEA study.

SGM retained TPH to conduct a mill optimization study and cost estimate for modifications of its Sutter Creek facility following various process upsets. A site visit was conducted March 27-28, 2014. The following areas for improvement were identified as a result of the site visit.

- Tails handling has to be rebuilt using a new slimes thickener and filter press at the sand building. The old thickener would be decommissioned and used as a process water storage tank.

- Secondary crushing has to be added. Final screen product will be 100% passing 1/2". Construction of the new crushing system will be inside a new building on the north side of the mill. The existing jaw crusher will be relocated to the new building.

- Process modifications to gravity (coarse gold jig, Knelson relocation, Wilfley table) and floatation (float flow reorganization, conditioning tank) are required to improve gold recovery.

- A gold room expansion is needed to accommodate installation of an existing 6x16 Wilfley table, automate processing, and provide increased security.

- Instrumentation and a programmable logic controller is an essential addition to the mill to maintain process control and determine metallurgical balances.

Table 17.1 shows the cost estimate for the retrofit and repairs recommended by TPH.

Table 17.1 Estimated Mill Repair Cost

Item	Totals (x1000)
Revised Grinding Circuit	$612.8
Gold Recovery Improvements	$17.2
Tailings Management	$477.5
Process Control and Instrumentation	$452.5
Gold Room Improvements	$234.5
Laboratory Upgrade	$46.3
Site Water Balance	$12.3
Cost Estimate	$1,853.0

18.0 SITE FACILITIES AND INFRASTRUCTURE

Completed site facility construction during 2012-2014 includes most of the infrastructure required for an operating mine, including an access road upgrade, waste rock liner, processing mill, offices and shop, site preparation, sewer, fire water, potable water, and power distribution.

18.1 Office Building/Dry/Shop/Warehouse

An existing building, measuring approximately 40ft by 160ft has been remodeled to house the mine office building, the mine dry, and the shop/warehouse (Appendix C gives a list of pertinent equipment and buildings). The building is large enough to house all that would be required for mine support.

All mine maintenance will be performed at the surface shop. Loaders, trucks, drills, and ancillary equipment will be brought to the surface for maintenance and repair.

18.2 Processing Mill and Backfill

SGM has constructed a 210 tpd mill based on the Danio design noted in the 2011 PEA study for the project. The mill houses the mill circuit, assay and metallurgical lab and offices for each, and the main mine air compressors. Conforming to Amador County building standards the building is approximately 60ft wide by 140ft long by 50ft high and is constructed of steel and concrete. The processing plant includes "ore" storage, crushing, grinding, flotation and gravity recovery circuit, concentrate drying, assay and metallurgical lab, secure gold room, and office and restrooms for the mill staff. The following is an itemized listing of the equipment installed in the Lincoln Mine mill as depicted on the mill flowsheet. The equipment is listed numerically and matches the numbering on the flowsheet shown in Figure 17.1. The mill equipment is comprised of a mixture of new and refurbished equipment.

1. Grizzly: The grizzly is over three separate course ore bins, each having a capacity of about 50tons for a total of 150 tons. This grizzly has 8" spacing.

2. Course Ore Bins: There are three separate course ore bins, each having its own feeder which transports the ore to the transverse belt.

3. Belt Feeders: Each course ore bin has a belt feeder under it, these feeders are 10' long and 4'wide and move the course ore out of the course ore bins onto the transverse belt.

4. Transverse Belt: This 60' by 24" conveyor belt moves the ore from the feeders to the vibrating screen.

5. Vibrating Screen: This is a 4' by 8' screen with a 1" screen installed; this screens the ore at 1" letting the -1" pass through the screen and on to the conveyor belt to the fine ore bin, circumventing the jaw crusher.

6. Jaw Crusher: This is a Pioneer 10" by 36" jaw crusher set at the minimum setting of 1".

7. Fine Ore Conveyors: There are two, 24" by 100', inclined conveyors that take the -1" ore up to the fine ore bin.

8. Fine Ore Bin: The fine ore bin has a capacity of 400 tons.

9. Rod Mill Feeder Belt: A 19' by 24" belt feeds the rod mill from the fine ore bin; this is a variable-speed belt so as to regulate the feed to the rod mill.

10. Rod Mill: This is a Marcy 5' by 12' steel-lined rod mill with a tube feed. At the discharge end there is trommel with 1/8" openings so as to give the correct feed to the Knelson concentrator.

11. Knelson Concentrator: This is a Model KC-XD20 concentrator set to handle the 1/8" rod mill discharge.

12. Knelson Tailings Cyclone Pump: This pump, a Galigher 4" vertical sand pump, pumps the Knelson tailings to the cyclone.

13. Krebs Cyclone: This 10" cyclone is set above the rod mill so that the oversize, +100 mesh, will drop back into the rod mill and the undersize, -100 mesh will report to the flotation circuit, target grind being 80 % - 100 mesh

14. Flotation Circuit: The flotation circuit consists of two # 24 Denver Sub A cells; one used as a rougher and one used as a cleaner. There are then four # 24 Denver DR cells used for the scavengers. At the proposed through put rate of two hundred tons per day there will be ample float time.

15. Tailings Pump: This a new Wilfley sand pump, model 1-10 AG, that pumps the tailings up 180' in elevation over a distance of 500 ' to the tailings thickener.

16. Tailings Thickener: The thickener is a 60' diameter Wemco, rake thickener, the tank is made of concrete.

17. Thickener Diaphragm Pump: This is a Denver 6" duplex diaphragm pump that pumps the thickened tailings to the dewatering screen.

18. Dewatering Screen: This is another new piece of equipment, a Deister model BFO-1510 DW with a 140-mesh polyurethane screen.

19. Final Tailings: Approximately 50% of the tailings will be dewatered with the 140-mesh screen down to approximately 15 % moisture. With the way the screen beds itself much of the material down to 400 or 500 mesh is screened out. These final tailings will be hauled off site by a local vendor, the other 50% of the tailings is to be pumped underground and used as back fill.

20. Screw Feeder: The Knelson concentrates are fed by gravity into a 7" by 10' long screw feeder having a storage capacity of approximately 1500 pounds. This is sufficient capacity to allow for running the finishing gravity circuit for no more than one day per week.

21. Regrind Ball Mill: The screw feeder feeds a Marcy 2’ by 3’ ball mill that is open circuit with over flow transported by launder to the Gemini table.

22. Gemini Table: This is a model CE - GT100 table; it takes the flow from the regrind mill and upgrades the Knelson concentrates producing 5 products.

23. Number One Knelson Product (Concentrates): This product has shown from test work to assay as high as 1,200 opt and will be fired and poured into doré on site.

24. Doré: Sutter plans to fire and pour doré bars over one day shift. The process will be to start early, fire, pour, sample and ship all on the same day, thereby reducing security risks.

25. Flotation Concentrate Filter: This filter is a Denver 6’ by 6 disc vacuum filter capable of dewatering the concentrates to about 15% moisture.

26. Super Sacks: The filtered flotation concentrate will drop straight into super sacks mounted below the filter. This equipment will have a scale so that bags can be filled to the proper weight and ready them for shipment.

27. Off-site Processing: Sutter is in discussions with several parties in relation to flotation concentrate off take. Longer term, the option is to install a cyanide plant to treat concentrates.

28. Return Water Tank: This 10,000-gallon tank is set up near the thickener to take the thickener over flow for mill process water.

29. Fresh Water Tank: This tank takes water pumped from underground to use as process make-up water in the mill.

30. Mill Water Pressure Pump: This pump is used to provide a steady head to the mill and is connected into both the return water tank and the fresh water tank.

The backfill material will be prepared, handled and delivered from the Sand Plant Pad located directly up the hill from the processing plant on the south side of the paved access road. The incoming tails will be sized if necessary, with undersize material dried, stacked and loaded into trucks and trailers for transport to the SFU. The backfill plant will utilize one or two large thickener tanks (depending on equipment availability), which will also provide as surge capacity, and an 8ft diameter disk filter. The plant area features a large laydown and parking location for several trailers, and a pole-barn style roof over the sand stack.

18.3 Processing Water Supply

Approximately 180,000 gallons per month of make-up water will be required for the mill operations. In addition to return water from the underground workings, a water main from the town of Sutter Creek is located on site. There is more than sufficient water available for the proposed operations. Water is sold by the Amador Water Agency at $3.60 per unit of 748 gallons, plus a monthly service charge of $252.25

for a 2in service supply line (2011 Rates). Typically, these rates increase at 7% per annum, and currently average just less than $1,300 per month.

Observations from the SBA decline indicate that the mine would not provide sufficient water to meet process water makeup requirements. However, decant water from the backfilling process would be returned to the mill.

An alternative to this source of process water would be the nearby North Star mine workings or the Talisman mine workings. A well could be drilled to intercept these workings and water, if no treatment was necessary, could be supplied to the mill at substantial savings.

18.4 Surface Fill Unit and Waste Storage

The Surface Fill Unit ("SFU") will impound undersize and whole mill tailings in a location to the east of the mine site (Swift Parcel). Dewatered tailings will be transported via 26-ton transfer dump trucks and trailers from the mill to the SFU, where they will be dumped/stacked and contoured.

Early cost estimation for a leak-detection-capable, double containment slurry line from the mill to the SFU indicated an insupportably high capital investment for the five year mine life. Truck transportation will be utilized to transport the tailings to the SFU, possibly requiring an amendment to the current Conditional Use Permit which was issued by the Amador County Planning Department in 1998. A complete study of the costs associated with the SFU was completed by Golder Associates in April 2010, and reported by Haskell (2010). These costs are integrated into the Preliminary Economic Analysis.

The Waste Rock Pile ("WRP") will be located very near the portal and will fill the small valley in front of the portal. Golder Associates completed a design and cost estimation for the WRP in March 2010. The WRP will feature geosynthetic clay and a double membrane liner system, with associated drainage controls and water diversions.

The area for the WRP is very small and must adhere to a permitted elevation limit of 1,160 ft., imposing significant restriction on volume. SGM would like to exceed the 1,160 ft. limit, but that may require an amendment to the Conditional Use Permit. As currently permitted, waste rock will effectively be stored at this location, up to the 1,150 contour. Additional material will be stored between the 1,150 and 1,160ft contours in quantities less than 5,000 tons, with the majority of the waste rock being purchased and hauled away for use as aggregate product. This is consistent with the Conditional Use Permit, which states: "...Permittee shall make construction rock available to the County...construction rock may be removed from the site..." (Amador County Conditional Use Permit #UP-97; 7-4, Mitigation Measure 9, issued September, 1998). The removal of construction rock by any purchaser was assumed to be cost neutral.

18.5 Diesel Fuel Supply and Distribution

Diesel fuel will be stored and supplied on the surface in a contained fueling and fuel storage location. Diesel fuel will be trucked to the property by an outside provider and transferred to the storage facility. The double-walled storage tanks and berms around the fueling location will prevent accidental releases and will comply with all Spill Prevention Containment and Control measures.

With the small amount of equipment employed underground and short mine life, the high cost of an underground fueling station cannot be justified and is not considered necessary. Equipment will be fueled at the beginning or end of each shift; no regular mid-shift refueling is anticipated.

18.6 Surface Materials and Equipment Storage

Surface materials and equipment storage will be located on the mill pad area and on the north side of Stringbean Alley. Material, such as timber, rock bolts, ventilation ducting, etc., will be stored with equipment, such as ventilation fans, slushers and tuggers, as space allows. The climate is not expected to adversely impact the storage of materials on the surface.

19.0 MARKET STUDIES AND CONTRACTS

Market studies have not been carried out because the anticipated product (gold doré) will be widely salable to metal refiners. No contracts have been established for the sale of gold doré or flotation concentrates.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Considerations – Reclamation, Remediation, and Bond Posting

Section 20.1 was prepared by David Cochrane of SGM and approved by Steve Lofholm of Golder Associates Inc.

SGM's Lincoln project is subject to reclamation, remediation, and financial assurance requirements, including bond posting (under applicable state statutes), regulations, and local requirements. These requirements primarily originate from two separate, but overlapping sets of requirements for mining projects like the Lincoln Mine project.

The Surface Mining and Reclamation Act of 1975 ("SMARA"), codified in Chapter 9, Division 2 of the Public Resources Code ("PRC"), requires the State Mining and Geology Board to adopt State policy for the reclamation of mined lands and the conservation of mineral resources. These policies are found in the California Code of Regulations ("CCR"), Title 14, Division 2, Chapter 8, Subchapter 1. The Office of Mine Reclamation ("OMR") administers these regulations through local Lead Agencies, such as the Amador County Planning Department ("ACPD") in the case of SGM's Lincoln Mine project.

The CCR, Title 27, Chapter 7, Subchapter 1, Article 1, Section 22470 *et seq*. contain requirements regulating the discharge of waste to land including those for reclamation (closure and post-closure), remediation (corrective action), and associated financial assurances for mining waste units. The State Water Resources Control Board ("SWRCB") administers these requirements through nine Regional Water Quality Control Boards ("RWQCB").

20.1.1 SMARA Requirements

SMARA requires, among other things, reclamation of all surface mining areas, including surface disturbances associated with underground mines like the Lincoln project. SMARA reclamation regulations (CCR, Title 14, Division 2, Chapter 8, Subchapter 1) assure that adverse environmental impacts are minimized and that mined lands are reclaimed to a usable condition. For each county, a Lead Agency is established to administer and enforce SMARA requirements with oversight and support of the OMR.

These regulations require that each mine operator prepare a Reclamation Plan for their mine and obtain approval of the plan from the Lead Agency and the OMR. The regulations also require that each operator prepare a Financial Assurance Cost Estimate ("FACE") to implement the Reclamation Plan. The FACE estimates the costs for the Lead Agency or OMR to implement the Reclamation Plan should they be required to do so. Operators report annually on their mining activities and update their FACE. The Lead Agency, in turn, conducts an annual inspection of the mine and reviews the updated FACE, ultimately making a finding of adequacy. The Lead Agency reports the results of their inspection and review of the FACE to OMR, which has 45 days to comment. The absence of comments from the OMR is deemed as concurrence by the Lead Agency with respect to adequacy of the FACE. The ACPD is the Lead Agency for SMARA in Amador County, including SGM's Lincoln Mine project.

SGM filed a Revised Stage II Reclamation Plan for the Lincoln project ("Reclamation Plan") on February 10, 1999. The ACPD approved SGM's Reclamation Plan on September 21, 1999. The Reclamation Plan covers an area of approximately 287 acres of potential surface disturbance related to SGM's permitted Lincoln project underground gold mine. The area includes: a) the western surface fill unit, dewatering plant, slurry pipeline, and roadways; b) portal terraces, existing waste rock pile, and related land within the mill site complex; and, c) an air shaft with access from a pre-mining access road. The slurry pipeline will most likely not be utilized but will be replaced by already included road disturbances. The Reclamation Plan anticipates that an Amendment will be required to add one or more previously identified surface fill units in the eastern portion of the area as currently proposed. The Reclamation Plan area includes an encroachment to former State Highway 49 (now a county road designated Old Highway 49) and 19 acres of the mill site/office complex area; the existing road will remain, and it is anticipated that the area will remain graded and landscaped to serve post-mining uses. Further, it anticipates the planned office, visitor's center, mill, shop, and warehouse building and other structures permitted in this area will remain and serve post-mining uses and re-activation of mining at a later time should conditions allow.

The approved Reclamation Plan for the Lincoln Mine project includes a range of reclamation activities to assure environmental impacts are minimized and mined lands are reclaimed to a usable condition. Specific reclamation steps include: a) re-contouring of cut-and-fill slopes; b) stabilization of mined slopes, waste dumps, tailings, road cuts, and other excavations and embankments; c) rehabilitation of pre-mining drainages affected by the operation; d) removal, disposal, or utilization of residual equipment, structures, refuse, etc.; e) protective measures to secure and minimize precipitous slopes, pits, shafts, or other hazards; f) control of contaminant such as fuels, lubricants, chemicals, etc.; g) protective measures against contamination of surface and groundwater; h) treatment of streambeds and banks to control erosion and sedimentation; and i) re-establishment of vegetation and aquatic life habitats.

According to records reviewed during this evaluation, SGM has completed a substantial portion of their obligations for existing disturbed areas under the Reclamation Plan. SGM will need to amend the Lincoln project Reclamation Plan to incorporate the planned and permitted but not yet constructed eastern surface fill unit, etc. (Behre Dolbear, 2007).

20.1.2 SWRCB Requirements

Pursuant to the CCR, Title 27, Chapter 7, Subchapter 1, Article 1, Section 22470 *et seq*. regulating the discharge of waste to land including mining waste units, the SWRCB working with the appropriate RWQCB administers and enforces reclamation (closure and post-closure), remediation (corrective action), and associated financial assurances for mine waste units. These requirements for mine waste units include preparation of closure and post-closure plans, which are then approved by the RQWCB; preparation of closure and post-closure cost estimates subject to review and approval by the RWQCB; periodic updates to closure and post-closure plans including cost estimates and providing financial assurances, satisfactory to the RWQCB, of the availability of funds sufficient to assure implementation of the closure; and post-closure plans by a third party, if required to do so by the RWQCB.

CCR, Title 27, Chapter 7, Subchapter 1, Article 1, Section 2580 (f) also includes requirements for remediation (corrective action) funding or financial assurances; however, these requirements do not apply to mine waste units. Regardless, this does not preclude the RWQCB (under authority other than this Section) from requiring financial assurance for a known or reasonably foreseeable release at mining waste units. In general, the RWQCB administers these and other requirements on mine operators or "dischargers" through the issuance or permits know as Waste Discharge Requirements ("WDR").

The Central Valley RWQCB ("CVRWQCB") is responsible for administering and enforcing these requirements at SGM's Lincoln project. In 2007, the CVRWQCB issued WDR Order No. R5-2007-0006 for the Lincoln project, including the construction of waste piles and expanded mining operations and in so doing, rescinded previously issued WDR Order No. R5-2005-0164. This permit allows SGM to discharge mine waste to three planned mine waste units: the WRP for storage and disposal of development or construction rock, and the SFU and underground workings, both for disposal of mill tailings. WDR Order No. R5-2007-0006 specifies the provisions and conditions that SGM must satisfy, including closure and post-closure (reclamation) requirements and corrective action (remediation) requirements for the Lincoln Mine project. WDR Order No. R5-2007-006 also specifies financial assurance requirements for closure and post-closure of the SFU, WRP, underground workings and mill as well as corrective action for a reasonably foreseeable release from the WRP, SFU, and underground workings.

20.1.3 Financial Assurance Requirements, Including Bond Posting

SMARA offers a range of financial assurance mechanisms to mine operators to demonstrate financial responsibility for reclamation of mine sites. These include surety bonds, certificates of deposit, letters of credit, pledges of revenue, and budget set asides. Depending on specific circumstances, some or all of these may be available to a particular mine operator.

For the Lincoln project, SGM has established a continuous (renews annually) certificate of deposit currently for $27,000 (ACPD, 2010a). This is sufficient to cover the current FACE of $26,660 (SGM, 2010a). ACPD has determined that this estimate is adequate (ACPD, 2010b) and received no comments from the OMR on this determination and the FACE, which were due by November 29, 2010.

When SGM amends their Reclamation Plan to include the eastern SFU, remove the western one, and make other changes, SGM will need to update their FACE and demonstrate responsibility for the nature and scope of reclamation steps to the satisfaction of the ACPD and the OMR (Behre Dolbear, 2007).

Similar to SMARA, the SWRCB/WDR financial assurance requirements also offer a similar range of financial assurance mechanisms that may be available to dischargers (mine operators), dependent on specific circumstances.

In 2005, SGM retained Golder to estimate the costs for closure and post-closure maintenance for full build-out of the SFU. In July/August 2007, Golder (Golder 2007a) updated the 2005 estimates for the SFU to account for proposed revisions to the mine operation. Golder (2007a) also prepared estimates of the anticipated closure and post-closure costs for the WRP, mill, and underground mine workings. In addition, Golder prepared cost estimates for implementing and completing corrective action that could be required in the event of a "reasonably foreseeable" release from the SFU, WRP, mill, or underground workings.

In 2010, SGM retained Golder to prepare construction-level design documents for the WRP and the initial phase (5-year/4.5 acre operation) of the SFU. The CVRWQCB approved these designs in August 2010 (CVRWQCB, 2010). Golder's services also included preparation of updated closure and post-closure cost estimates for the WRP and the SFU, including both the initial phase and the complete build-out (24 acres). Table 20.1 summarizes the most current WDR cost estimates for the Lincoln Mine project.

Table 20.1 Summary of SWRCB/WDR Financial Assurance Cost Estimates for the Lincoln Mine Project

Waste Management Unit	Closure Cost Estimate	Postclosure Cost Estimate	Corrective Action Cost Estimate
Surface Fill Unit[1]	$603,660	$962,745	$943,600
Waste Rock Pile	$260,026	Unit will be clean-closed	$547,300
Underground Workings	$48,163	$1,536,218	$113,256
Mill Complex[2]	$70,000	$0	$0

[1]Initial development phase (5-year capacity/4.5 acres)

[2]Mill will be decomminssioned after closure and converted to a tourist attraction.

Source: Golder 2007a excepting SFU and WRP closure and postclosure costs (Golder 2010)

These costs are preliminary estimates, and the actual amounts of financial assurance that must be provided by SGM will be negotiated with the appropriate government agencies. Currently, the CVRWQCB does not require demonstration of financial assurances for these cost estimates as these project elements have not been constructed. Demonstrations of financial assurances are required prior to placement of waste in the respective mining waste units (WDR Order no. R5-2007-0006). Also, the timing of corrective action costs is uncertain. These could be incurred at any time during the life of the operation or may never be incurred (Behre Dolbear, 2007).

21.0 CAPITAL AND OPERATING COSTS ESTIMATE

21.1 Capital Cost Estimate

Estimated capital costs, including nine months of pre-production development, totals $12,474,700, and is summarized in Table 21.1. The details of the capital cost estimate are shown in the subsections that follow.

Table 21.1 Estimated Capital Cost

Item	Total (x1000)
Mine Equipment Capital	$3,635.4
Process Plant Modifications	$2,326.3
Pre-production Development	$1,797.1
Pre-production Labor	$4,716.0
Totals	$12,474.7

21.1.1 Mining Equipment

The estimated cost of mine equipment is shown in Table 21.2, and is based on mostly used equipment as desired by SGM.

Table 21.2 Estimated Mine Equipment Cost ($000's)

Item	# New	# Used	Totals	$/unit new	$/unit used	Totals
4 cy LHD		1	1		$300.0	$300.0
2.5 cy LHD	0	4	4	$330.0	$245.0	$980.0
UG Truck 22 ton		2	2	$600.0	$342.5	$685.0
Drill Jumbo - Single Boom	1		1	$440.0		$440.0
Booster Fan, 40 HP		1	1		$30.0	$30.0
Stope/Heading Fan		4	4		$5.3	$21.4
Slushers, 15HP		8	8		$22.5	$180.0
Scrapers, 36" wide		8	8		$1.5	$12.0
Jackleg, complete		15	15		$6.0	$90.1
Stopers, complete		15	15		$7.1	$106.2
Air Tuggers		8	8		$6.8	$54.0
Sump Pumps		1	1		$20.0	$20.0
Heading Pumps		4	4		$5.0	$20.0
Explosive Loaders		15	15		$0.5	$8.1
Safety & Consume		30	30		$5.6	$168.2
Cap Lamps		100	100		$0.4	$35.0
UG Flammables Storage		10	10		$0.5	$5.0
Pickup Trucks			2		$15.0	$30.0
Blast Monitoring			1		$10.0	$20.0
Misc.			1		$100.0	$100.0
Subtotal Mine Equipment						$3,304.9
Contingency						$330.5
Total Mine Equipment						$3,635.4

21.1.2 Pre-Production Development

Pre-production development entails all development work completed to access and begin to mine the number of stopes required for year 1. This would include additional decline footage, level installation, dump pocket/chute installation, scram and raise development, and ventilation installation for a total of 13 stope panels. Pre-production development is estimated to cost about $1,797,100 without labor costs; non-labor costs are summarized in Table 21.3.

Table 21.3 Pre-Production Capital Development ($000's)

Item	Drift Size	Length (ft)	Cost/unit*	Item
Main Decline	12x15	391	$645	$252.1
Level Decline	8x8	409.5	$301	$123.2
Level	8x8	2105	$301	$633.3
Level X-Cuts	8x8	954.5	$301	$287.2
Decline X-Cuts	12x15	125	$645	$80.6
Scram in Ore	3x8	516.5	$57	$58.7
Orepass in Ore	3x8	397	$96	$76.3
Manway in Ore	3x8	791	$96	$152.1
Orepass in Waste	3x8	227	$154	$35.0
Manway in Waste	3x8	121	$154	$18.7
Scram in Waste	3x8	62	$76	$4.7
Main Fan Installation				
Chutes (Decline)		3	$25,000	$75.0
Totals				**$1,797.1**
Note: Not Including Labor				

21.1.3 Processing Capital Cost Estimate

Mill retrofit and repairs recommended by TPH were summarized in Section 17.6. In addition, other vendors made estimates for improved dust collection and mill automation. The estimated total cost required to correct mill issues is shown in Table 21.4.

Table 21.4 Process Plant Cost ($000's)

Task Cost	Estimate ($000's)
Revised Grinding Circuit	$612.8
Gold Recovery Improvements	$17.2
Tailings Management	$477.5
Process Control and Instrument	$452.5
Gold Room Improvements	$234.5
Laboratory Upgrade	$46.3
Site Water Balance	$12.3
Subtotal MPH Estimate	$1,853.0
Improved dust suppression	$71.4
Mill Automation	$190.5
Subtotal	$2,114.8
Contingency (10%)	$211.5
Subtotal Mill Improvements	$2,326.3

Mill automation and dust control upgrades were also included in the PEA estimate to modify the existing mill.

21.1.4 Mine Rescue Station

It has been assumed that no other underground mine rescue team exists within a practical distance to the Lincoln Mine project. A mine rescue station will need to be included within Capital Expenditure to comply with the Mine Safety and Health Act. A total of $350,000 has been included to cover the purchase of the necessary equipment: breathing apparatus (14), gas meters, first aid equipment, and the apparatus bench equipment which includes a dryer, and tank refill station. This estimated item includes the minimum 80 hours of training for 14 people.

21.1.5 Owners Cost

The estimated owners cost during the pre-production period is shown in Table 21.5. The pre-production period is considered the first nine months of year 1.

Table 21.5 Estimated Pre-Production Owners Cost ($000's)

Department / Position	Jan	Feb	March	April	May	June	July	August	Sept	Total Preproduction
MINE										
Mine Superintendent	12.3	12.3	12.3	12.3	12.3	12.3	12.3	12.3	12.3	110.9
Mine Foreman		9.3	18.6	37.2	37.2	37.2	37.2	37.2	37.2	251.2
Miner I		35.2	70.4	105.6	158.3	193.5	211.1	211.1	211.1	1,196.3
Miner II			16.1	32.2	64.3	96.5	96.5	96.5	96.5	498.6
Equipment Operator			15.1	30.2	60.3	90.5	90.5	90.5	90.5	467.5
Laborer			20.1	30.2	40.2	40.2	40.2	40.2	40.2	251.3
Lead Mechanic/Electrician			8.0	16.1	16.1	16.1	16.1	16.1	16.1	104.6
Mechanic			14.1	21.1	28.1	28.1	28.1	28.1	28.1	175.9
Electrician			7.0	14.1	14.1	14.1	14.1	14.1	14.1	91.5
Subtotal Mine	**12.3**	**56.8**	**181.7**	**298.9**	**431.1**	**528.6**	**546.2**	**546.2**	**546.2**	**3,147.9**
PROCESS										
Process Superintendent	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	104.4
Process Foreman								8.7	17.4	26.1
Metallurgist/Lab Super		10.2	10.2	10.2	10.2	10.2	10.2	10.2	10.2	81.2
Lab Technician									5.4	5.4
Lead Operator									7.0	7.0
Operator									6.3	6.3
Mill Labor									5.0	5.0
Sand Plant Operator									5.0	5.0
Subtotal Plant	**11.6**	**21.8**	**21.8**	**21.8**	**21.8**	**21.8**	**21.8**	**30.5**	**68.0**	**240.5**
General and Administrative										
ENGINEERING & GEOLOGY										
Chief Engineer	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3	119.6
Engineer/Survey			9.1	18.1	18.1	18.1	18.1	18.1	18.1	117.8
Chief Geologist			10.6	10.6	10.6	10.6	10.6	10.6	10.6	74.4
Sampler			6.9	13.8	20.7	27.6	27.6	27.6	27.6	151.5
Subtotal Engineering & Geology	**13.3**	**13.3**	**39.9**	**55.8**	**62.7**	**69.6**	**69.6**	**69.6**	**69.6**	**463.4**
ADMINISTRATION										
General Manager	15.5	15.5	15.5	15.5	15.5	15.5	15.5	15.5	15.5	139.2
Mine Clerk/Payroll/HR			3.8	3.8	3.8	3.8	3.8	3.8	3.8	26.8
Maintenance/Warehouse Superintendent			11.6	11.6	11.6	11.6	11.6	11.6	11.6	81.2
Laborer/Operator				5.0	10.1	10.1	10.1	10.1	10.1	55.3
EHS Coordinator	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	65.3
Security			8.0	16.1	16.1	16.1	16.1	16.1	16.1	104.6
Custodial			2.5	5.0	5.0	5.0	5.0	5.0	5.0	32.7
Subtotal Administration	**22.7**	**22.7**	**48.7**	**64.3**	**69.3**	**69.3**	**69.3**	**69.3**	**69.3**	**505.0**
Subtotal Labor	**59.9**	**114.6**	**292.1**	**440.7**	**584.8**	**689.2**	**706.8**	**715.5**	**753.1**	**4,356.8**
Phone, Office Consumables	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	18.0
Local Taxes, Property Tax	8.3	8.3	8.3	8.3	8.3	8.3	8.3	8.3	8.3	75.0
Environmental Compliance & Other	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	12.5	112.5
Insurance	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	140.6
Eng Supplies			1.0	2.0	2.0	2.0	2.0	2.0	2.0	13.0
Totals	**98.4**	**153.0**	**331.5**	**481.2**	**625.3**	**729.7**	**747.3**	**756.0**	**793.5**	**4,716.0**

21.2 Manpower

Staff scheduling for mining operations of this nature would consist of two shifts of 10 hours each for the mine and three shifts of eight hours each for the mill. The mine will operate seven days per week, 350 days per year. Four shift crews will be necessary to fill the rotation of seven days on, seven days off for the mine. Milling will require only three shift crews, working five days per week, with weekends off. Table 21.6 shows the staff required for the operation.

Table 21.6 Manpower Estimate

Department / Position	Number
MINE	
Mine Superintendent	1
Mine Foreman	4
Miner I	24
Miner II	12
Equipment Operator	12
Laborer	8
Lead Mechanic/Electrician	2
Mechanic	4
Electrician	2
Subtotal Mine	**69**
PROCESS	
Process Superintendent	1
Process Foreman	2
Metallurgist/Lab Super	1
Lab Technician	2
Lead Operator	3
Operator	3
Mill Labor	3
Sand Plant Operator	4
Subtotal Plant	**19**
ENGINEERING & GEOLOGY	
Chief Engineer	1
Engineer/Survey	2
Chief Geologist	1
Sampler	4
Subtotal Engineering & Geology	**8**
ADMINISTRATION	
General Manager	1
Mine Clerk/Payroll/HR	1
Maintenance/Warehouse	1
Laborer/Operator	2
EHS Coordinator	1
Security	4
Custodial	2
Subtotal Administration	**12**
Total Property	**108**

21.3 Mining Operating Costs Estimate

The mining operating cost estimate is divided into several subgroups: Level Development, Decline Development, Raise Mining, Scram Mining, Backstope Mining, and Haulage and Ventilation. Note that the labor costs are provided in a separate table and are not included in the unit cost tables.

21.3.1 Level Development

Level development is entirely driven at 8ft wide by 8ft high (8ft x 8ft). Crosscuts and drifts share the same profile: an arched back drift with a spring-line at 4ft to the center of radius. Development advance is assumed at 6ft per round, one round per shift. The labor requirement is estimated at 2.4 man-shifts per round. Due to short length, 8ft x 8ft decline development costs are estimated as equivalent to the near-horizontal development. The cost is summarized per drift-foot in Table 21.7.

Table 21.7 8x8 Development Unit Cost Summary

Item	$/ft
Mining Consumables	$80.57
Crew Consumables	$8.22
Maintenance	$34.18
Utility Installation	$49.59
Equipment	$4.96
Haulage	$26.70
Compressed Air & Vent	$57.40
Subtotal Level Cost/ft	$261.62
Contingency (15%)	$39.24
TOTAL LEVEL COST/ft	**$300.86**

21.3.2 Decline Development

The SBA decline will be extended by 391ft, with an additional 125ft of crosscuts. The excavation size is nominally 12ft by 15ft with a nearly square or flat back. Crosscuts driven from the SBA decline will also be driven at 12ft x 15ft. Development advance is assumed at 10ft per round, one round per shift. The labor requirement is estimated at 4 man-shifts per round. Cost for all 12ft x 15ft development is summarized in Table 21.8.

Table 21.8 12x15 Development Unit Cost Summary

Item	$/ft
Mining Consumables	$234.84
Crew Consumables	$5.94
Maintenance Hourly	$100.00
Utility	$51.20
Equipment	$116.69
Haulage	$29.82
Compressed Air & Vent	$22.22
Subtotal Drift Cost/ft	$560.71
Contingency (15%)	$84.11
TOTAL DRIFT COST/ft	**$644.82**

21.3.3 Raise Mining Cost Estimate

Raise mining cost estimation is based upon a single two-man crew, working ten hour shifts, mining at a rate of 6ft per day. Round length is designed at 6ft. The raise is mined in ore, with an optimum dimension of 3ft by 8ft. The raise is timbered on one side for the manway and left open to rock for an "ore" pass/timber slide on the other. If a raise is placed mid-stope panel, stulls and lagging will be placed on both narrow sides. Utilities are installed as either permanent or temporary. Temporary utilities are removed and replaced with permanent utilities as the stope mining advances upward. The "ore" (or waste) from each round will be mucked by an equipment operator from the base of the raise to each level's dump pocket. The chute side of the timbered raise will also serve as the skip side for hoisting of materials to the face, if necessary.

The raise is mined during stope development to provide safe access and flow-through ventilation to the miners. In some cases, raises are mined through waste material for connection purposes. Although raise heights vary in the current mine design, few raises will be greater than 100ft. A 100ft average raise height was used for cost estimation.

Raise costs were developed on a cost per foot basis and converted to cost/ton for raises mined in ore. The estimated raise cost is shown in Table 21.9.

Table 21.9 Raise Development Unit Cost

Raise Cost Summary	
Item	**$/ft**
Mining Consumables	$82.32
Crew Consumables	$11.47
Maintenance	$15.87
Timber	$26.40
Equipment	$4.30
Subtotal Raise Cost/ft	$140.36
Contingency (10%)	$14.04
TOTAL RAISE COST/ft	**$154.40**
Item	**$/ton**
Subtotal Raise Cost/ton	$70.46
Ventilation and Compressed Air	$12.44
Haulage	$4.51
Contingency (10%)	$8.74
TOTAL RAISE COST/ton	**$96.15**

21.3.4 Scram Mining Cost Estimate

Scram mining cost estimation is based upon a single two-man crew, working in ten-hour shifts and mining at a cumulative rate of 12ft per day. Ideally and where possible, each miner will work in the opposite direction from each other from the same crosscut access, and cycle one 6ft round per day. The scram drift dimension will be 3ft by 8ft.

Scram miners will slush the "ore grade" material to the respective muck bay (crosscut) from which they are working. The "ore grade" material will then be loaded and hauled to each level's dump pocket by an equipment operator, who will be mucking from multiple faces in one day. Ventilation will be accomplished via heading fans and bag or duct to the face. No utilities will be placed, and little ground support is anticipated due to the short stand-up life before the panel is mined. No timber will be placed during scram mining. Table 21.10 summarizes the scram mining cost estimate.

Table 21.10 Scram Development Unit Cost Estimate

Scram Cost Summary	
Item	**$/ft**
Mining Consumables	$50.28
Crew Consumables	$7.53
Maintenance/Fuel	$7.73
Equipment	$3.66
Subtotal Scram Cost/ft	$69.20
Contingency (10%)	$6.92
TOTAL SCRAM COST/ft	**$76.12**
Item	**$/ton**
Subtotal Scram Cost/Ton	$34.74
Ventilation and Compressed Air	$12.44
Haulage	$4.51
Contingency (10%)	$5.17
TOTAL SCRAM COST/ton	**$56.86**

21.3.5 Back-Stoping Cost Estimate

Ideally, one three-man crew consisting of two Miners I and one Miner II will mine two adjacent panels simultaneously. The three-man crew will cycle two stope cuts at 90 hours (4.5 days), exclusive of backfilling and backfill cure/decant time. Each stope cut of 100ft long by 6ft high by 3ft wide will yield 150 tons. Timber costs for lifting the stope and building the backfill barrier are included within stope costs.

Back-stope costing is based upon a 200-hole round, blasted with ANFO and EZ-Drifter®-type detonators. Once the round is blasted, a slusher will be used to muck the "ore grade" material to the nearest "ore" chute, typically no more than 100ft away. The "ore" will flow by gravity to the muck bay, where it will be mucked by an equipment operator to the nearest level ore pass. Timber will be installed as necessary, and the stope will be backfilled. All timber costs, construction costs, and labor costs entailed with backfilling are included within backstope costing. These costs are estimated realizing that most stope panels share the costs of a center raise. Utilities are assumed to be already installed by the raise crew.

Back-stope costs were determined as a cost per horizontal foot length of stope and converted to cost per ton. Table 21.11 shows the estimated backstoping cost.

Table 21.11 Back-stoping Unit Cost Estimate

Summary - Backstoping	
Item	**$/ton**
Mining Consumables	$19.13
Crew Consumables	$3.39
Maintenance/Fuel	$5.89
Lift Costs	$2.70
Backfilling Cost	$7.50
Equipment	$5.54
Subtotal Stope Cost/ton	$44.15
Ventilation and Compressed Air	$12.44
Haulage	$4.51
Contingency (10%)	$6.11
TOTAL STOPE COST/ton	**$67.21**

21.3.6 Haulage, Compressed Air, and Ventilation

Haulage, compressed air, and ventilation costs were estimated on a cost per ton basis. Ventilation costs are based upon the results of the software simulation for the expected fans, with an additional consideration for development fans and ducting. Compressed air costs were estimated for a combined 600 HP compressor, capable of 3000 cfm. Other compressed air costs, such as couplings, pipe, etc., are assigned to scram, stope, or raise mining costs.

It is estimated that haulage will cost $4.51 per ton mined (Table 21.12), and compressed air and ventilation will cost $12.44 per ton mined (Table 21.13).

Table 21.12 Haulage Operating Cost

Item	$/ton
Maintenance & Fuel Hourly	$2.95
Equipment	$0.66
Subtotal Haulage cost/ton	$3.61
Contingency (25%)	$0.90
TOTAL Haulage Cost/ton	**$4.51**

Table 21.13 Ventilation and Compressed Air Operating Cost

Ventilation	Cost/hr	#	hr/day	Cost/day		Cost/ton
Fan - Electricity - 75HP motor	3.84	2	24	184.32		$1.23
Fan Consumables	10.97	2	24	526.56		$3.51
Compressed Air				**Cost/day**		**Cost/ton**
cfm	40.75	0.8	24	782.4		$5.22
					Subtotal	$9.96
* Western Mining & Milling				Contingency (25%)		$2.49
					TOTAL	**$12.44**

21.3.7 Mining Labor

Table 21.14 summarizes the mining labor cost estimate. The mine employs a total of 69 people in four crews.

Table 21.14 Mine Labor Estimated Cost

Department / Position	Number	Salary	Period	Burden	Annual Per Person	Annual Totals	Annual Per Ton
MINE							
Mine Superintendent	1	$102,000	$/Yr	45%	$147,900	$147,900	$2.82
Mine Foreman	4	$77,000	$/Yr	45%	$111,650	$446,600	$8.51
Miner I	24	$35.00	$/Hr	45%	$105,560	$2,533,440	$48.26
Miner II	12	$32.00	$/Hr	45%	$96,512	$1,158,144	$22.06
Equipment Operator	12	$30.00	$/Hr	45%	$90,480	$1,085,760	$20.68
Laborer	8	$20.00	$/Hr	45%	$60,320	$482,560	$9.19
Lead Mechanic/Electrician	2	$32.00	$/Hr	45%	$96,512	$193,024	$3.68
Mechanic	4	$28.00	$/Hr	45%	$84,448	$337,792	$6.43
Electrician	2	$28.00	$/Hr	45%	$84,448	$168,896	$3.22
Subtotal Mine	**69**					**$6,554,116**	**$124.84**

21.3.8 Mining Cost Summary

Overhand cut-and-fill mining is a high-cost method of extracting ore. Labor costs constitute a high percentage, approximately 70%, of the total mining costs. The unique geometry, grade, and narrow vein widths of the Lincoln-Comet deposit leave little alternative to a high-selectivity method and consequently higher mining cost per mined unit. Mining operating costs are summarized in Table 21.15.

Table 21.15 Estimated Mining Cost Summary

Item	$/unit	Units	Units/Yr	$/Yr	% of Total	$/ton ore
8x8 Level	$300.86	$/ft	2,280.9	$686,241	5.62%	$13.07
Raise -WASTE	$154.38	$/ft	188.6	$29,109	0.24%	$0.55
Scram - WASTE	$76.12	$/ft	0.0	$0	0.00%	$0.00
Raise -ORE	$96.15	$/ore ton	1,672.2	$160,780	1.32%	$3.06
Scram - ORE	$56.86	$/ore ton	1,664.6	$94,651	0.77%	$1.80
Stope Mining	$67.21	$/ore ton	49,164.0	$3,304,312	27.04%	$62.94
Management/Labor	$124.84	$/ore ton	52,500.8	$6,554,202	64.81%	$124.84
Level Chutes	$25,000.00	each	1.0	$25,000	0.20%	$0.48
Totals				**$10,854,296**	**100.00%**	**$206.74**

21.4 Processing Cost Estimate

21.4.1 Processing Direct Consumables

The process plant operating cost estimate was provided by Paul E. Danio and Associates LLC (Danio, 2010).

All process consumables are estimated from the metallurgical test work and are included in the operating cost estimate. Wear parts and lubricants are estimated from industry sources. Direct consumables are shown in Table 21.16.

Table 21.16 Processing Consumable Consumption Rates

Item	Rate
Lime/Acid	0.25 LB/T
Copper Sulfate	0.25 LB/T
Potassium Amyl Xanthate	0.25 LB/T
AC-208	0.25 LB/T
AC-3477	0.25 LB/T
Water	180,000 GAL/M
Wear Parts	2# STEEL/TON ORE
Assays	36 ASSAY/DAY (Gold Fire Assays)

21.4.2 Process Power Consumption

Power cost was determined at an average load of 320 KW demand and $0.12/KW-HR, as shown in Table 21.17.

Table 21.17 Process Power Requirement Summary

Item	Quantity	HP/Unit	Total HP
Belt Feeder	3	2	6
Transfer Belt	1	2	2
Transfer Belt	2	10	20
Jaw Crusher	1	50	50
Belt Feeder	2	2	2
Vibratory Screen	1	5	5
Rod Mill	1	100	100
Trash Screen	1	2	2
Knelson Concentrator	1	10	10
Sump Pump(s)	2	25	50
Floats	4	7.5	30
Regrind Mill	1	7.5	7.5
Dewatering Screw	1	1.5	1.5
Finishing Table	1	1.5	1.5
Disk Filter	1	2	2
Vacuum Pump	1	25	25
Tailings Pump	1	25	25
Tailings Thickener	1	5	5
Disk Filter (Tailings)	1	5	5
Vacuum Pump	1	60	60
Sludge Pump	1	5	5
Lighting and Power	1	20	20
Total Connected Load (HP)			**434.5**
Total Power Consumption (KW)			**324.1**

21.4.3 Process Manpower

Three shifts will operate for eight hours each, five days per week. It is assumed that the mill will operate 52 weeks per year. The process manpower and labor operating cost estimate is shown in Table 21.18.

Table 21.18 Process Plant Labor Cost Estimate

PROCESS	Number	Salary	Period	Burden	Annual Per Person	Annual Totals	Annual Per Ton
Process Superintendent	1	$96,000	$/Yr	45%	$139,200	$139,200	$2.65
Process Foreman	2	$72,000	$/Yr	45%	$104,400	$208,800	$3.98
Metallurgist/Lab Super	1	$84,000	$/Yr	45%	$121,800	$121,800	$2.32
Lab Technician	2	$45,000	$/Yr	45%	$65,250	$130,500	$2.49
Lead Operator	3	$28	$/Hr	45%	$84,448	$253,344	$4.83
Operator	3	$25	$/Hr	45%	$75,400	$226,200	$4.31
Mill Labor	3	$20	$/Hr	45%	$60,320	$180,960	$3.45
Sand Plant Operator	4	$20	$/Hr	45%	$60,320	$241,280	$4.60
Subtotal Plant	**19**					**$1,502,084**	**$28.61**

21.4.4 Process Plant Operating Cost Summary

The total mill operating cost is estimated to be $42.10 per ton of "ore" milled, including offsite concentrate processing. The cost breakdown is shown in Table 21.19.

Table 21.19 Process Plant Operating Cost Estimate

Item	Annual Cost	$/ton
Labor & Administration	$1,502,084	$28.61
Power & Water	$278,424	$5.30
Process Consumables	$58,500	$1.11
Assay Consumables	$79,200	$1.51
Maintenance and Wear Parts	$237,024	$4.51
Toll Processing	$55,126	$1.05
Total Process Operating Cost	$2,210,358	$42.10

21.4.5 Freight Costs

Freight costs were determined based upon loading of 22 tons of bagged flotation concentrate per truck, traveling a one-way distance of 400 miles. Costs were estimated at $75 per ton, or approximately $0.175 per ton-mile. This cost includes insurance.

For doré product, shipment to Salt Lake City via secure transport is estimated at $0.72 per ounce for 1,000-ounce (Au content) shipments according to a quotation from Brinks Global in 2010.

21.5 General and Administrative Cost Estimate

Mine management and non-direct labor are comprised of the mine labor not accounted for within direct mining costs; for example, the mine foreman, mine superintendent, and mine laborers. Employment costs include a 45% burden, accounting for payroll taxes, benefits, and insurance.

Maintenance and warehouse costs have been estimated based upon a staffing requirement of two lead mechanics/electrician, four mechanics, and two electricians. Direct maintenance costs of labor have been included within cost per ton and cost per foot calculations and are, therefore, not included in the above cost estimation. Employees of the maintenance shop are all hourly employees, with a 45% employee burden included.

The General and Administrative labor cost is shown in Table 21.20.

Table 21.20 General and Administrative Labor Cost Estimate

ENGINEERING & GEOLOGY	Number	Salary	Period	Burden	Per Person	Totals	Per Ton
Chief Engineer	1	$110,000	$/Yr	45%	159500	$159,500	$3.04
Engineer/Survey	2	$75,000	$/Yr	45%	108750	$217,500	$4.14
Chief Geologist	1	$88,000	$/Yr	45%	127600	$127,600	$2.43
Sampler	4	$57,000	$/Yr	45%	82650	$330,600	$6.30
Subtotal Engineering & Geology	**8**					**$835,200**	**$15.91**
ADMINISTRATION							
General Manager	1	$128,000	$/Yr	45%	185600	$185,600	$3.54
Mine Clerk/Payroll/HR	1	$46,000	$/Yr	45%	66700	$66,700	$1.27
Maintenance/Warehouse	1	$96,000	$/Yr	45%	139200	$139,200	$2.65
Laborer/Operator	2	$20	$/Hr	45%	60320	$120,640	$2.30
EHS Coordinator	1	$60,000	$/Yr	45%	87000	$87,000	$1.66
Security	4	$16	$/HR	45%	48256	$193,024	$3.68
Custodial	2	$10	$/HR	45%	30160	$60,320	$1.15
Subtotal Administration	**12**					**$852,484**	**$16.24**
Total General and Administrative	**20**					**$1,687,684**	**$32.15**

In addition to the labor cost, the General and Administrative costs include all costs that are not included in the mining and processing cost estimates. These include state and local taxes, property tax, insurance, supplies, automotive expenses, and environmental compliance expenses. A summary of the General and Administrative total cost estimate is shown in Table 21.21.

Table 21.21 General and Administrative Cost Estimate Summary

Item	$/Year	$/Ton Ore
Engineering and Geology Labor	$835,200	$15.91
Engineering and Geology Supplies	$24,000	$0.46
Administrative & Warehouse Labor	$852,484	$16.24
Administrative & Warehouse Supplies	$24,000	$0.46
Local Taxes, Sales Tax, Property Tax	$100,000	$1.90
Insurance	$250,000	$4.76
Environmental Compliance and Other	$150,000	$2.86
Corporate Administrative Charges	$180,000	$3.43
Total General and Administrative	**$2,415,684**	**$46.01**

21.6 Operating Cost Summary

A summary of the estimated operating cost is shown in Table 21.22.

Table 21.22 Operating Cost Summary

Item	$/Year (000's)	$/Ton Ore	$/oz Au Sold
Mining	$10,854.3	$206.75	$492.02
Processing	$2,210.3	$42.10	$101.20
G & A	$2,415.9	$46.01	$110.73
Totals	**$15,480.5**	**$294.86**	**$703.95**

Note that the overall average costs are slightly different as these are averages during the years of full production.

22.0 ECONOMIC ANALYSIS

A preliminary economic assessment of the Lincoln Mine project was performed based upon the determined mining rate and capital and operating costs. Cash flow projections were completed based on assumed recoveries, both in situ and during processing, payment for products sold at the approximate current 3-year trailing average, plus two-year forward estimated gold price. The gold price used in this preliminary economic assessment is $1,200.00 per ounce of gold.

This preliminary economic assessment is preliminary in nature, includes 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, and there is no certainty that the preliminary assessment will be realized.

22.1 Royalties

The project carries a 4% production royalty based on a pay value that is net of costs to process the product to refined gold, including transportation costs. An additional 0.5% NSR royalty is held by a consultant and is limited in aggregate to $1,000,000. U.S. Energy Corporation has a Net Profits Interest Royalty (“NPIR”) of 5% until the total amount of $4.6 million is paid, and a 1% NPIR thereafter. A 4% royalty on gross revenue is included in the pretax cashflow evaluation. This should be slightly more than the combination of all of the royalties.

22.2 Working Capital

Working capital for the project is estimated at the first three months operating cost, or $3.22 million. Working capital is set aside in year 1 and fully recovered in year 3.

22.3 Pre-tax Cash-flow Evaluation

The pre-tax cash-flow evaluation is shown in Table 22.1. The company has in excess of $30 million in tax write-offs to offset any taxes so an after-tax cash-flow evaluation is not applicable. The pre-tax cash-flow evaluation indicates an internal rate of return (“IRR”) of 63.7% while the net present value (“NPV”) at 5% is $23,411,300 (Table 22.1).

Table 22.1 Pre-Tax Cash-flow Evaluation

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	TOTAL
PRODUCTION								
Waste Tons (000'S)	39.6	13.7	12.2	12.2	12.2			89.8
Mine Production - Ore Tons (000'S)	8.4	52.5	52.5	52.5	52.5	26.4		244.8
Grade (ounce per ton)	0.46	0.46	0.46	0.46	0.46	0.46		0.46
Contained Oz Au (000'S)	3.8	24.0	24.0	24.0	24.0	12.1		111.9
Production - Gravity (000's oz Au)	2.7	16.8	16.8	16.8	16.8	8.4		78.3
Production - Flotation (000's oz Au)	1.0	6.2	6.2	6.2	6.2	3.1		29.1
Production - Flotation (000's oz Au Paid- 8	0.8	5.3	5.3	5.3	5.3	2.7		24.7
Total Sales (000's oz Au)	3.5	22.1	22.1	22.1	22.1	11.1		103.0
Gold Price - $/oz Au	1200	1200	1200	1200	1200	1200		
Gross Revenue ($000's)	$4,244.2	$26,491.7	$26,491.7	$26,491.7	$26,491.7	$13,260.3		$123,471.4
Royalty								
4.0% Gross Royalty (~Net of all royalties	$169.6	$1,058.7	$1,058.7	$1,058.7	$1,058.7	$529.9		$4,934.2
Net Revenue	$4,074.6	$25,433.1	$25,433.1	$25,433.1	$25,433.1	$12,730.4		$118,537.2
OPERATING COSTS								
Mining ($000's)								
Development - Waste (000's)	$0.0	$902.7	$686.2	$686.2	$686.2	$0.0		$2,961.4
Development - Ore (000's)	$0.0	$255.4	$255.4	$255.4	$255.4	$127.7		$1,149.4
Ore Mining	$565.3	$3,304.3	$3,304.3	$3,304.3	$3,304.3	$1,643.8		$15,426.3
Mine Labor	$1,638.5	$6,554.2	$6,554.2	$6,554.2	$6,554.2	$3,280.7		$31,136.0
Total Mining ($000)	$2,203.8	$11,016.6	$10,800.2	$10,800.2	$10,800.2	$5,052.1		$50,673.2
Processing ($000's)	$481.9	$2,210.3	$2,210.3	$2,210.3	$2,210.3	$1,099.9		$10,423.1
General and Administrative ($000's)	$538.1	$2,415.9	$2,415.9	$2,415.9	$2,415.9	$1,202.3		$11,404.0
Totals ($000's)	$3,223.9	$15,642.8	$15,426.4	$15,426.4	$15,426.4	$7,354.3		$72,500.2
Net Profit ($000's)	$850.7	$9,790.2	$10,006.7	$10,006.7	$10,006.7	$5,376.1		$46,037.0
Capital Investment	$12,474.7							$12,474.7
Working Capital	$3,223.9		($3,223.9)					$0.0
Closure Costs							$3,998.4	$3,998.4
Cash Flow	($14,848.0)	$9,790.2	$13,230.6	$10,006.7	$10,006.7	$5,376.1	($3,998.4)	$29,563.8
Cumulative Cashflow	($14,848.0)	($5,057.7)	$8,172.8	$18,179.5	$28,186.2	$33,562.3	$29,563.8	
NPV (5%)							$23,411.3	
NPV (8%)							$20,368.7	
IRR							63.7%	

22.4 Sensitivity Analysis

A sensitivity analysis was completed for the cash-flow shown in Table 22.1. The internal rate of return (IRR) sensitivity to variations in gold price, operating cost, and initial capital cost is shown in Figure 22.1, while the net present value (NPV) sensitivity is shown in Figure 22.2.

Figure 22.1 IRR Sensitivity

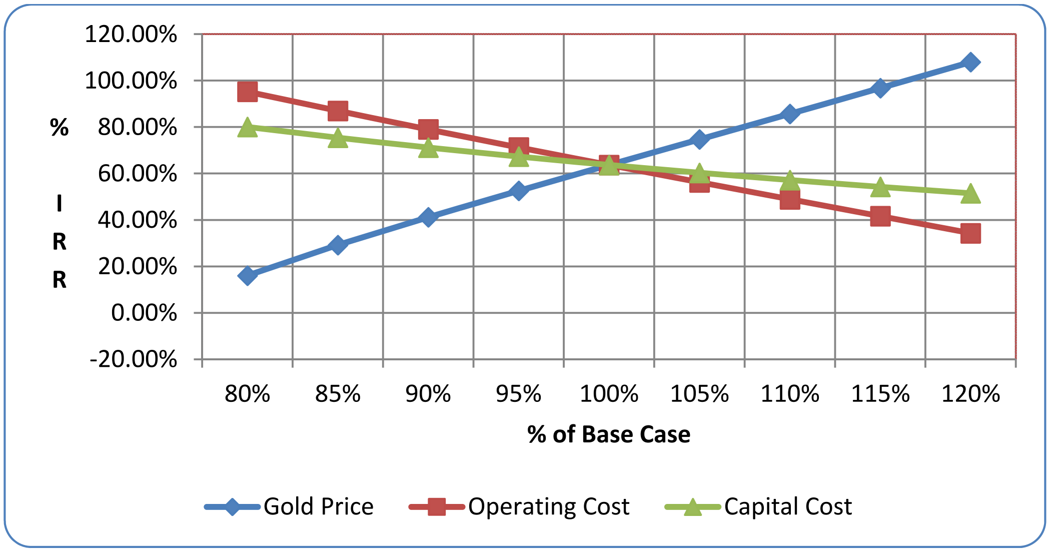
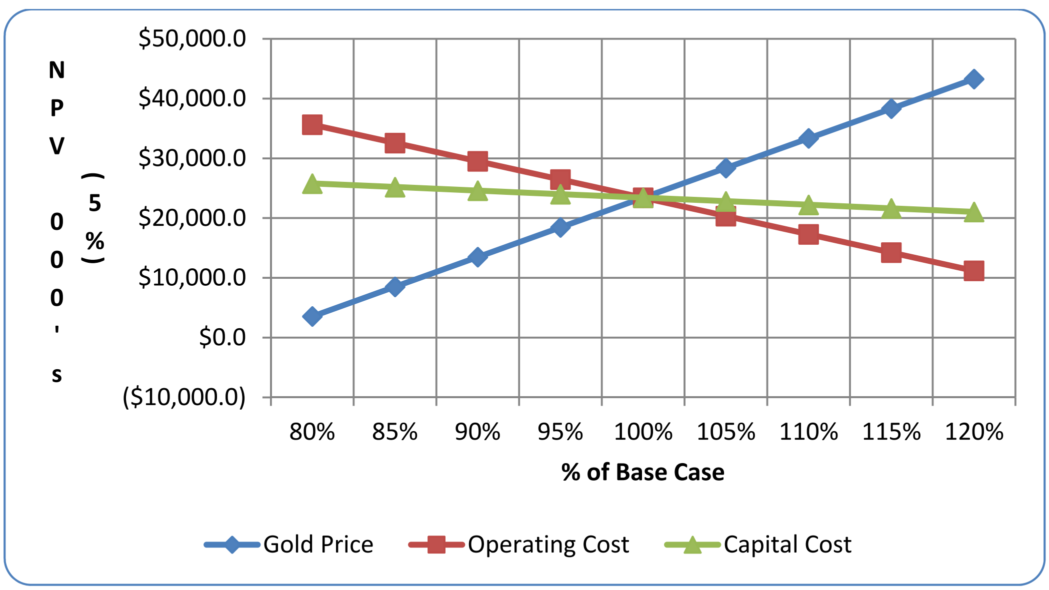


Figure 22.2 NPV (5%) Sensitivity



23.0 ADJACENT PROPERTIES

MDA is not aware of any information from adjacent properties relevant to this report. However, the property controlled by SGM is bounded to the north and south by other properties not controlled by SGM that were historic gold producers. There are also patented gold claims west of SGM's property that contain mineralized veins.

24.0 OTHER RELEVANT DATA AND INFORMATION

MDA is not aware of any other relevant information related to the Lincoln-Comet or Keystone resources on the Lincoln Mine property that has not been described in this report but that would change the conclusions or interpretations.

25.0 INTERPRETATIONS AND CONCLUSIONS

The Lincoln Mine property lies within the most productive portion of the historic Mother Lode in the western foothills of the Sierra Nevada. Eight major past-producing Mother Lode mines within the project area together accounted for 3.4 million ounces of gold production prior to 1953, or about 25% of the entire Mother Lode lode gold production. Despite this historic production from old mines on the current Lincoln Mine property, there seems to have been little historic mining and exploration in the Lincoln-Comet resource area, probably due to the absence of surface vein outcroppings in this area. However, substantial mining activity took place along the strike of the Lincoln and Comet veins, both north and south of the resource area.

The Lincoln Mine project resources occur within two deposits: Lincoln-Comet and Keystone (see Table 25.1). Approximately 20 percent of the project resources are classified as Indicated, all within the Lincoln-Comet. There are no Measured resources. No mineral reserves have been identified on the project and mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 25.1 Lincoln Mine Project Resources

Deposit	Classification	Au Cutoff (oz Au/ton)	Tons	Grade (oz Au/ton)	oz Au
Lincoln-Comet	Indicated	0.12	152,000	0.401	61,000
Lincoln-Comet	Inferred	0.12	506,000	0.254	128,000
Keystone	Inferred	0.12	399,000	0.243	97,000
total	Inferred	0.12	905,000	0.249	225,000

The Lincoln and Comet resource estimated in this report consists of mineralized vein zones of orogenic (mesothermal) type that trend N30°W and generally dip steeply west at an average of 70°. Most of the resource is hosted by northwest-striking, steeply dipping basaltic to andesitic metavolcanic flows and tuffs that are part of the Late Jurassic Mariposa Formation; the southeastern part of the resource is hosted within an overlying metavolcaniclastic and epiclastic unit within the Mariposa Formation. Gold-quartz-ankerite veins controlled by shear zones and cutting the Mariposa Formation contain free gold and 1-2% accessory sulfides. Within the Lincoln-Comet deposits, about 20% of the gold occurs as coarse grains up to 1/8in. in size.

The veins of the Lincoln-Comet deposit differ in three ways from other Mother Lode vein systems:

- In contrast to the main veins of the district that may extend down plunge many times their strike length, in the Lincoln-Comet area horizontal continuities of geology and grade are greater than vertical continuities;

- In contrast to the other quartz veins in the project area that dip east, most of the Lincoln and Comet veins dip west; and

- Mineralized shoots in the Lincoln and Comet zones have gentle to sub-horizontal plunges, different from those of the main veins.

MDA believes that the Lincoln Mine property, and specifically the Lincoln-Comet area, is a project of merit. The current classified Lincoln-Comet resource at a 0.12oz Au/ton cutoff grade contains an Indicated resource of 152Kt at an average grade of 0.401oz Au/ton and an Inferred resource of 506Kt at an average grade of 0.254oz Au/ton. The total gold resource is contained in over 30 distinct veins, many of which branch off the main through-going vein structures.

The most important characteristics that impact the Lincoln-Comet resource estimate are the strongly anastomosing nature of the narrow, gold-bearing veins and the significant mineral variability within the veins. The mineral variability is on a sub-sample scale to a mining scale and results in uncertainty in grade estimation both in the resource model and also in mine planning and reconciliation. Away from the underground development, there is spatial uncertainty in the geologic model due to the more widely spaced drill data and the highly variable, branching nature of the vein system.

There are no Measured resources within the Lincoln-Comet deposit, and just 30% of the resource is classified as Indicated. The lack of Measured and limited amount of Indicated material are a result of the spatial and grade estimation uncertainties inherent within the high-grade, coarse-gold deposit. Additional drilling within the currently defined deposit, especially away from the underground development, could materially change the existing resource with the discovery of either localized, high-grade mineralization within known veins or new veins branching off the main structures. There is potential for expanding the Inferred resource by drilling both down-dip and along strike to the northwest and southeast. Increasing the Indicated resource, though, would likely entail further underground development and tightly-spaced drilling.

An issue that affects the Lincoln-Comet resource model and estimate is that the property boundary impinges on the resource along the northeast boundary of the Comet zone. The existing resource reflects the current property boundary; any changes to this boundary would have a material effect on the resource model and estimate.

The Keystone resource estimated in this report consists of two mineralized Mother lode-style vein zones, the West Contact zone and the Medean zone, that trend N30°W and generally dip steeply east at an average of 50° to 70°. The mineralization style is similar to the Lincoln-Comet mineralization in that the gold-bearing veins, usually 1 to 4ft in thickness, occur as fissure veins within the structural zones.

The “K5” structure/vein is the dominant vein in the West Contact zone while the “K13” vein is the dominant vein in the Medean zone. Minor veins occur both footwall and hanging wall to these veins but the current resource model is based on just the K5 and K13 veins.

The current classified Keystone resource at a 0.12oz Au/ton cutoff grade contains an Inferred resource of 399Kt at an average grade of 0.243oz Au/ton. The resource is undiluted and based on a cross-sectional polygonal model.

There are no Measured or Indicated resources within the Keystone deposit. The lack of Measured and Indicated material are a result of the spatial and grade estimation uncertainties due to the limited, widely spaced drill data and also the inherent grade uncertainties within high-grade, coarse-gold deposits. Additional drilling within the currently defined deposit could materially change the existing resource with the discovery of either localized, high-grade mineralization within the known veins, or new veins branching off the main structures.

Tightly spaced drilling could result in an upgrade in classification though, as in the Lincoln-Comet, some underground development is advised to better characterize the local grade variability along the veins. An Indicated classification would also warrant the construction of a three dimensional block model and grade estimate to better characterize the local grade variability and vein location.

There is potential for expanding the Inferred resource by drilling both down-dip and along strike to the northwest and southeast.

An issue that affects the Keystone resource model and estimate is that the current resource is spatially related to historic underground development. Historic maps and reports, along with the incomplete 3D drawings available to MDA, show historic mine access and development on multiple levels in close proximity to the current resource. Voids were encountered within the modeled vein intercepts in a few of the "K5" drill holes. There is the possibility that some of the current resource in both veins has been mined out.

The Keystone resource is undiluted and some dilution is expected, depending on proposed mining methods. The effect on the resource is unknown but it is possible that some portions of the veins would no longer be considered economic under certain circumstances.

25.1 Risks and Opportunities

25.1.1 Risks

MDA believes that the biggest risk with the Lincoln Mine project is the ability to identify what is "ore grade", and what is not, inside the vein. SGM should plan an extensive sampling program with the initial mining to determine what level of sampling is appropriate to identify the ore-grade mineralization.

Additionally, the normal risks associated with underground mining are present, such as dilution and mining cost. When the veins present in the deposit intersect, they present some geometry problems that may cause higher than normal dilution and/or "ore" loss.

The project is sensitive to variations in gold price, which is both a risk and an opportunity.

The risks associated with the Keystone resource concern the relatively wide drill spacing and resulting spatial and grade uncertainties. There is also the presence of significant historic production associated with the two mineralized veins and the possibility that some portion of the current resource has been mined out.

25.1.2 Opportunities

The main opportunity at the property is believed to be the potential to increase resources down-dip of the current mineralization. While the drilling has indicated a drop off in grade, there are a number of veins that have not been adequately drilled down-dip of the current resources. In addition, consideration should be given to deep exploration by gathering as much data as possible on historic production on the property and from neighboring deposits.

Other opportunities are also risks such as dilution, gold price, capital and operating cost. The mine may experience lower dilution than predicted. Capital cost savings may be possible in the plant and mine areas.

26.0 RECOMMENDATIONS

With regards to the Lincoln-Comet resource, there are the following recommendations for further work:

- Additional twin and replicate analyses should be conducted on SGM core holes to bring greater confidence to the existing assay data used in the resource estimate. This should cost an estimated $5,000.

- In concert with the development work proposed below, a limited underground drilling program should be planned to test extensions of the known high-grade mineralization both down-dip and along strike from the proposed stope panels.

With regards to development of the Lincoln-Comet resource, the following work is recommended:

- An extensive sampling program coinciding with further development work to determine what level of sampling is appropriate to identify the potentially economic mineralization. Cost has been accounted for in development costs.

- During the pre-production development period as planning for the project proceeds, a test mining period should be included for the purpose of completing a detailed evaluation of the stope panels required to achieve production. The cost for test mining has been accounted for in development costs.

- A detailed development and mining schedule should be completed, at an estimated cost of $25,000, to provide a constant mill feed, optimize the equipment and labor, minimize ventilation requirements and re-handling of material, and maximize efficiency of the mining operation.

- Further testing of tailings and concentrate material characteristics should be completed prior to final backfill engineering. The estimated cost for this testing is $40,000.

- Although the need for ground support during development and production mining phases is expected to be light, with increasing mining depth a future review of the ground support program will be needed. MDA recommends that a ground support plan be designed by a geotechnical engineer. The estimated cost of a ground support plan is $20,000.

These recommendations do not constitute separate phases.

With regards to the Keystone resource, additional drilling is recommended both down-dip and along strike to the south of the current resource to determine potential size and extent of the mineralized veins. Extending the veins to the south would bring the Keystone resource closer to the current Lincoln-Comet underground development and result in easier and less costly underground access into the Keystone veins.

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Zimmerman, J. E., 1983, *The geology and structural evolution of a portion of the Mother Lode Belt, Amador County, California*: University of Arizona, M. S. thesis.

28.0 DATE AND SIGNATURE PAGE

Effective Date of report: July 2, 2015

The data on which the contained resource estimates are based was current as of the Effective Date.

Completion Date of report: July 2, 2015

"Paul G. Tietz" Date Signed:
Paul G. Tietz, C. P. G. July 2, 2015

"Neil Prenn" Date Signed:
Neil B. Prenn, P.Eng. July 2, 2015

"Steve Ristorcelli" Date Signed:
Steven Ristorcelli, C. P. G. July 2, 2015

"Corby G. Anderson" Date Signed:
Corby G. Anderson, CEng FIMMM, FIChemE July 2, 2015

29.0 CERTIFICATE OF AUTHORS

PAUL TIETZ, P. GEO.

I, Paul Tietz, P. Geo., do hereby certify that:

1. I am currently employed as Senior Geologist for Mine Development Associates, Inc. located at 210 South Rock Blvd., Reno, Nevada 89502 and

2. I graduated with a Bachelor of Science degree in Biology/Geology from the University of Rochester in 1977, a Master of Science degree in Geology from the University of North Carolina, Chapel Hill in 1981, and a Master of Science degree in Geological Engineering from the University of Nevada, Reno in 2004.

3. I am a Certified Professional Geologist (#11004) with the American Institute of Professional Geologists.

4. I have worked as a geologist continuously for 38 years since graduation from undergraduate university.

5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

6. I am one of the authors of this technical report titled “*Updated Technical Report, Lincoln Mine Project, Amador County, California*” prepared for Sutter Gold Mining Inc. and dated July 2, 2015. Except for those issues discussed in Section 3.0, I am responsible for Section 14.3 and take co-responsibility for all other sections of the Technical Report except for Sections 16.0, 17.0, 18.0, 19.0, 20.0, 21.0, and 22.0.

7. I visited the Lincoln Mine project site on April 19, 2009 and June 11, 2009, and on May 12th and 13th, 2015. I co-authored the “*Technical Report, Lincoln-Comet Project, Amador County, California*” prepared for Sutter Gold Mining Inc. and dated March 31, 2011.

8. To the best of my knowledge, information and belief, the technical report contains the necessary scientific and technical information to make the technical report not misleading.

9. I am independent of Sutter Gold Mining Inc. and related companies applying all of the tests in Section 1.5 of National Instrument 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with the requirements of that instrument and form.

Dated this July 2, 2015.

“Paul Tietz"
Paul Tietz

Paul Tietz
Print Name of Qualified Person

AUTHORS' CERTIFICATES

I, Neil B. Prenn, P. Eng., do hereby certify that:

1. I am currently employed as Principal Engineer by Mine Development Associates, 210 South Rock Blvd., Reno, Nevada 89502.
2. I graduated with an Engineer of Mines degree from the Colorado School of Mines in 1967.
3. I am a Registered Professional Mining Engineer in the state of Nevada (#7844) and a member of the Society of Mining Engineers and councilor-at-large for the Mining and Metallurgical Society of America.
4. I have worked as an engineer for a total of 48 years. I have had training and responsibility for managing mining operations which include all facets of mining and processing gold ores.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am one of the authors of the technical report titled "*Updated Technical Report, Lincoln Mine Project, Amador County, California*" prepared for Sutter Gold Mining Inc. and dated July 2, 2015. I am responsible for Sections 1.9, 16.0, 17.0, 18.0, 19.0, 20.0, 21.0, and 22.0, and co-responsible for Sections 25.0 and 26.0, except for those issues discussed in Section 3.0.
7. I visited the Lincoln-Comet property on November 10, 2003 and June 20, 2015. I have not had prior involvement with the property that is the subject of this Technical Report, other than prior reports as noted in the references.
8. To the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.
9. I am independent of Sutter Gold Mining Inc. and related companies as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with the requirements of that instrument and form.
11. The technical report contains information relating to mineral titles, permitting, environmental issues, regulatory matters and legal agreements. I am not a legal, environmental or regulatory professional, and do not offer a professional opinion regarding these issues.
12. A copy of this report is submitted as a computer readable file in Adobe Acrobat© PDF© format. The requirements of electronic filing necessitate submitting the report as an unlocked, editable file.

Dated this July 2, 2015

REGISTERED PROFESSIONAL ENGINEER-STATE OF NEVADA
NEIL B. PRENN
MINING
No. 7844

Neil B. Prenn

AUTHORS' CERTIFICATES

I, Steven Ristorcelli, C. P. G., do hereby certify that:

1. I am currently employed as Principal Geologist by:

Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502.

2. I graduated with a Bachelor of Science degree in Geology from Colorado State University in 1977 and a Master of Science degree in Geology from the University of New Mexico in 1980.

3. I am a Registered Professional Geologist in the states of California (#3964) and Wyoming (#153) and a Certified Professional Geologist (#10257) with the American Institute of Professional Geologists.

4. I have worked as a geologist continuously for 37 years since graduation from undergraduate university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am one of the authors of the report entitled "*Updated Technical Report, Lincoln Mine Project, Amador County, California*" prepared for Sutter Gold Mining Inc. and dated July 2, 2015. Except for those issues discussed in Section 3.0, I take co-responsibility for all sections of the Technical Report except for Sections 4.4, 14.3, 16.0, 17.0, 18.0, 19.0, 20.0, 21.0, and 22.0.

7. I visited the project on June 11, 2009 and on June 18, 2009, and I co-authored the "*Technical Report, Lincoln-Comet Project, Amador County, California*" prepared for Sutter Gold Mining Inc. and dated March 31, 2011.

8. To the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

9. I am independent of Sutter Gold Mining Inc. and related companies as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with the requirements of that instrument and form.

12. A copy of this report is submitted as a computer readable file in Adobe Acrobat© PDF© format. The requirements of electronic filing necessitate submitting the report as an unlocked, editable file.

Dated this July 2, 2015.

"Steve Ristorcelli"

Signature of Qualified Person
Steven Ristorcelli

AUTHORS' CERTIFICATES

I, Corby G. Anderson CEng FIMMM, FIChemE, do hereby certify that:

1. I am currently employed as an Engineer by: Allihies Engineering Incorporated, Thornton Building, 65 East Broadway Stree, Suite 326, Butte, Montana 59701, USA.

2. I graduated with a Bachelor of Science degree in Chemical Engineering from Montana State University 1979, a Master of Science degree in Metallurgical Engineering from Montana Tech in 1984, and a PhD in Mining Engineering – Metallurgy from the University of Idaho in 1987.

3. I am a Qualified Professional as a registered member 01079QP of the Mining and Metallurgical Society of America and a Fellow of the Institute of Materials, Minerals and Mining.

4. I have worked as an engineer continuously for over 36 years since graduation from undergraduate university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the following sections of this technical report entitled "*Updated Technical Report, Lincoln Mine Project, Amador County, California*" prepared for Sutter Gold Mining Inc. and dated July 2, 2015: 1.6 Metallurgical Testing and 13.0 Mineral Processing and Metallurgical Testing.

7. I have had no prior involvement with the property or project. I have not visited the project.

8. To the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

9. I am independent of Sutter Gold Mining Inc. and all their subsidiaries as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with the requirements of that instrument and form.

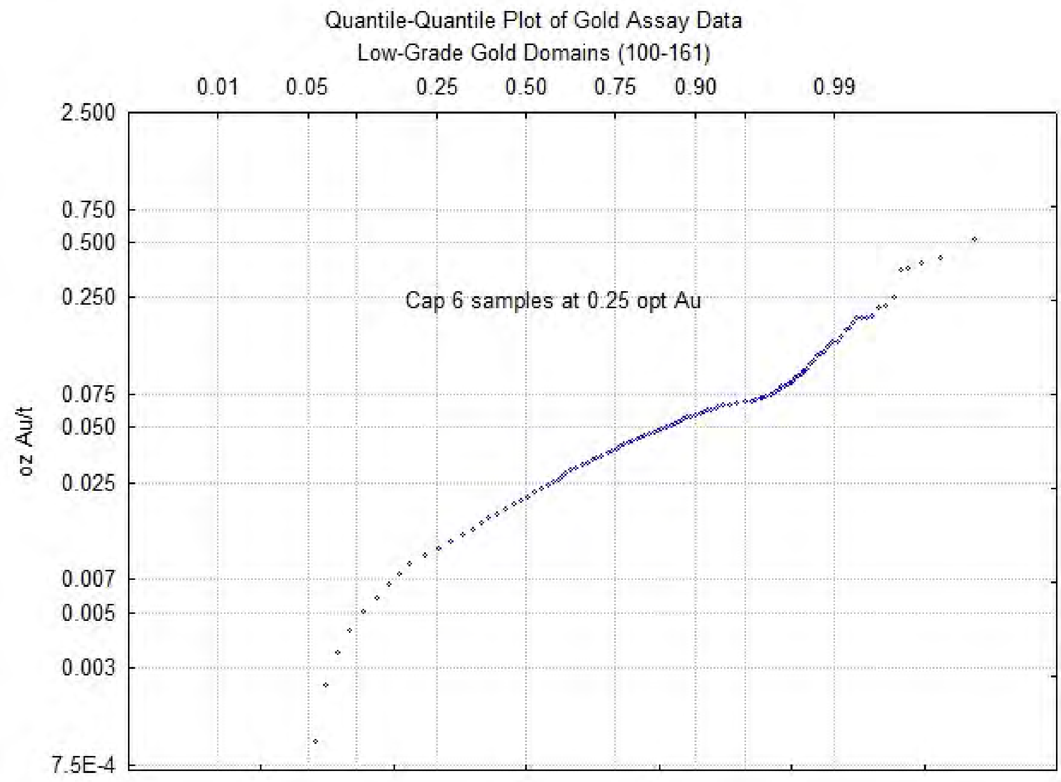
11. The Technical Report contains information relating to mineral titles, permitting, environmental issues, regulatory matters and legal agreements. I am not a legal, environmental or regulatory professional, and do not offer a professional opinion regarding these issues.

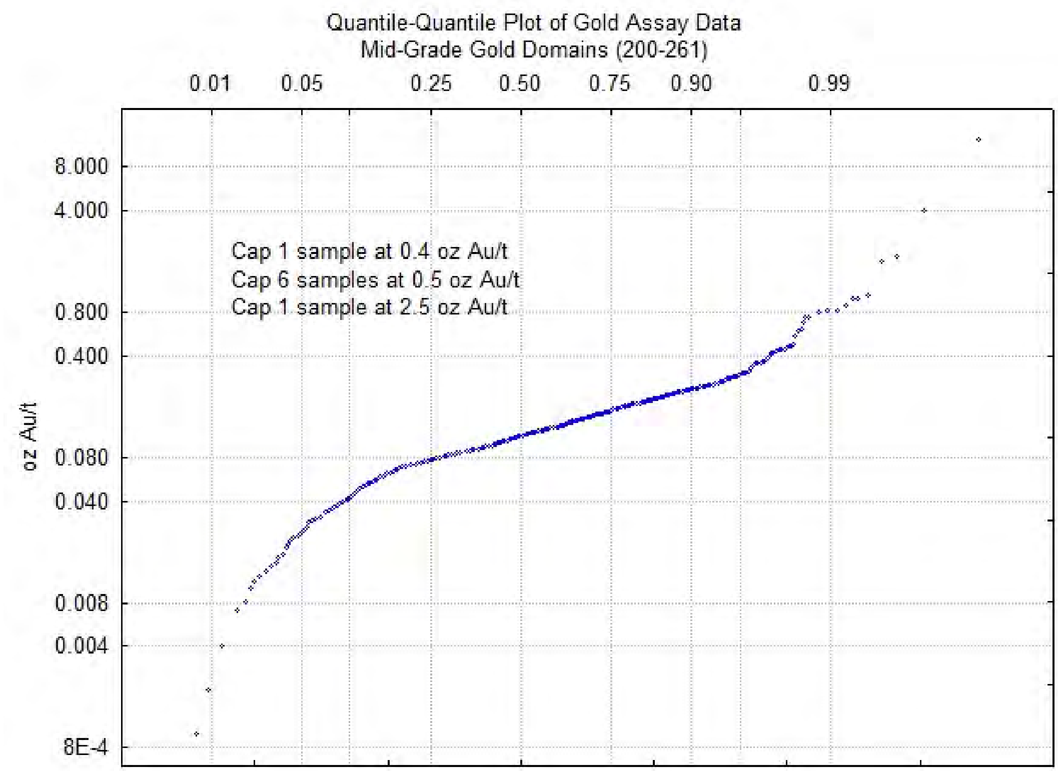
12. A copy of this report is submitted as a computer readable file in Adobe Acrobat PDF format. The requirements of electronic filing necessitate submitting the report as an unlocked, editable file.

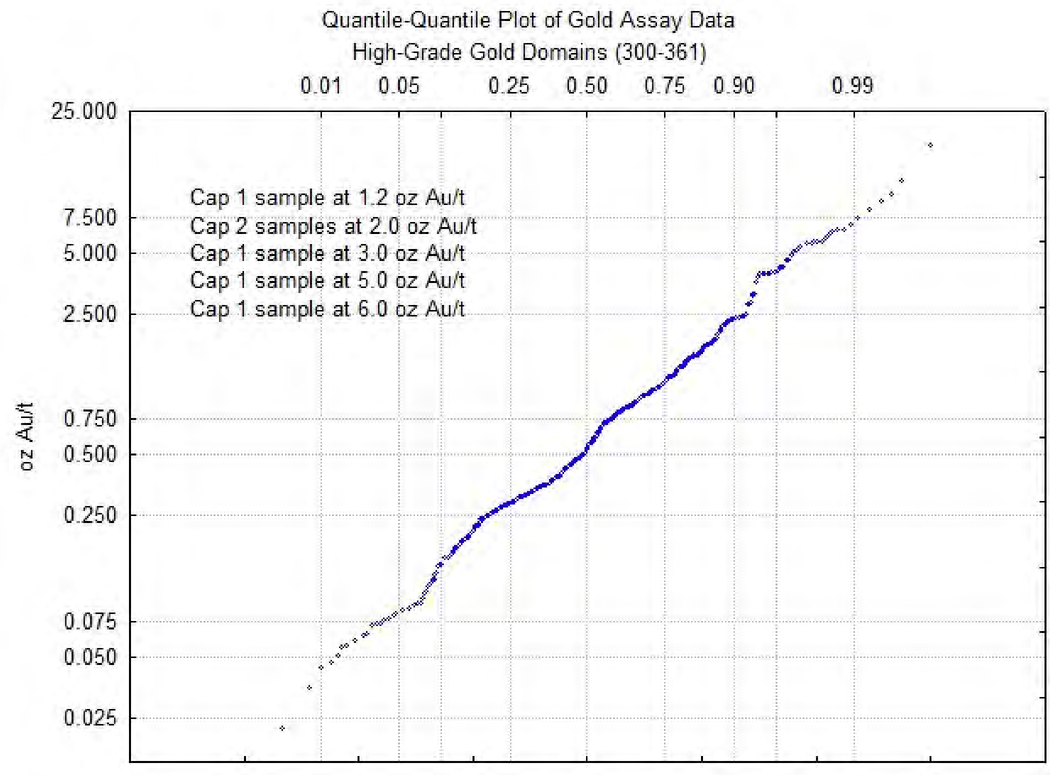
Dated this 2nd day of July, 2015

"Corby G Anderson"
Signature of Qualified Person
Corby G. Anderson

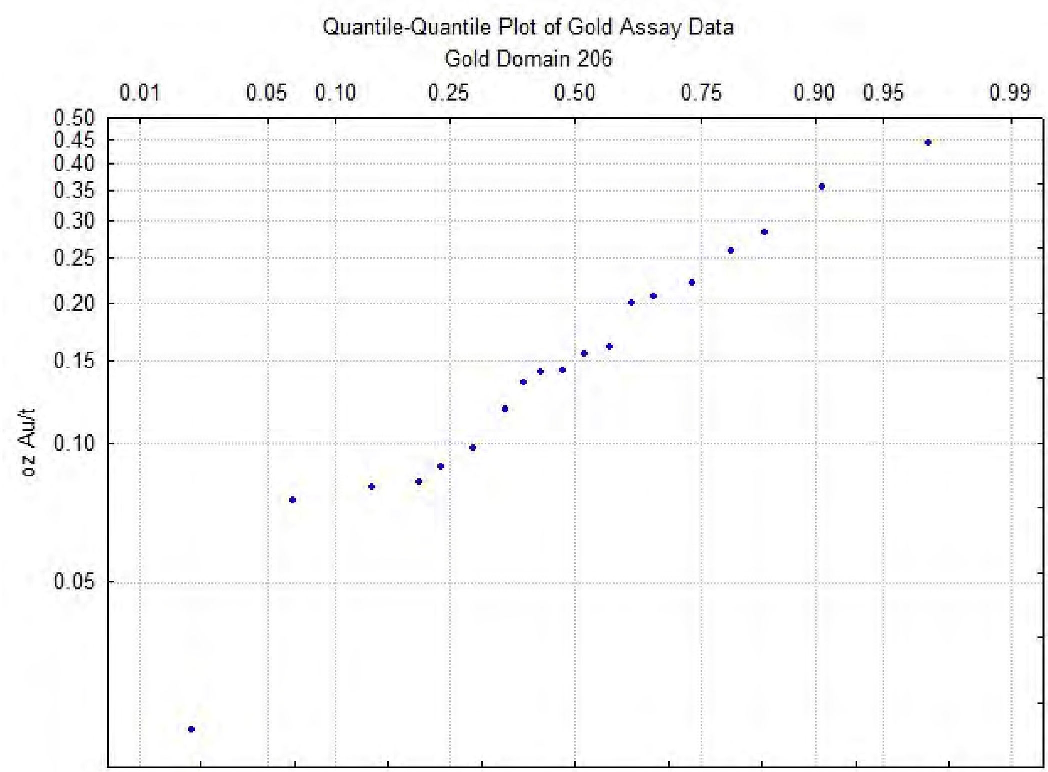
APPENDIX A
Lincoln-Comet Gold Domain Cumulative Probability Plots – Sample Assay Data


Quantile-Quantile Plot of Gold Assay Data
Low-Grade Gold Domains (100-161)
0.01
0.05
0.25
0.50
0.75
0.90
0.99
2.500
0.750
0.500
0.250
0.075
0.050
0.025
0.007
0.005
0.003
7.5E-4
oz Au/t
Cap 6 samples at 0.25 opt Au

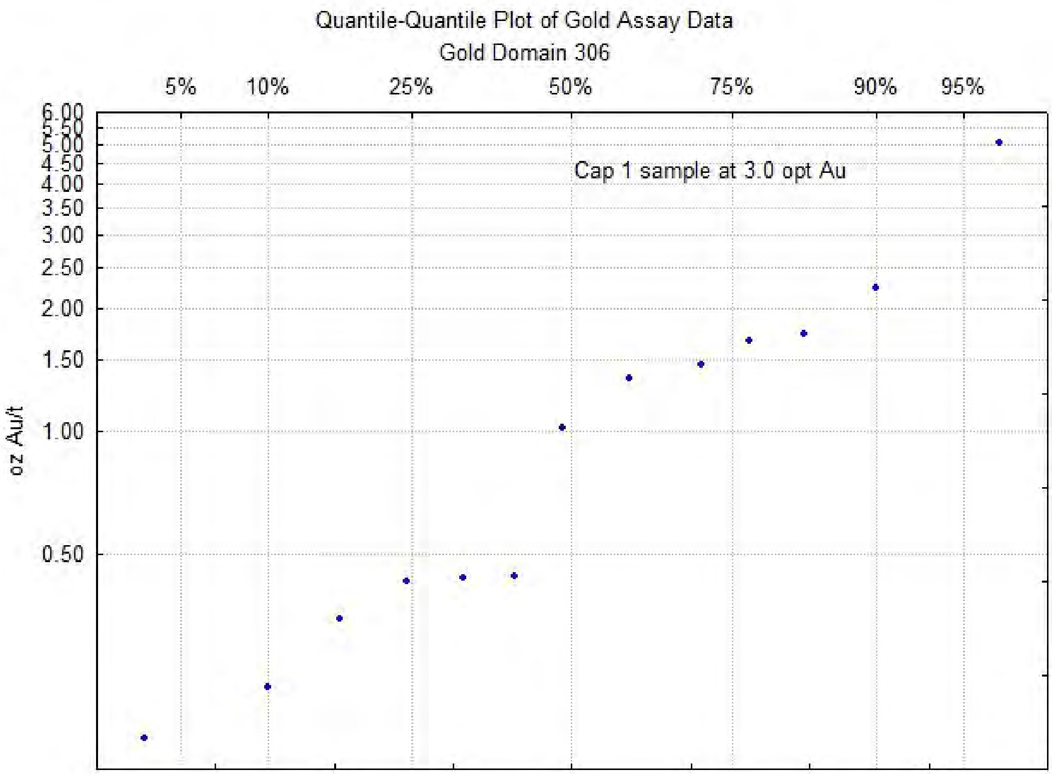

Quantile-Quantile Plot of Gold Assay Data
Mid-Grade Gold Domains (200-261)
0.01
0.05
0.25
0.50
0.75
0.90
0.99
8.000
4.000
0.800
0.400
0.080
0.040
0.008
0.004
8E-4
oz Au/t
Cap 1 sample at 0.4 oz Au/t
Cap 6 samples at 0.5 oz Au/t
Cap 1 sample at 2.5 oz Au/t

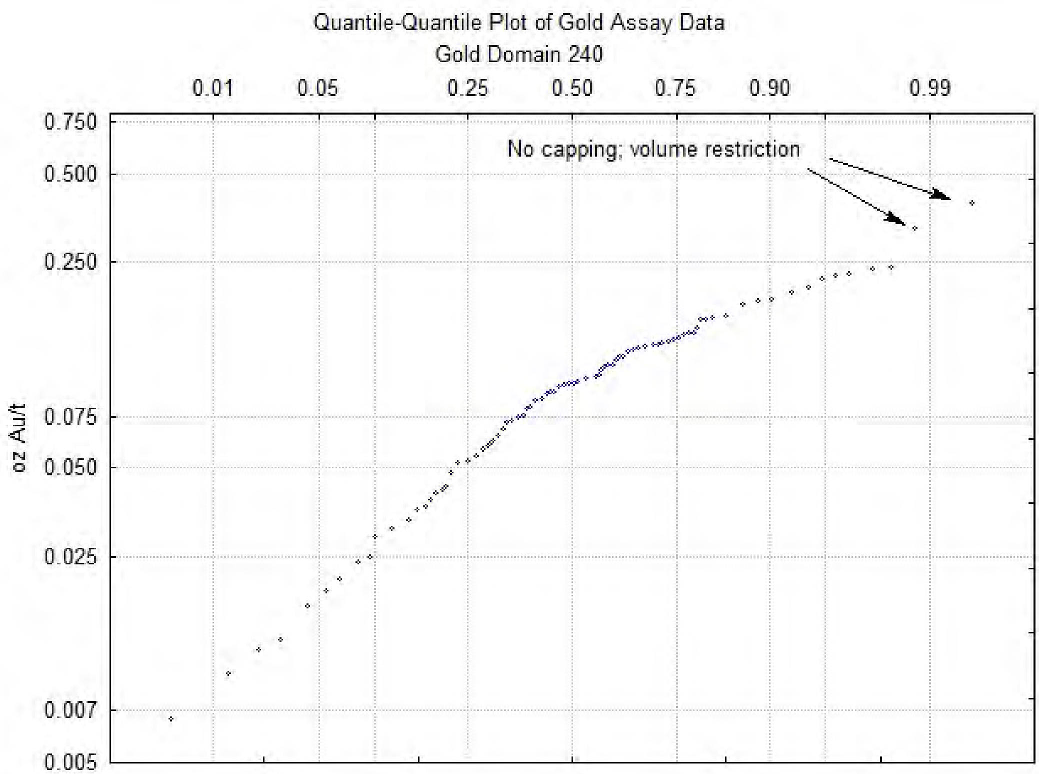


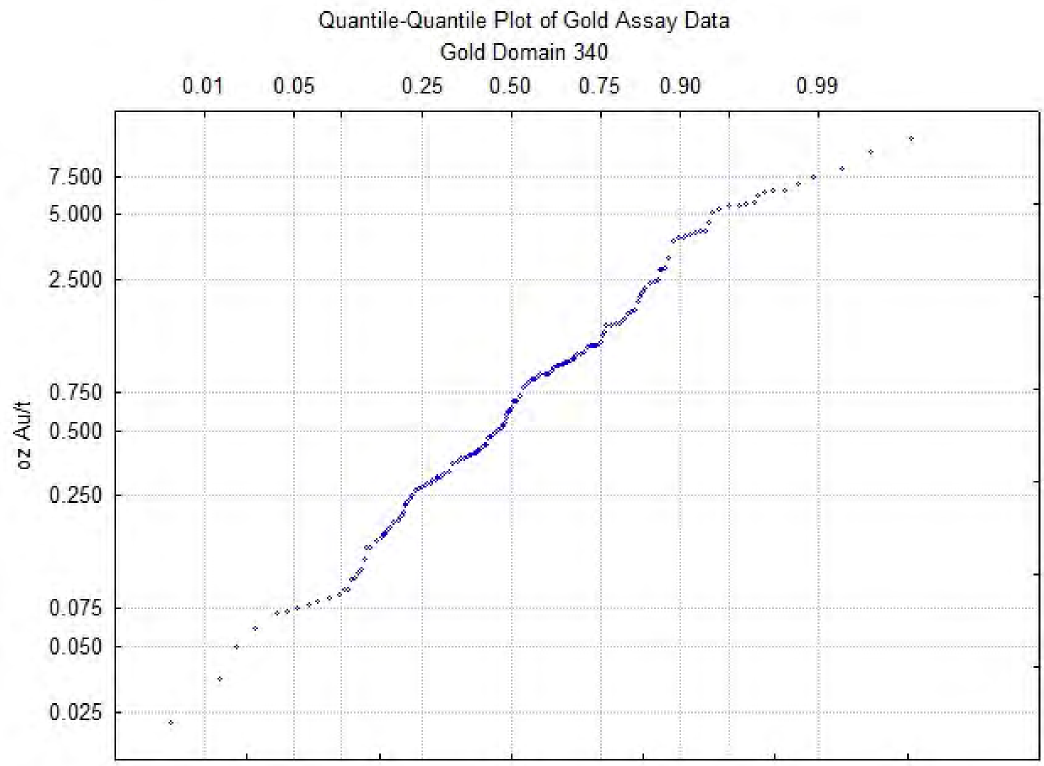
Quantile-Quantile Plot of Gold Assay Data
High-Grade Gold Domains (300-361)
0.01 0.05 0.25 0.50 0.75 0.90 0.99
25.000
7.500
5.000
2.500
0.750
0.500
0.250
0.075
0.050
0.025
oz Au/t
Cap 1 sample at 1.2 oz Au/t
Cap 2 samples at 2.0 oz Au/t
Cap 1 sample at 3.0 oz Au/t
Cap 1 sample at 5.0 oz Au/t
Cap 1 sample at 6.0 oz Au/t

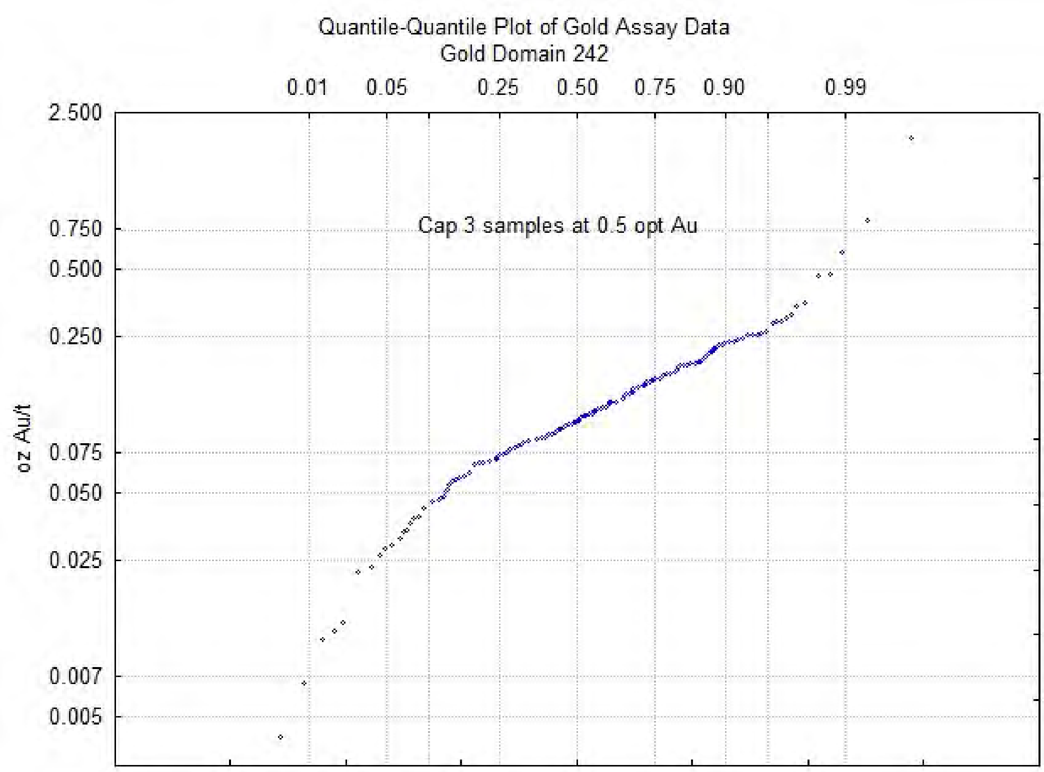


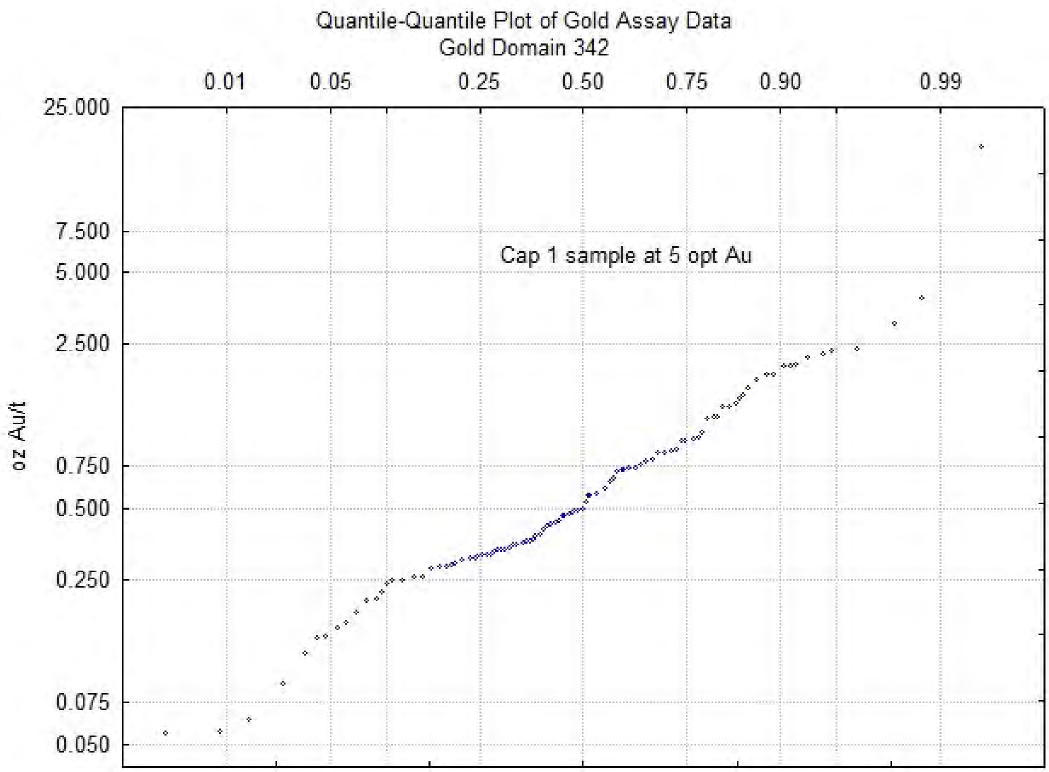
Quantile-Quantile Plot of Gold Assay Data
Gold Domain 206
0.01 0.05 0.10 0.25 0.50 0.75 0.90 0.95 0.99
0.50
0.45
0.40
0.35
0.30
0.25
0.20
0.15
0.10
0.05
oz Au/t

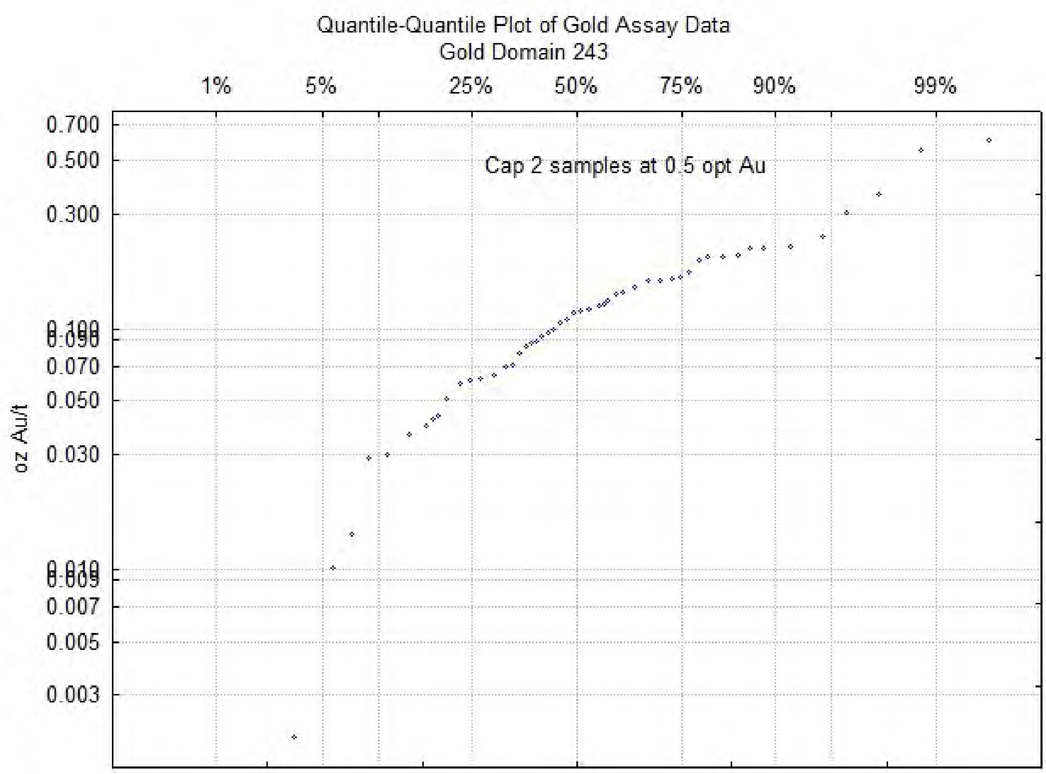

Quantile-Quantile Plot of Gold Assay Data
Gold Domain 306
5%
10%
25%
50%
75%
90%
95%
6.00
5.50
5.00
4.50
4.00
3.50
3.00
2.50
2.00
1.50
1.00
0.50
oz Au/t
Cap 1 sample at 3.0 opt Au

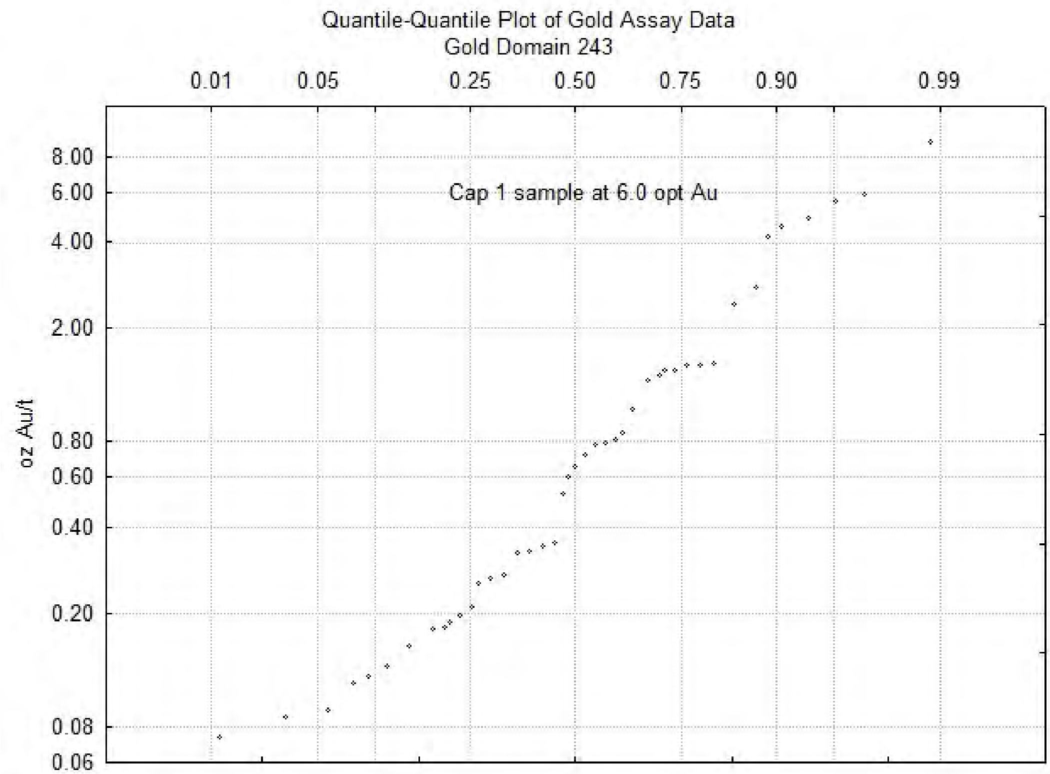

Quantile-Quantile Plot of Gold Assay Data
Gold Domain 240
0.01
0.05
0.25
0.50
0.75
0.90
0.99
0.750
0.500
0.250
0.075
0.050
0.025
0.007
0.005
oz Au/t
No capping; volume restriction

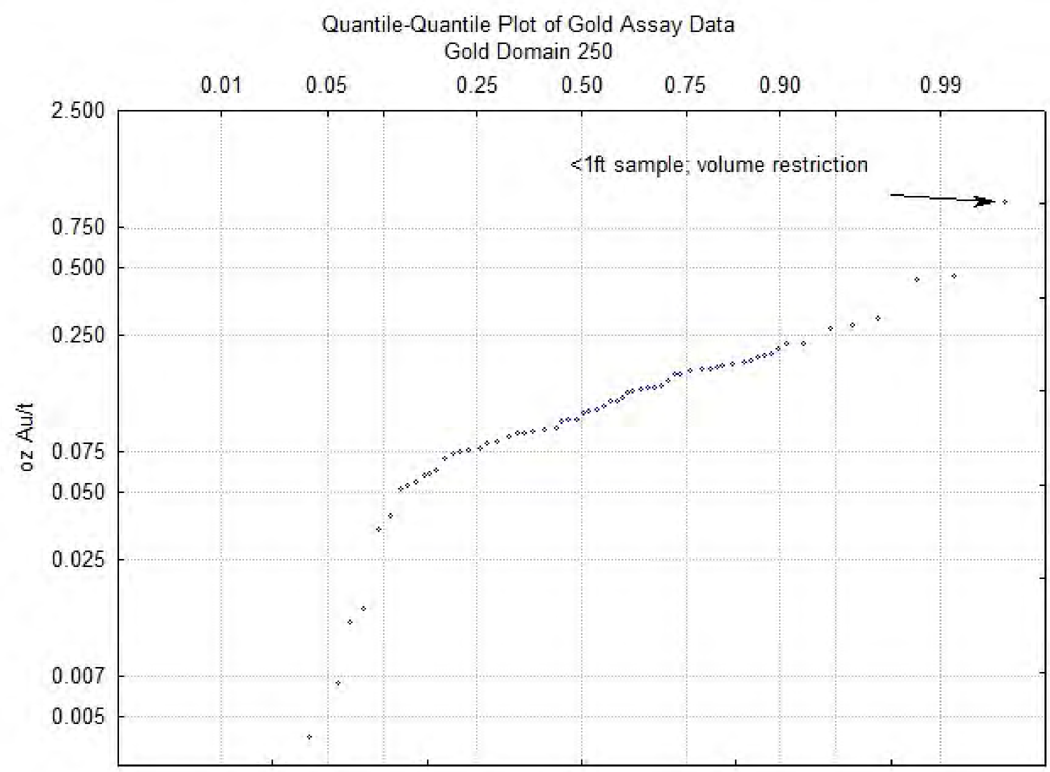

Quantile-Quantile Plot of Gold Assay Data
Gold Domain 340
0.01
0.05
0.25
0.50
0.75
0.90
0.99
7.500
5.000
2.500
0.750
0.500
0.250
0.075
0.050
0.025
oz Au/t

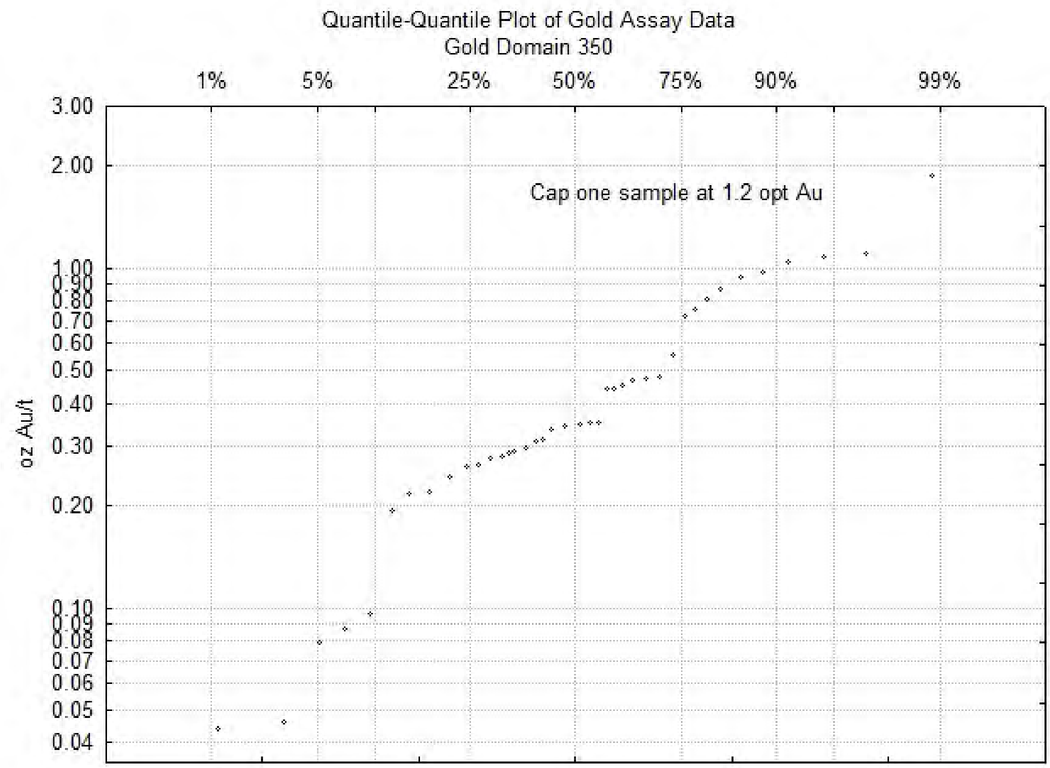

Quantile-Quantile Plot of Gold Assay Data
Gold Domain 242
0.01
0.05
0.25
0.50
0.75
0.90
0.99
2.500
0.750
0.500
0.250
0.075
0.050
0.025
0.007
0.005
oz Au/t
Cap 3 samples at 0.5 opt Au


Quantile-Quantile Plot of Gold Assay Data
Gold Domain 342
0.01
0.05
0.25
0.50
0.75
0.90
0.99
25.000
7.500
5.000
2.500
0.750
0.500
0.250
0.075
0.050
oz Au/t
Cap 1 sample at 5 opt Au


Quantile-Quantile Plot of Gold Assay Data
Gold Domain 243
1%
5%
25%
50%
75%
90%
99%
0.700
0.500
0.300
0.100
0.090
0.070
0.050
0.030
0.010
0.009
0.007
0.005
0.003
oz Au/t
Cap 2 samples at 0.5 opt Au


Quantile-Quantile Plot of Gold Assay Data
Gold Domain 243
0.01
0.05
0.25
0.50
0.75
0.90
0.99
8.00
6.00
4.00
2.00
0.80
0.60
0.40
0.20
0.08
0.06
oz Au/t
Cap 1 sample at 6.0 opt Au


Quantile-Quantile Plot of Gold Assay Data
Gold Domain 250
0.01
0.05
0.25
0.50
0.75
0.90
0.99
2.500
0.750
0.500
0.250
0.075
0.050
0.025
0.007
0.005
oz Au/t
<1ft sample; volume restriction


Quantile-Quantile Plot of Gold Assay Data
Gold Domain 350
1%
5%
25%
50%
75%
90%
99%
3.00
2.00
1.00
0.90
0.80
0.70
0.60
0.50
0.40
0.30
0.20
0.10
0.09
0.08
0.07
0.06
0.05
0.04
oz Au/t
Cap one sample at 1.2 opt Au

APPENDIX B
Lincoln-Comet Gold Domain Composite Statistics

All Coded Composites (weighted - domains 100+200+300)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2276	3.393	3.500	1.494	0.440	0.100	5.000	ft
Au_Grade	2276	0.204	0.044	0.581	2.853	0.000	11.190	oz Au/t

Low Grade Composites (weighted - domain 100s)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	1238	3.616	4.000	1.476	0.408	0.100	5.000	ft
Au_Grade	1238	0.028	0.022	0.023	0.818	0.000	0.250	oz Au/t

Medium Grade Composites (weighted - domain 200s)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	648	3.234	3.000	1.474	0.456	0.300	5.000	ft
Au_Grade	648	0.136	0.116	0.117	0.860	0.000	2.500	oz Au/t

High Grade Composites (weighted - domain 300s)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	390	2.95	3.000	1.455	0.493	0.200	5.000	ft
Au_Grade	390	1.01	0.610	1.208	1.196	0.000	11.190	oz Au/t

Composites Coded to Domain 100 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	119	3.661	4.500	1.510	0.412	0.500	5.000	ft
Au_Grade	119	0.03	0.025	0.019	0.636	0.002	0.078	oz Au/t

Composites Coded to Domain 102 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	23	3.848	5.000	1.577	0.410	1.000	5.000	ft
Au_Grade	23	0.023	0.019	0.013	0.560	0.000	0.064	oz Au/t

Composites Coded to Domain 103 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	6	3.25	3.500	1.943	0.598	1.000	5.000	ft
Au_Grade	6	0.036	0.034	0.016	0.437	0.014	0.059	oz Au/t

Composites Coded to Domain 105 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	21	3.724	5.000	1.586	0.426	1.000	5.000	ft
Au_Grade	21	0.033	0.031	0.034	1.027	0.000	0.219	oz Au/t

Composites Coded to Domain 106 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	76	3.757	5.000	1.491	0.397	0.500	5.000	ft
Au_Grade	76	0.028	0.023	0.016	0.552	0.000	0.071	oz Au/t

Composites Coded to Domain 107 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	7	4.371	5.000	1.494	0.342	1.000	5.000	ft
Au_Grade	7	0.051	0.024	0.066	1.290	0.005	0.199	oz Au/t

Composites Coded to Domain 109 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	17	4.176	5.000	1.581	0.378	0.500	5.000	ft
Au_Grade	17	0.031	0.032	0.018	0.601	0.004	0.062	oz Au/t

Composites Coded to Domain 110 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	12	3.708	4.250	1.484	0.400	1.000	5.000	ft
Au_Grade	12	0.039	0.026	0.036	0.945	0.011	0.153	oz Au/t

Composites Coded to Domain 116 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	4.5	4.500	0.707	0.157	4.000	5.000	ft
Au_Grade	2	0.014	0.016	0.002	0.176	0.012	0.016	oz Au/t

Composites Coded to Domain 120 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	13	3.362	3.500	1.527	0.454	0.500	5.000	ft
Au_Grade	13	0.026	0.028	0.011	0.410	0.007	0.041	oz Au/t

Composites Coded to Domain 121 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	56	4.014	5.000	1.277	0.318	1.000	5.000	ft
Au_Grade	56	0.033	0.021	0.033	1.018	0.000	0.171	oz Au/t

Composites Coded to Domain 123 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	36	3.914	5.000	1.740	0.445	0.100	5.000	ft
Au_Grade	36	0.024	0.023	0.014	0.598	0.000	0.051	oz Au/t

Composites Coded to Domain 125 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	2.25	2.250	1.768	0.786	1.000	3.500	ft
Au_Grade	2	0.016	0.020	0.008	0.530	0.002	0.020	oz Au/t

Composites Coded to Domain 126 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	27	3.952	5.000	1.208	0.306	2.000	5.000	ft
Au_Grade	27	0.027	0.023	0.018	0.650	0.001	0.066	oz Au/t

Composites Coded to Domain 127 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	4	3.75	3.500	0.957	0.255	3.000	5.000	ft
Au_Grade	4	0.031	0.032	0.022	0.708	0.013	0.070	oz Au/t

Composites Coded to Domain 128 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	11	3.455	3.000	1.422	0.412	1.000	5.000	ft
Au_Grade	11	0.023	0.027	0.012	0.502	0.009	0.050	oz Au/t

Composites Coded to Domain 129 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	7	3.643	4.500	1.651	0.453	1.000	5.000	ft
Au_Grade	7	0.029	0.029	0.011	0.383	0.004	0.047	oz Au/t

Composites Coded to Domain 130 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	8	3.038	3.000	1.074	0.354	1.000	5.000	ft
Au_Grade	8	0.02	0.010	0.016	0.794	0.004	0.048	oz Au/t

Composites Coded to Domain 131 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	56	3.589	4.000	1.449	0.404	1.000	5.000	ft
Au_Grade	56	0.024	0.021	0.014	0.564	0.002	0.070	oz Au/t

Composites Coded to Domain 132 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	2.5	2.500	0.707	0.283	2.000	3.000	ft
Au_Grade	2	0.034	0.025	0.012	0.357	0.025	0.047	oz Au/t

Composites Coded to Domain 134 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	5	3.54	3.000	0.871	0.246	3.000	5.000	ft
Au_Grade	5	0.029	0.036	0.011	0.381	0.015	0.040	oz Au/t

Composites Coded to Domain 136 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	3	3.667	3.500	1.258	0.343	2.500	5.000	ft
Au_Grade	3	0.021	0.022	0.009	0.438	0.013	0.035	oz Au/t

Composites Coded to Domain 137 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	9	3.511	5.000	1.817	0.517	0.600	5.000	ft
Au_Grade	9	0.038	0.035	0.022	0.591	0.008	0.087	oz Au/t

Composites Coded to Domain 138 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	5	3.84	4.000	1.236	0.322	2.200	5.000	ft
Au_Grade	5	0.027	0.020	0.014	0.525	0.014	0.045	oz Au/t

Composites Coded to Domain 139 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	13	2.9	3.000	1.552	0.535	0.100	5.000	ft
Au_Grade	13	0.043	0.030	0.031	0.731	0.004	0.110	oz Au/t

Composites Coded to Domain 140 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	91	3.189	3.000	1.472	0.462	0.500	5.000	ft
Au_Grade	91	0.029	0.024	0.024	0.812	0.000	0.143	oz Au/t

Composites Coded to Domain 141 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	21	3.324	3.500	1.406	0.423	1.000	5.000	ft
Au_Grade	21	0.026	0.021	0.019	0.730	0.004	0.068	oz Au/t

Composites Coded to Domain 142 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	247	3.646	4.000	1.459	0.400	0.400	5.000	ft
Au_Grade	247	0.026	0.022	0.020	0.770	0.000	0.128	oz Au/t

Composites Coded to Domain 143 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	66	3.683	4.250	1.536	0.417	0.400	5.000	ft
Au_Grade	66	0.026	0.021	0.027	1.008	0.000	0.168	oz Au/t

Composites Coded to Domain 144 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	20	3.88	4.500	1.271	0.327	2.000	5.000	ft
Au_Grade	20	0.027	0.023	0.017	0.645	0.004	0.071	oz Au/t

Composites Coded to Domain 145 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	26	3.423	3.750	1.683	0.492	0.500	5.000	ft
Au_Grade	26	0.026	0.023	0.016	0.608	0.002	0.064	oz Au/t

Composites Coded to Domain 146 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	3	3.000	2.828	0.943	1.000	5.000	ft
Au_Grade	2	0.019	0.019	0.000	0.000	0.019	0.019	oz Au/t

Composites Coded to Domain 148 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	4	3.625	4.250	1.887	0.521	1.000	5.000	ft
Au_Grade	4	0.032	0.028	0.019	0.596	0.010	0.057	oz Au/t

Composites Coded to Domain 150 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	106	3.437	3.000	1.395	0.406	0.200	5.000	ft
Au_Grade	106	0.026	0.021	0.022	0.845	0.000	0.149	oz Au/t

Composites Coded to Domain 151 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	62	3.55	4.000	1.447	0.407	0.500	5.000	ft
Au_Grade	62	0.029	0.022	0.023	0.807	0.004	0.182	oz Au/t

Composites Coded to Domain 160 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	4	4.225	4.400	0.866	0.205	3.100	5.000	ft
Au_Grade	4	0.034	0.022	0.017	0.494	0.015	0.053	oz Au/t

Composites Coded to Domain 161 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	49	3.576	4.000	1.581	0.442	0.200	5.000	ft
Au_Grade	49	0.029	0.021	0.036	1.242	0.000	0.250	oz Au/t

Composites Coded to Domain 200 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	37	3.059	2.500	1.539	0.503	0.500	5.000	ft
Au_Grade	37	0.17	0.123	0.156	0.915	0.066	0.813	oz Au/t

Composites Coded to Domain 202 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	9	3.778	4.500	1.679	0.444	0.500	5.000	ft
Au_Grade	9	0.133	0.116	0.056	0.425	0.012	0.226	oz Au/t

Composites Coded to Domain 203 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	10	4.45	5.000	1.165	0.262	2.000	5.000	ft
Au_Grade	10	0.11	0.110	0.039	0.355	0.046	0.171	oz Au/t

Composites Coded to Domain 205 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	14	3.829	4.750	1.587	0.415	0.500	5.000	ft
Au_Grade	14	0.18	0.122	0.201	1.114	0.053	0.801	oz Au/t

Composites Coded to Domain 206 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	15	3.433	3.700	1.277	0.372	1.300	5.000	ft
Au_Grade	15	0.178	0.142	0.081	0.454	0.075	0.355	oz Au/t

Composites Coded to Domain 207 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	4	4.000	1.414	0.354	3.000	5.000	ft
Au_Grade	2	0.14	0.135	0.006	0.044	0.135	0.147	oz Au/t

Composites Coded to Domain 209 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	5	3.6	5.000	1.949	0.541	1.000	5.000	ft
Au_Grade	5	0.11	0.136	0.033	0.296	0.072	0.141	oz Au/t

Composites Coded to Domain 210 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	3	3.733	5.000	2.194	0.588	1.200	5.000	ft
Au_Grade	3	0.132	0.130	0.011	0.086	0.127	0.163	oz Au/t

Composites Coded to Domain 220 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	7	2.757	2.500	1.371	0.497	1.000	5.000	ft
Au_Grade	7	0.253	0.102	0.341	1.350	0.079	1.021	oz Au/t

Composites Coded to Domain 221 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	8	2.563	2.250	1.178	0.460	1.000	5.000	ft
Au_Grade	8	0.111	0.122	0.066	0.594	0.017	0.218	oz Au/t

Composites Coded to Domain 223 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	24	4.313	5.000	1.301	0.302	1.000	5.000	ft
Au_Grade	24	0.108	0.105	0.045	0.422	0.001	0.208	oz Au/t

Composites Coded to Domain 225 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	3	2.333	2.000	0.577	0.247	2.000	3.000	ft
Au_Grade	3	0.153	0.154	0.078	0.509	0.048	0.222	oz Au/t

Composites Coded to Domain 226 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	14	3.179	3.000	1.552	0.488	1.000	5.000	ft
Au_Grade	14	0.19	0.147	0.181	0.953	0.086	0.887	oz Au/t

Composites Coded to Domain 228 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	6	3.167	4.000	2.206	0.697	0.500	5.000	ft
Au_Grade	6	0.346	0.162	0.456	1.319	0.096	2.500	oz Au/t

Composites Coded to Domain 229 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	4	3.5	4.000	1.915	0.547	1.000	5.000	ft
Au_Grade	4	0.107	0.106	0.009	0.079	0.099	0.129	oz Au/t

Composites Coded to Domain 230 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	2.5	2.500	0.707	0.283	2.000	3.000	ft
Au_Grade	2	0.198	0.195	0.004	0.019	0.195	0.202	oz Au/t

Composites Coded to Domain 231 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	17	3.265	3.000	1.470	0.450	0.500	5.000	ft
Au_Grade	17	0.131	0.122	0.081	0.616	0.015	0.437	oz Au/t

Composites Coded to Domain 234 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	5	3.6	3.000	0.894	0.248	3.000	5.000	ft
Au_Grade	5	0.142	0.123	0.064	0.450	0.082	0.239	oz Au/t

Composites Coded to Domain 237 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	5	3	2.000	1.620	0.540	1.500	5.000	ft
Au_Grade	5	0.156	0.181	0.044	0.279	0.092	0.193	oz Au/t

Composites Coded to Domain 238 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	5	2.98	3.000	1.569	0.527	1.300	5.000	ft
Au_Grade	5	0.15	0.112	0.072	0.482	0.112	0.343	oz Au/t

Composites Coded to Domain 240 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	88	2.712	2.500	1.252	0.462	0.300	5.000	ft
Au_Grade	88	0.103	0.096	0.056	0.537	0.010	0.239	oz Au/t

Composites Coded to Domain 241 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	8	2.5	3.000	1.225	0.490	0.500	4.000	ft
Au_Grade	8	0.36	0.246	0.490	1.364	0.068	1.755	oz Au/t

Composites Coded to Domain 242 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	172	3.426	3.600	1.505	0.439	0.500	5.000	ft
Au_Grade	172	0.124	0.112	0.071	0.568	0.004	0.467	oz Au/t

Composites Coded to Domain 243 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	53	3.025	2.500	1.436	0.475	0.800	5.000	ft
Au_Grade	53	0.124	0.115	0.083	0.671	0.002	0.500	oz Au/t

Composites Coded to Domain 244 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	3	2.8	2.000	1.929	0.689	1.400	5.000	ft
Au_Grade	3	0.146	0.116	0.075	0.512	0.109	0.302	oz Au/t

Composites Coded to Domain 245 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	10	3.65	4.250	1.510	0.414	1.000	5.000	ft
Au_Grade	10	0.114	0.111	0.048	0.418	0.067	0.221	oz Au/t

Composites Coded to Domain 246 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	6	1.833	1.650	0.731	0.399	0.900	3.000	ft
Au_Grade	6	0.113	0.148	0.066	0.583	0.020	0.195	oz Au/t

Composites Coded to Domain 248 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	4	4.000	1.414	0.354	3.000	5.000	ft
Au_Grade	2	0.089	0.085	0.004	0.049	0.085	0.094	oz Au/t

Composites Coded to Domain 250 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	61	3.239	3.000	1.332	0.411	1.000	5.000	ft
Au_Grade	61	0.127	0.111	0.061	0.485	0.000	0.451	oz Au/t

Composites Coded to Domain 251 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	33	3.273	3.000	1.538	0.470	0.600	5.000	ft
Au_Grade	33	0.177	0.147	0.114	0.647	0.067	0.681	oz Au/t

Composites Coded to Domain 260 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	3.85	3.850	1.626	0.422	2.700	5.000	ft
Au_Grade	2	0.097	0.094	0.003	0.035	0.094	0.101	oz Au/t

Composites Coded to Domain 261 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	15	2.667	2.500	1.577	0.592	0.500	5.000	ft
Au_Grade	15	0.111	0.096	0.055	0.496	0.001	0.236	oz Au/t

Composites Coded to Domain 302 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	2	4	4.000	1.414	0.354	3.000	5.000	ft
Au_Grade	2	0.383	0.371	0.017	0.043	0.371	0.403	oz Au/t

Composites Coded to Domain 303 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	6	3.117	3.500	1.524	0.489	1.000	5.000	ft
Au_Grade	6	0.738	0.714	0.347	0.470	0.371	1.622	oz Au/t

Composites Coded to Domain 306 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	8	3.688	4.000	1.361	0.369	2.000	5.000	ft
Au_Grade	8	1.114	1.036	0.521	0.467	0.318	1.723	oz Au/t

Composites Coded to Domain 309 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	3	4.667	5.000	0.577	0.124	4.000	5.000	ft
Au_Grade	3	0.341	0.312	0.071	0.209	0.267	0.430	oz Au/t

Composites Coded to Domain 321 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	7	2.286	2.000	0.756	0.331	2.000	4.000	ft
Au_Grade	7	0.658	0.392	0.583	0.886	0.000	1.527	oz Au/t

Composites Coded to Domain 323 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	6	1.75	1.500	1.368	0.782	0.300	4.200	ft
Au_Grade	6	0.617	0.270	0.653	1.059	0.222	2.723	oz Au/t

Composites Coded to Domain 330 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	4	2.5	3.000	1.000	0.400	1.000	3.000	ft
Au_Grade	4	2.289	2.319	1.215	0.531	1.068	3.867	oz Au/t

Composites Coded to Domain 340 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	169	2.954	2.800	1.439	0.487	0.500	5.000	ft
Au_Grade	169	1.243	0.803	1.499	1.206	0.022	11.190	oz Au/t

Composites Coded to Domain 342 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	99	3.333	3.500	1.534	0.460	0.200	5.000	ft
Au_Grade	99	0.81	0.573	0.736	0.908	0.120	5.000	oz Au/t

Composites Coded to Domain 343 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	44	2.091	1.900	1.116	0.534	0.500	5.000	ft
Au_Grade	44	1.18	0.602	1.555	1.318	0.073	6.000	oz Au/t

Composites Coded to Domain 346 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	8	1.462	1.350	0.870	0.595	0.500	3.000	ft
Au_Grade	8	1.093	0.670	1.179	1.079	0.223	3.870	oz Au/t

Composites Coded to Domain 350 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	30	3.183	3.000	1.222	0.384	1.000	5.000	ft
Au_Grade	30	0.471	0.310	0.328	0.696	0.180	1.864	oz Au/t

Composites Coded to Domain 351 (weighted)

Item	Count	Mean	Median	Std. Dev.	CV	Min.	Max.	Units
Length	4	3.825	3.650	0.888	0.232	3.000	5.000	ft
Au_Grade	4	0.896	0.911	0.071	0.079	0.799	0.980	oz Au/t

APPENDIX C
List of Mining Equipment and Buildings

Sutter Gold Mining Company Fixed Asset List

Maintenance Shop Equipment List

Qty.	Item
1	Bin Grade Five Nuts & Bolts
1	Clinton vise
1	Baldor #7 Grinder
2	Sets Victor Oxy-Acet Regulators
1	Norco 22 Ton Jack
1	Armstrong Chain Wrench
1	Drill Press Vice
1	Battery Charger
2	Wilton Vices
1	Bins Grade 8 Nuts & Bolts
1	Rigid BC-810 Chain Vice
1	Greenfield/ Little Giant Tap And Die Set
1	55 Gallon Hand Truck
4	Pair Nylon Lifting Straps
1	Oxy –Acet Bottle Cart
1	Ingersoll Rand #182 Needle Scaler
1	55 Ton Press
1	Jet Band Saw
1	Ton Puller Kit
1	Ingersoll Rand Air Compressor
1	UCIMU Saimp Metal Lathe
1	Acids/Corrosives Cabinet
1	Automotive Fluids Cabinet
1	Paint Cabinet
1	Coll Crimp Hose Machine
1	Miller Intelliweld 650 Welder
1	Welding Table
1	Welding Cabinet
2	Gray Shelving Units
1	Tool Bench
3	Air Hoses
1	Solvent Tank/ Parts Washer
1	Black and Decker Grinder

Under Ground Equipment List

Qty.	Item
1	50 H.P. Vent Fan
1	75 H.P. Vent Fan
1	500 KVA Transformer
2	Pemco Power Skids
5	13 H.P. Flygt Pumps
1	Complete Water Treatment Center
	3000' Of 36" Vent Pipe
	3000' Each Of 2", 4", & 6" Diameter Grooved Pipe

Vehicles

Qty.	Item
1	1996 Chevy Suburban
1	2001 Dodge Truck
1	1983 Dodge Caravan
1	JCI Model 1504 Haul Truck
1	Eimco 950 Mechanics Truck
1	Eimco 975 Flatbed Truck
1	John Deere Gator
1	Liftall Industrial Forklift
3	Wallace Personnel Vehicles (Leased by Tour)

Buildings

Qty.	Item
1	Shop Building
1	Gift Shop with Restrooms
1	Hi Resolution Theatre
1	Coach Change Trailer
1	Red Barn (10'x12')
1	Miners Dry (10'x12')
8	40' Sea Containers
1	Lundlee Property (furnished)

Miscellaneous

Qty.	Item
1	30 Mine Lamp Charger
30	Mine Lamps
1	Diesel Fuel Tank
1	Propane Tank
1	Used Oil Facility
1	Jaw Crusher
1	Roll Crusher
1	Steel Rack
3	Trailers
28	Sizing Sieves
8	Drägger Mine Rescue Gear Sets
1	Drägger Pressure Tester
1	Welding Curtain
1	¾ H.P. Dust Master
1	40 lb Accountogram
1	Balance Beam Scale
4	Vacuum Canisters
Misc. Vic Pipe	
Utilities/ Power Lines	
Misc. Geological Supplies	
Misc. Mine Supplies	

Appendix D

Quotations for Representative Equipment Costs

(From Danio, 2010)

Nelson Machinery & Equipment Ltd.

Mining & Mineral Process Equipment ♦Liquidations ♦Appraisals

1470 RUPERT STREET, NORTH VANCOUVER, BC V7J 1E9 PH.(604)985-5331 FAX (604)985-2074
VISIT US ON THE INTERNET! http://www.nelsonmachinery.com
e-mail: sales@nelsonmachinery.com

PLEASE DELIVER THIS QUOTATION TO:

ATTN: PAUL DANIO
DANIO & ASSOCIATES
DENVER. CO.
720-201-8887
pauldanio@hotmail.com

QUOTATION NUMBER: 17775

DATE: AUGUST 20, 2010

PAGES SENT: 4 OF 5

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
1	**PACKAGE CONSISTING OF (4) USED DENVER THICKENER MECHANISMS** - **JOY DENVER #60 GEAR BOXES** - **RATIO IS 90:1** - **0.1 RPM** - **5 HP DRIVES** - **HAND MECHANICAL LIFT** - **GEARS VERY GOOD** - **INCLUDES SHAFTS, RAKES, AND BRIDGE** - **WILL NOT SELL MECHANISMS SEPARATE** **PACKAGE PRICE, "AS IS" FOB SOUTH EAST USA…………………..**	**$ 175,000.00 PACKAGE PRICE U.S. FUNDS**
1	**USED 6' X 4' HARDINGE CONICAL BALL MILL** **C/W 75 HP MOTOR, C/W GOOD STRAIGHT CUT GEARS** **80 – 85% RUBBER LINERS ON CONE, UNIT HAS A STEEL FRAME AVAILABLE** **PRICE, "AS IS", FOB S.W. USA ……………………………………..**	**$ 59,500.00 U.S. FUNDS**
	Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	

FOB Point: AS NOTED	Delivery:	Signed:
Payment Terms: TO BE ARRANGED		**ROD ENGEL, SALES MANAGER**

Nelson Machinery & Equipment Ltd.

Mining & Mineral Process Equipment ♦Liquidations ♦Appraisals

1470 RUPERT STREET, NORTH VANCOUVER, BC V7J 1E9 PH.(604)985-5331 FAX (604)985-2074
VISIT US ON THE INTERNET! http://www.nelsonmachinery.com
e-mail: sales@nelsonmachinery.com

PLEASE DELIVER THIS QUOTATION TO:

ATTN: PAUL DANIO
DANIO & ASSOCIATES
DENVER. CO.
720-201-8887
pauldanio@hotmail.com

QUOTATION NUMBER: 17775

DATE: AUGUST 20, 2010

PAGES SENT: 5 OF 5

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
1	**USED 5' DIA X 10' MARCY ROD MILL** **MOUNTED ON STRUCTURAL STEEL BASE** **INCLUDING:** - **STEEL LINERS IN VERY GOOD CONDITION** - **100 HP MOTOR AND DRIVE** - **DRUM FEEDER** **REPORTED IN GOOD OVERALL CONDITION** **"AS IS" FOB, WESTERN USA.................. ....**	**$ 79,500.00** **U.S. FUNDS**
1	**USED 5' DIA X 10' HARDINGE ROD MILL (1940)** **INCLUDING:** - **CHUTE FEED, OVERFLOW DISCHARGE** - **STEEL LINERS** - **100 HP MOTOR** - **FALK REDUCERS** - **SOME SPARES** - **9 TON GRINDING RODS** **"AS IS" CONDITION, ON FOUNDATIONS** **WESTERN USA................................** **NOTE: REMOVAL ADDITIONAL AT COST**	**$ 49,500.00** **U.S. FUNDS**
	Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	

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pauldanio@hotmail.com

QUOTATION NUMBER: 17779

DATE: AUGUST 20, 2010

PAGES SENT: 1

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
1	**USED 3' X 8' TELSMITH, SINGLE DECK VIBRATING SCREEN C/W SUBFRAME, MOTOR AND DRIVE PRICE, "AS IS", FOB CALIFORNIA USA ....**	**$ 6,500.00 U.S. FUNDS**
1	**USED 10" X 36" PARKER, SINGLE TOGGLE OVERHEAD ECCENTRIC, ROLLER BEARING JAW CRUSHER REPORTED TO BE IN GOOD CONDITION PRICE, "AS IS", FOB EASTERN CANADA ....**	**TO FOLLOW**
1	**USED 18" X 24" UNIVERSAL SINGLE TOGGLE OVERHEAD ECCENTRIC, ROLLER BEARING JAW CRUSHER S/N 114X14 INCLUDING: 60HP MOTOR 480 VOLT 3PH 60 HZ PRICE, "AS IS", FOB POTOSI, MISSOURI ....**	**$ 27,500.00 U.S. FUNDS**
	Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	

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DENVER. CO.
720-201-8887
pauldanio@hotmail.com

QUOTATION NUMBER: 17775

DATE: AUGUST 20, 2010

PAGES SENT: 1 OF 5

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
1	REBUILT BANK OF 8, DENVER #21 SUB A FLOTATION CELLS INCLUDING: - NEW SINGLE OVERFLOW TANKS - REBUILT OR NEW AIR HEADER/AGITATOR SUPPORT BEAM SET UP FOR (4) DUAL DRIVES - NEW BOTTOM LINERS WITH ½" BONDED RUBBER TOP SURFACE - NEW BOTTOM SIDE LINERS (MILD STEEL) - NEW WEIR GATES WITH MANUAL HANDWHEEL ACTUATORS - FROTH PADDLE ASSEMBLY WITH MOTOR/REDUCER SLIP BELT DRIVE - NEW DEFLECTORS - (8) REBUILT AGITATOR / MECHANISMS WITH • RE-MACHINED BEARING AND SEAL SHAFT SURFACES • NEW OR REBUILT DRAFT TUBE AND HOOD ASSEMBLIES • NEW URETHANE IMPELLERS AND DIFFUSERS • 1 ½" AIR CONTROL VALVES - (4) NEW 15HP 1200 RPM TEFC MOTORS 480 VOLT 3PH 60HZ - (8) V-BELT DRIVES - (4) DUAL DRIVE BELT GUARDS - (1) COAT EPOXY PRIMER, (2) COATS EPOXY PAINT ON ALL SURFACES - 6 MONTH LIMITED WARRANTY EXCLUDING: - LAUNDERS (CLIENT TO ADVISE QUANTITY AND ARRANGEMENT) PRICE, FOB SAVONA, B.C.…………………… DELIVERY: 12-14 WEEKS Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	 $ 129,600.00 U.S. FUNDS NET

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Payment Terms: TO BE ARRANGED		ROD ENGEL, SALES MANAGER

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PLEASE DELIVER THIS QUOTATION TO:
ATTN: PAUL DANIO
DANIO & ASSOCIATES DENVER
COLORADO
220-201-8887
EMAIL:

QUOTATION NUMBER: 17783

DATE: August 23, 2010

PAGES SENT: 1

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
1	USED 2' X 3' DENVER BALL MILL - STEEL LINERS - OVERFLOW DISCHARGE - 7.5HP, 550V, 700RPM, ODP MOTOR - MOUNTED ON 6" H-BEAM FRAME PRICE, CHECKED OVER, CLEANED & PAINTED IN WORKING CONDITION, FOB SAVONA, B.C.	$ 15,500.00 U.S. FUNDS
1	USED 8' DIA X 7 DISC EIMCO AGI-DISC FILTER. INCLUDING - MOTORS & DRIVES EXCLUDING – VACUUM EQUIPMENT PRICE, "AS IS" FOB, SPOKANE, WA..	$17,500 U.S. FUNDS
	OPTION: PRICE REFURBISHED FOB, SAVONA, B.C..................................	$39,500.00 U.S. FUNDS
	Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	

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Payment Terms: TO BE ARRANGED		ROD ENGEL, SALES MANAGER

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ATTN: PAUL DANIO

DANIO & ASSOCIATES DENVER
220-201-8887
pauldanio@hotmail.com

QUOTATION NUMBER: 17790

DATE: AUGUST 25, 2010

PAGES SENT: 1

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
1	**USED TRIO 10" X 30" JAW CRUSHER** - STEEL FABRICATED STRESS RELIEVED MAINFRAME - REVERSIBLE CAST MANGANESE CRUSHER DIES - SPHERICAL SELF ALIGNING GREASE LUBRICATED ROLLER BEARINGS - 30HP ELECTRIC MOTOR AND V-BELT DRIVE WITH GUARD - 60% OR BETTER REMAINING ON JAW DIES - 8" FEED MAXIMUM SIZE - 23 TPH THROUGH THE CRUSHER SET AT 1" MIN CLOSED SIDE SETTING - 60 TPH THROUGH THE CRUSHER SET AT 3-1/2" MAX CLOSED SIDE SETTING - MODEL # PEX1030 PRICE, "AS IS" FOB WESTERN USA .... - SEE ATTACHED PHOTOS AS WELL AS TECH DATA Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	$ 21,500.00 U.S. FUNDS

FOB Point: AS NOTED	Delivery:	Signed:
Payment Terms: TO BE ARRANGED		BARRY PACCAGNAN, SALES REPRESENTATIVE

Nelson Machinery & Equipment Ltd.

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PLEASE DELIVER THIS QUOTATION TO:	
ATTN: PAUL DANIO DANIO & ASSOCIATES DENVER. CO. 720-201-8887 pauldanio@hotmail.com	QUOTATION NUMBER: 17775 DATE: AUGUST 20, 2010 PAGES SENT: 2 OF 5

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
1	**USED BANK OF 6 #18 SPECIAL DENVER FLOT CELLS** - **CELL TO CELL FLOW CONTROL GATES** - **OPEN STAND PIPES** - **24 CU. FT. PER CELL** - **LESS MOTORS** **REPORTED IN GOOD CONDITION** **PRICE, "AS IS", FOB SAVONA, B.C. ……………………………………..**	**$ 27,000.00** **U.S. FUNDS**
1	**REMANUFACTURED EIMCO AGIDISC, 4' DIAMETER X 3 DISC, ROTARY VACUUM FILTER** - **STAINLESS STEEL VAT, STAINLESS STEEL CENTERSHAFT, 11" STAINLESS STEEL VALVE HEAD, 11" - 75:1 MAIN GEAR REDUCER, SECONDARY REDUCER WITH VFD READY MOTOR (VFD NOT INCLUDED), STAINLESS STEEL SCRAPER ASSEMBLIES WITH UHMW-PE SCRAPER TIPS, STAINLESS STEEL AND POLYPROPYLENE GRID INSERT SECTORS (NO COVERS INCLUDED), STAINLESS STEEL RADIAL RIDS AND NUTS, STAINLESS STEEL SECTOR CLAMPS, SECTOR BAGS INSTALLED ON SECTORS** - **RECONDITIONED TO INCLUDE ALL NEW BEARINGS, BEARING LINERS, WEAR PLATE, MOTORS, SEALS,GASKETS, MULTIFLEX CONNECTORS, SCRAPER TIPS, BLAST CLEANED TO REMOVE ALL RESIDUES, EPOXY PAINTED CAST IRON COMPONENTS SUCH AS DRIVE HOUSING AND TRUNNION BEARING HOUSINGS**	
	Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	

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QUOTATION NUMBER: 17775

DATE: AUGUST 20, 2010

PAGES SENT: 3 OF 5

In response to your recent Inquiry we are pleased to offer the following:

Quantity	Description	Price
	PRICE, FOB PEARLAND, TEXAS .... - CRATE, WOODEN, FOR EXPORT SHIPPING AND STORAGE OF ABOVE 4' DIA. X 3 DISK FILTER, (CONTRACTED CRATING) @ COST PLUS 15%	$ 31,670.00 U.S. FUNDS
1	VACUUM SYSTEM FOR ABOVE INCLUDING: - NEW 700 ACFM VACUUM PUMP WITH MOTOR AND DRIVE PACKAGE, HDG CARBON STEEL SKID, NEW STAINLESS STEEL RECEIVER, NEW CARBON STEEL SEPARATOR/SILENCER, (NO FILTRATE PUMP), INTERCONNECTING VACUUM PIPING, VACUUM PUMP SEAL WATER SYSTEM PRICE, FOB PEARLAND, TEXAS	$ 25,300.00 U.S. FUNDS
1	KROGH FILTRATE PUMP PACKAGE #550H INCLUDING 1HP MOTOR, DRIVE & BELT GUARD PRICE, FOB PEARLAND, TEXAS	$ 3,960.00 U.S. FUNDS
	RE ABOVE 2 ITEMS: AVAILABILITY: 2-3 WEEKS A.R.O. AND DEPOSIT TERMS: 50% DEPOSIT / BALANCE FOLLOWING INSPECTION AT OUR FACILITY AND APPROVAL FOR SHIPMENT	
	Prices valid for 30 days unless otherwise specified. No warranties are implied or expressed unless otherwise noted above. Taxes extra where applicable All items quoted subject to prior sale or other disposition	

FOB Point: AS NOTED	Delivery:	Signed:
Payment Terms: TO BE ARRANGED		**ROD ENGEL, SALES MANAGER**

4	#380 X 10:1 Hub City Gear Boxes	$1023.65 ea
4	2hp 1800 TEFC Motors.	230.00
30'	#140 R/V Chain	39.[illegible] ft
4	#140 Conn. Links	8.96 ea
4	#140 offset	19.13 ea.
4	140 SF-13 Sprkt.	147.89
4	SF-1 1/2 Bushing	17.64 =
4	140 J-40 Sprkt.	408.65 =
4	J-3 15/16 Bushing	82.34 =
30'	#80 R/V Chain	9.54 ft =
4	80B-17 X ? Bore Sprkt. 2 3/4 Max Bore.	45.73 =
560'	24" 2ply 3/16 x 1/16 Conveyor Belt.	9.60 ft =
50'	24" 3ply 1/4 x 1/16	12.80 ft =

Any Questions, Pls Call
Fred Green
303-308-1290

4	V Belt drives.	$200.00 ea

4	= 16" X 26" CF-drum w/3/8 Herringbone Lagging E-2 7/16 Bushings	611.00 ea.
4	= 14" X 26" CF-Wing Pulleys w/ E-2 7/16 Bushings	498.00
75	B-4-20TE-24-SB Idlers	$85.17 ea
30	B-4-R-24 w/4 1/2 drops Returns.	$48.66 EA
4	PWS-208-24 Takeups w/2 7/16 Bearings.	249.00 EA
2	12" dia X 26" CF drums w/3/8" Herringbone Lagging E-2 7/16"	$511.00 ea.
2	12" dia X 26" CF Wing Pulleys w/E-2 7/16"	$447.00 EA
	2 7/16" SP Keyed Shaft	$49.00/Ft =
	2 7/16 ✓ ✓ No-Key Shaft.	$38.00/Ft =
8	2 7/16 Pillow-Blocks	$101.18 EA =
4	V-Belt Guards.	$233.00 EA.
4	TXT-425 25:1 Gear Box	$1396.00 ea
4	TXT-4X 2 7/16 Bushings	$158.00 ea
4	TA-4 Motor Mounts	$298.00 ea
4	10HP 1800 TEFC motors.	$449.00 EA.
100'	1/2" X 6" Skirtboard Rubber	2.52 Ft.

1