MINE DEVELOPMENT ASSOCIATES
A Division of RESPEC

Estimated Mineral Resources and Preliminary Economic Analysis, Strong and Harris Copper-Zinc-Silver Project, Cochise County, Arizona

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1.0 SUMMARY (ITEM 1)

Mine Development Associates, a Division of RESPEC ("MDA") has prepared this technical report and Preliminary Economic Assessment ("PEA") on the Strong and Harris copper-zinc-silver deposit, located in Cochise County, Arizona, at the request of Excelsior Mining Corp. ("Excelsior"), a Canadian company listed on the Toronto Stock Exchange (TSX:MIN) and the OTC Markets (OTCQX: EXMGF).

This report has been prepared under the supervision of Jeff Bickel, C.P.G. and Senior Geologist for MDA, Michael M. Gustin, C.P.G. and Senior Geologist for MDA, Thomas L. Dyer, P.E. and Senior Engineer for MDA, and Robert ("Rob") Bowell, Ph.D., C.Chem., C.Geol., and F.I.M.M.M., corporate consultant in geochemistry with SRK Consulting, in accordance with the 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 amended. Mr. Bickel, Mr. Gustin, Mr. Dyer, and Mr. Bowell are Qualified Persons under NI 43-101 and have no affiliation with Excelsior except that of independent consultant/client relationships.

The Effective Date of this technical report is September 9, 2021.

1.1 Property Description and Ownership

The Strong and Harris project consists of 35 patented mining claims, 113 unpatented claims, and two parcels of fee land that together cover approximately 2,255 acres in Sections 13, 14, 23 and 24 of Township 15S, Range 22E, Cochise County, Arizona. The current annual holding costs for the Strong and Harris project are estimated at $18,870. Greenstone Excelsior Holdings L.P. ("Greenstone") holds a 3.0% gross revenue royalty over the Strong and Harris project. Royal Crescent Valley, Inc. ("Royal Crescent") holds a 2.5% net smelter returns ("NSR") royalty interest in minerals produced and sold from the 35 patented claims. These 35 patented claims are also subject to Production Payment Agreements that provide for a non-participating payment of $0.02 per pound of copper payable when copper prices are in excess of $1.00 per pound, and is capped at an aggregate of $1,000,000, of which $416,435 has been paid as of August 12, 2021. A portion of the Strong and Harris project is also subject to a Metal Stream Agreement with Triple Flag Mining Finance Bermuda Ltd. ("Triple Flag") that is applicable to all oxide minerals production.

1.2 Exploration and Mining History

Modern-era exploration of the Strong and Harris project commenced in 1964. More than 100,000 feet of rotary and core drilling were done from the mid-1960s through 1992 by the Superior Oil Company ("Superior"), Cyprus Mining ("Cyprus"), Continental Exploration ("Continental"), Continental Materials, Beard Mining Company ("Beard"), AZCO and Manazanita Hills Inc. ("Manzanita"). There has been no historical production from the Strong and Harris deposit. Excelsior purchased the property in 2019.

1.3 Geology and Mineralization

The Strong and Harris copper-zinc-silver deposit is situated within the Cochise mining district in an alluvium-covered valley on the east side of the Little Dragoon Mountains. Beneath the alluvium, mineralized Paleozoic rocks below the valley fill sediments strike approximately 315^{o} azimuth and dip 30^{o} to 45^{o} northeast. These Paleozoic rocks include the Carboniferous Escabrosa Limestone, the Carboniferous Black Prince Limestone, the Carboniferous to Permian Horquilla Limestone, the Permian Earp Formation, and the Permian Colina Limestone. The Horquilla Limestone is intruded by a thin mafic sill in the area of Strong and Harris known locally as the "Peabody Sill".

The Strong and Harris mineralization is a sub-type of, or related to, a copper skarn system. Copper-zinc-silver mineralization at Strong and Harris is characterized by lenses of sulfide minerals emplaced more-or-less parallel to layering in favorable lithologic units, usually along bedding planes or in disseminated masses and blebs. The mineralization is typically accompanied by calc-silicate alteration of the carbonate host-rock. Less frequently, the mineralization is hosted in quartz +/- calcite +/- feldspar veins. Sub-units of the Earp Formation, particularly those immediately below its upper contact with the Colina Limestone, were the most favorable sites for deposition of the copper, zinc and silver minerals. The Strong and Harris deposit has been oxidized to varying degrees that generally decrease with depth, resulting in oxide, transitional and sulfide zones or geometallurgical material types.

1.4 Metallurgical Testing and Mineral Processing

The metallurgical testwork and assessment of mineral processing options are at a preliminary state of study. Historical testwork and analogue studies from mineralization in other parts of the district have been used to develop a conceptual flowsheet for the sulfide, transitional and oxide types of mineralized material.

Copper, zinc and potentially silver will be produced from the sulfide mineralized material, and copper and possibly zinc from the oxide mineralized material. Transitional material will be sent to whichever process option appears the most beneficial for that mineralized material.

Oxide mineralized material will be processed through sulfuric acid heap leaching. Copper and zinc from this material will be recovered by solvent extraction process.

Sulfide mineralized material will be milled to produce a concentrate by conventional methods using two-stage crushing, then grinding, followed by a two-stage flotation approach to produce copper concentrate (incorporating silver credit) and zinc concentrate. Metal recoveries are expected to be in the range of:

- Heap leach copper recovery of 92.3%;
- Copper flotation recovery of 80.1%;
- Zinc heap leach recovery of 83.3%; and
- Flotation zinc recovery of 69.7%;

Silver recovery is considered to be 0% for both leach and flotation processes, but is likely to provide a silver credit in the sulfide concentrates.

1.5 Mineral Resource Estimate

MDA constructed stratigraphic interpretations on a set of vertical, digital cross sections oriented at 045° azimuth through the Strong and Harris deposit. These sections were spaced at 200-foot intervals over a strike extent of 10,000 feet. MDA also interpreted oxidation domains on the cross sections using logging data and the ratio of soluble copper assays ("CuOx") to total copper assays ("Cu"). The mineralization was assigned to oxide, transitional, or sulfide material types (domains).

Low-, mid-, and high-grade mineral domains were modeled for each metal to respect the lithologic, structural, and oxidation interpretations of the deposit. The 200-foot-spaced cross-sectional mineral-domain polygons were used to code 20 x 20 x 20 (x, y, z)-foot blocks that comprise a digital model rotated to a bearing of 315°. MDA used estimated ratios to code the Strong and Harris block model with soluble copper values. The mineralization has a variety of orientations. Wireframe solids were therefore created to encompass model areas with similar mineral domain orientations, and the solids were used to code the model blocks to these areas on a block-in/block-out basis.

Copper, zinc, and silver grades, as well as soluble copper ratios, were interpolated using inverse distance, ordinary kriging, and nearest neighbor methods. The mineral resources reported herein were estimated by inverse distance interpolation as this method led to results that most appropriately respected the drill data and geology of the deposit.

The Strong and Harris project mineral resources (Table 1.1) have been estimated to reflect potential open-pit extraction and potential processing by heap leaching and concentration, depending on the oxidation zone of the mineralization. The estimated resources are entirely classifies as Inferred. The classification is based on the confidence in the underlying data which are largely historical. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1.1 Strong and Harris Mineral Resources
(0.1% Cu cutoff)

Classification	Tons	% Cu	% CuOx	% Zn	oz Ag/ton	lbs Cu	lbs CuOx	lbs Zn	oz Ag
Inferred	76,161,000	0.52	0.33	0.56	0.12	794,049,000	500,155,000	858,425,000	9,515,000

1. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
2. The estimate of mineral resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
3. Rounding as required by reporting guidelines may result in apparent discrepancies between tons, grade, and contained metal content.

1.6 Preliminary Economic Assessment

The preliminary economic assessment ("PEA") presented in this report considers open-pit mining of the Strong and Harris copper-zinc-silver deposits. A PEA is preliminary in nature and 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 resources. There is no certainty that the conclusions reached in the PEA will be realized. Mineral resources that are not mineral reserves do not have the demonstrated economic viability.

The PEA economic evaluation results in a mine life of about seven years; anticipates approximately 54 million tons processed with average of 0.56% copper and 0.68% zinc grades; $328 million in initial capital costs; operating costs of $1.76 per pound of equivalent copper; and an average annual production of 62 million pounds of copper and 82 million pounds of zinc. The cash-flow model results in $325,466,000 in pre-tax NPV (at discount rate of 8%) with an after-tax NPV of $186,958,000 (8%) and a 19% IRR. Payback on initial investment is estimated to be 3.1 years.

1.7 Conclusions and Recommendations

The authors conclude that Strong and Harris is a project of merit that warrants further investment to take the project to the next level. Exploration potential for additional bulk-tonnage mineralization at the Strong and Harris project remains significant. There is an excellent opportunity to expand the extents of the current resources both down dip and along strike.

The authors recommend a work program with an estimated cost of $4.91 million as summarized in Table 1.2. Drilling should be a major component of future expenditures, including infill drilling to obtain samples for the metallurgical program, and step-out drilling to expand the current resources and test peripheral targets. Comprehensive metallurgical test work should also be part of future expenditures.

Table 1.2 Excelsior Cost Estimate for the Recommended Program

Item	Estimated Cost US$
Exploration Core Drilling (6, 000 feet)	$720,000
Infill Core Drilling (20,000 feet)	$2,500,000
Metallurgical / Infill Core Drilling (6,000 feet)	$720,000
Geological Modeling	$50,000
Land Holding Costs	$20,000
Metallurgy	$500,000
Geotechnical Studies	$200,000
Resource Update and Technical Report	$200,000
Total	$4,910,000

2.0 INTRODUCTION AND TERMS OF REFERENCE (ITEM 2)

2.1 Project Scope and Terms of Reference

The purpose of this report is to provide a technical summary of the Strong and Harris copper-zinc-silver project, including a maiden estimate of the mineral resources, in support of a preliminary economic assessment ("PEA"), at the request of Excelsior Mining Corp. ("Excelsior"). The Strong and Harris project lies within the historic Cochise mining district of southeastern Arizona, a few miles north of Excelsior's Gunnison in-situ leach copper mine. Excelsior is a publicly-traded Canadian company with its head office located in Phoenix, Arizona (TSX: MIN; OTCQX: EXMGF; FSE: 3XS).

The Strong and Harris project occupies the northern portion of Excelsior's Johnson Camp property, which is contiguous with Excelsior's Gunnison property. Historical copper production from open-pit operations occurred at the Johnson Camp mine from 1975, most recently by Nord Resources Corporation in 2008 until 2010. In 2020, Excelsior commenced in-situ recovery ("ISR") mining of oxidized copper deposits from the Gunnison portion of the property using conventional solvent extraction-electrowinning ("SX-EW") technology. The copper-bearing fluids are pumped to facilities at the Johnson Camp mine for SX-EW processing. The ISR mining and SX-EW extraction of oxide copper from the Gunnison deposits has been summarized in the feasibility study ("FS") of Zimmerman et al. (2016) prepared by M3 of Tucson Arizona. The Strong and Harris deposit consists of copper-zinc-silver sulfide mineralization that cannot be processed in the Johnson Camp facilities, would require separate infrastructure and will not be developed using common infrastructure located at Johnson Camp or Gunnison. Because of the need for separate infrastructure, Excelsior considers the Strong and Harris project to be located within a separate but contiguous mineral property here and subsequently termed the "Strong and Harris project".

The mineral resources of this technical report were estimated and classified under the supervision of Mr. Jeffrey Bickel, Senior Staff Geologist with Mine Development Associates ("MDA"), a division of RESPEC located in Reno, Nevada. Mr. Bickel is a qualified person under Canadian National Instrument 43-101 ("NI 43-101") and has had no affiliations with the property, Excelsior, or Excelsior's subsidiaries, except that of an independent consultant/client relationship since February of 2020. From 2011 to 2020 Mr. Bickel was employed by Excelsior as a Senior Project Geologist and Technical Services Manager, and Mr. Bickel worked on the Strong and Harris portion of the property, as well as the Johnson Camp and Gunnison properties during that time. Mr. Michael M. Gustin, C.P.G. and Senior Geologist for MDA, co-authored Sections 7 and 14 of this report. Mr. Gustin is a qualified person under NI 43-101 and has had no affiliations with the property, Excelsior, or Excelsior's subsidiaries, except that of an independent consultant/client relationship

The mineral resources reported herein are estimated to the standards and requirements stipulated in accordance with NI 43-101 and are considered the current mineral resources for the Strong and Harris project. These estimated mineral resources do not supersede the estimated reserves and resources for the Gunnison property reported by Zimmerman et al (2016) which is currently operating as a separate project that does not share common infrastructure with the Strong and Harris project.

Mr. Thomas L. Dyer, Senior Engineer with MDA, has supervised the preparation of the PEA presented in Sections 15, 16, and 18 through 22 of this report. Mr. Dyer is a qualified person under NI 43-101 and has no affiliations with the property, Excelsior, or Excelsior's subsidiaries, except that of an independent consultant/client relationship.

Section 13 (Mineral Processing and Metallurgical Testing) was prepared by Mr. Robert ("Rob") Bowell, Ph.D., C.Chem., C.Geol., and F.I.M.M.M., corporate consultant in geochemistry with SRK Consulting. Mr. Bowell is a qualified person under NI 43-101 and has had no affiliations with the property, Excelsior, or Excelsior's subsidiaries, except that of an independent consultant/client relationship. Mr. Bowell was assisted by Mr. W.J. Schlitt, a metallurgical consultant based in Knightsen, California. Mr. Bowell takes responsibility for Section 17 (Recovery Methods) which was prepared by Mr. Schlitt.

The scope of this study included a review of pertinent technical reports and data provided to MDA by Excelsior relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, drilling programs, and metallurgy. This report is based almost entirely on data and information derived from work done by historical operators and Excelsior. The authors have reviewed much of the available data and have made judgments about the general reliability of the underlying data. Where deemed either inadequate or unreliable, the data were either eliminated from use or procedures were modified to account for lack of confidence in suspect information. The authors have made such independent investigations as deemed necessary in their professional judgment to be able to reasonably present the conclusions, interpretations, and recommendations presented herein.

For this report, Mr. Bickel visited the Strong and Harris project beginning in 2020 and most recently on multiple occasions between February 12 and March 26, 2021. These site visits included... Mr. Dyer visited the project site on March 18 and March 19, 2021. Mr. Dyer observed current conditions at the Strong and Harris project, visited the Johnson Camp mine and discussed potential locations of site infrastructure and access with Excelsior management.

Mr. Bowell visited the site on the 27th to 29th of September 2021.

The Effective Date of the resource estimate, the PEA and this technical report is September 9, 2021.

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

In this report, measurements are generally reported in Imperial units (U.S. Customary system of units). Where information was originally reported in metric units, MDA has made the conversions as shown below. In some cases of historical data, originally reported metric units are reported without conversion in order to avoid changes to precision due to rounding.

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

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	

Frequently used acronyms and abbreviations

AA	atomic absorption spectrometry
Ag	silver
Au	gold
CaO	Calcium oxide
CF	cash flow
CFR	code of federal regulations
cm	centimeters
CN	cyanide
cog	cutoff grade
con	concentrate
core	diamond core-drilling method
CRM	certified reference material
Cu	copper

CuOx	soluble copper
CV	coefficient of variation
ºC	degrees centigrade
°F	degrees Fahrenheit
Fe	iron
ft	foot or feet
g	grams
gpd	gallons per day
GPS	global positioning system
g/L	grams per liter
g/t	grams per tonne
G&A	general and administrative
ICP	inductively coupled plasma analytical method
in.	inch or inches
IRR	internal rate of return
ISR	in-situ recovery
K	thousands
kg	kilograms
km	kilometers
KTon	thousand tons
kV	kilovolts
l	liter
lbs	pounds
LOM	life of mine
µm	micron
m	meters
Ma	million years old
max	maximum
mi	mile or miles
min	minimum
mL	milliliters
mm	millimeters
Na	sodium
NaF	sodium fluoride
NPV	net present value
NSR	net smelter return
oz	ounce
PEA	preliminary economic assessment
pH	negative of the log of hydrogen ion concentration
ppm	parts per million
ppb	parts per billion
QA/QC	quality assurance and quality control

RC	reverse-circulation drilling method
Rec	recovery
ROM	run of mine
RQD	rock-quality designation
SO_2	sulfer dioxide
SX-EW	solvent extraction electrowinning
SX	solvent extraction
st dev	standard deviation
t	metric tonne or tonnes
TPY	nons per year
ton	Imperial short to
Yr	year
Zn	zinc

3.0 RELIANCE ON OTHER EXPERTS (ITEM 3)

Mr. Bickel and Mr. Dyer are not experts in legal matters, such as the assessment of the validity of mining claims, mineral rights, and property agreements in the United States or elsewhere. Furthermore, the authors did not conduct any investigations of the environmental, social, or political issues associated with the Strong and Harris project and are not experts with respect to these matters. The authors have therefore relied fully upon information and opinions provided by Excelsior and Mr. Roland Goodgame, Senior Vice President for Business Development ("SVPBD") at Excelsior, with regards to the following:

- Section 4.2, 4.3, and 4.4, which pertain to land and mineral tenure were provided by Mr. Goodgame in a document received via email from Mr. Goodgame titled "Section 4" and dated September 2, 2021; and
- Section 4.5, which pertains to legal agreements and encumbrances.

The authors have relied fully upon information and opinions provided by Excelsior's environmental expert, Mrs. Cindi Byrns, with respect to environmental and permitting matters. Sections 4.6, 4.7 and 0, which pertain to environmental liabilities and permits, were provided by Mrs. Cindi Byrns in a project communication via email on June 30, 2021.

The authors have fully relied on Excelsior to provide complete information concerning the pertinent legal status of Excelsior and its affiliates, as well as current legal title, material terms of all agreements, and material environmental and permitting information that pertains to the Strong and Harris project.

4.0 PROPERTY DESCRIPTION AND LOCATION (ITEM 4)

The authors are not experts in land, legal, environmental, and permitting matters and express no opinion regarding these topics as they pertain to the Strong and Harris project. Subsections 4.2, 4.3, 4.4 and 4.5 were prepared under the supervision of Mr. Roland Goodgame, Excelsior's Senior Vice-President for Business Development (see Section 3.0). Mrs. Cindi Byrns of Excelsior, an expert in environmental and permitting matters, prepared Sections 4.6. and 4.7.

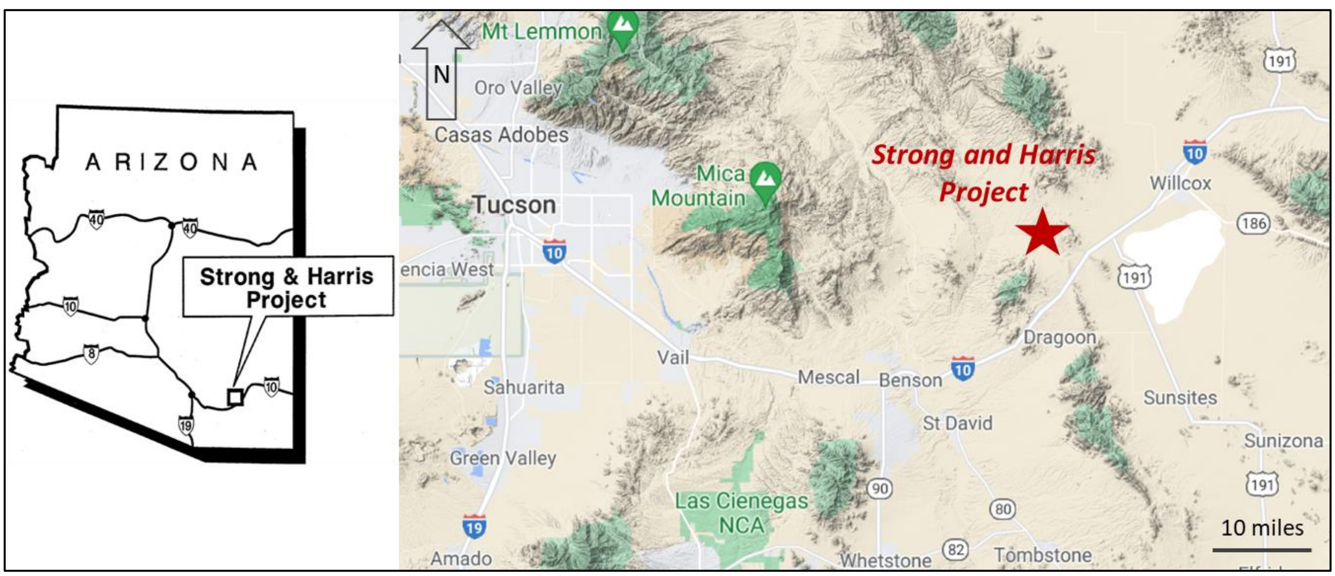
Because of the need for separate infrastructure, Excelsior considers the Strong and Harris project to be located within a separate but contiguous mineral property termed the "Strong and Harris project". The Strong and Harris project is held by Excelsior through its wholly owned subsidiaries Excelsior Mining Arizona, Inc. ("Excelsior Arizona") and Excelsior Mining Holdings, Inc. ("Excelsior Holdings"). In December 2015 Excelsior purchased all assets of Nord Resources Corporation, as they relate to the Johnson Camp property, through a court-appointed receiver. Additional, in October 2019 Excelsior purchased the Strong and Harris claims from the Strong & Harris Trust.

Mr. Bickel and Mr. Dyer do not know of any significant factors and risks that may affect access, title, or the right or ability to perform work on the property, beyond what is described in this report.

4.1 Location

The Strong and Harris project is located in Cochise County, Arizona approximately 65 miles east of Tucson, Arizona (Figure 4.1). The property is centered on geographic coordinates of 32º 07' 41" North latitude and 110º 03' 44" West longitude.

Figure 4.1 Location Map for the Strong and Harris Project

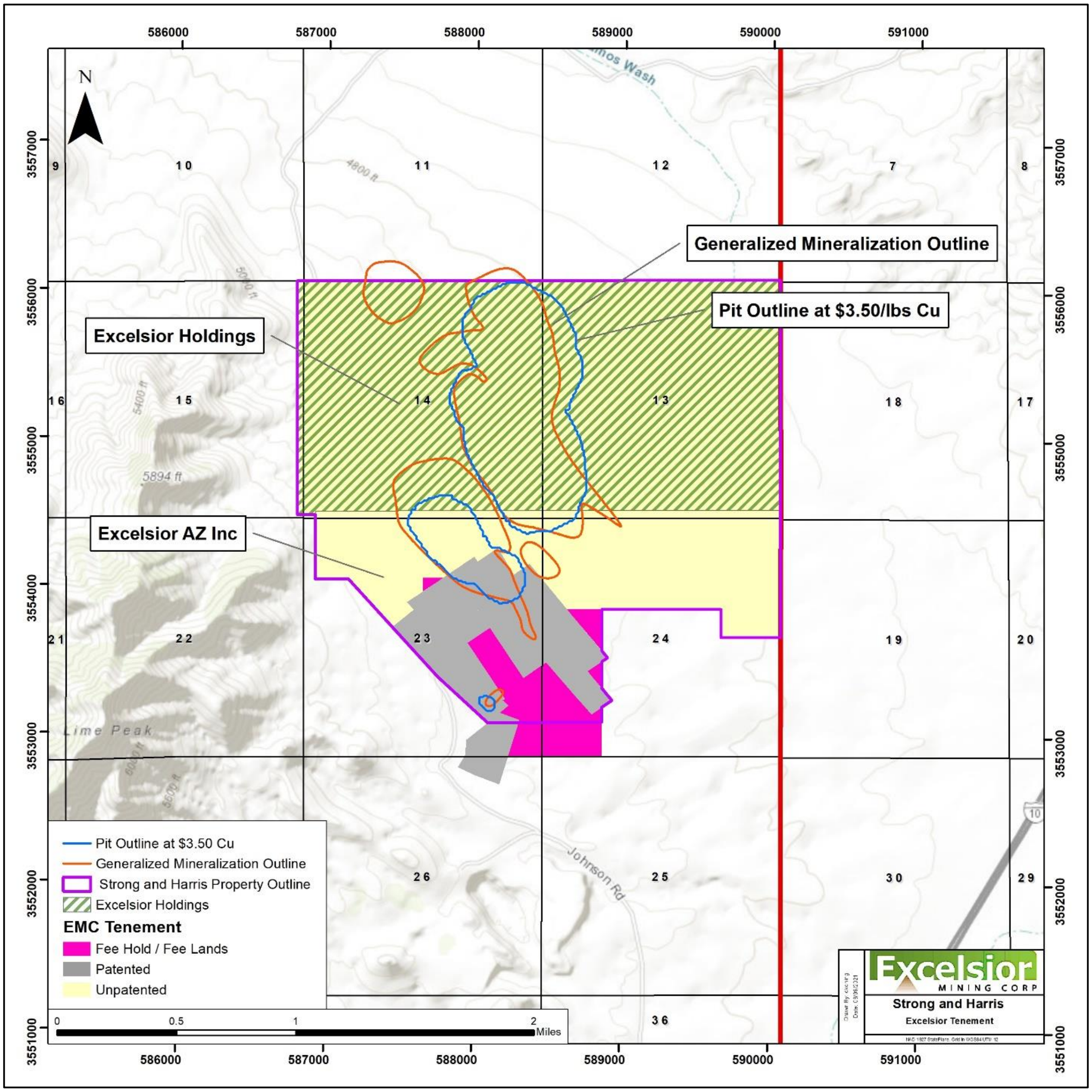


4.2 Land Area

The Strong and Harris project is shown in Figure 4.2. The property consists of 35 patented mining claims, 113 unpatented claims, and two parcels of fee land that together cover approximately 2,255 acres in Sections 13, 14, 23 and 24 of Township 15S, Range 22E, Gila-Salt River Meridian. A listing of the unpatented and patented mining claims is given in Appendix A.

Figure 4.2 Strong and Harris Project Map

(from Excelsior, September, 2021)



Note: coordinates given in UTM NAD27, Arizona State Plane East, 1,000 foot intervals.

Ownership of the unpatented mining claims is in the name of the holder (locator), subject to the paramount title of the United States of America, under the administration of the U.S. Bureau of Land Management ("BLM"). Under the Mining Law of 1872, which governs the location of unpatented mining claims on federal lands, the locator has the right to explore, develop, and mine minerals on unpatented mining claims without payments of production royalties to the U.S. government, subject to the surface management regulation of the BLM. As of the Effective Date, annual claim-maintenance fees are the only federal payments related to unpatented mining claims, and Excelsior represents these fees have been paid in full to September 1, 2022. The current annual holding costs for the Strong and Harris project are estimated at $18,870 (Table 4.1), including the county recording fees.

Table 4.1 Estimated Annual Holding Costs for the Strong and Harris Project

Claim Type	Quantity	Approximate Area	Annual Holding Costs	Surface Rights
Federal Patented Lode Mining Claims	35	243 acres	$444	Controlled by Excelsior
Federal Unpatented Mining Claims	113	1,908 acres	$18,315	Subject to US mining law
Fee Land Parcels	2	104 acres	$111	Controlled by Excelsior
Total	**150**	**2,255 acres**	**$18,870**	

Excelsior has rights to use the surface of the project that is in the form of federal patented lode mining claims and fee land parcels. The federal unpatented claims grant surface access but do not provide for surface ownership. However, surface rights on the property pose no problem to development. Unpatented mining claims give the owner the right to develop and exploit valuable minerals contained within the claim, so long as the claim is properly located and validly maintained.

4.3 Patented Mining and Unpatented Mining Claims

The property includes 35 patented mining claims held by Excelsior Arizona totaling 243 acres as listed list in Appendix A. The patented claims include full surface and mineral rights, subject to State and Federal environmental regulations. The claims are located on the ground and have no expiration dates.

A total of 77 unpatented mining claims are held by Excelsior Holdings that cover 1,296 acres, and 36 unpatented mining claims are held by Excelsior Arizona that cover 612 acres. A list of the unpatented claims is provided in Appendix A. The unpatented claims are for minerals only, with no surface ownership. The BLM requires that all unpatented claims use a rental year from September 1 through August 31; claims for which fees are not paid by August 31st are automatically forfeit. The claims otherwise have no expiration dates and under current mining law can be held indefinitely if properly maintained. The claims are located on the ground and the location descriptions are filed with the BLM..

4.4 Fee Land

There are two parcels of fee land, known as Parcel 3 and Parcel 4, all situated in Township 15S, Range 22E. Parcel 3 is situated in Section 24 and covers approximately 7.4 acres. Parcel 4 is situated in Sections 23 and 24, and covers approximately 96.7 acres.

4.5 Agreements and Encumbrances

Greenstone Royalty: Greenstone Excelsior Holdings L.P. ("Greenstone") holds a 3.0% gross revenue royalty over the Strong and Harris project. The gross revenue royalty is defined as royalty percentage times receipts, which is the sum of physical product receipts and deemed receipts. The Greenstone royalty applies to the entirety of the Strong and Harris project and production therefrom.

Sections 23 and 24: The following royalties and stream only apply to the portion of the Strong and Harris project that is located in land Sections 23 and 24, and production therefrom.

Royal Crescent Valley, Inc. ("Royal Crescent") holds a 2.5% net smelter returns ("NSR") royalty interest in minerals produced and sold from the 15 patented claims. These 15 patented claims are also subject to the terms of a "Royalty Deed and Assignment of Royalty," recorded with the Cochise County Recorder's Office on June 19, 2009, at No. 2009-14847, and the "Grant of Production Payment" recorded with the Cochise County Recorder's Office on June 10, 1999 at No. 1999-18419, as modified by a certain "Assignment of Production Payment" between Arimetco, Inc. and Styx Partners, L.P. (collectively, the "Production Payment Agreements"). The Production Payment Agreements provide for a non-participating payment of $0.02 per pound out of production during the calendar month in which copper produced from the 15 patented claims. The production payment is only payable when copper prices are in excess of $1.00 per pound and is capped at an aggregate of $1,000,000, of which $416,435 has been paid as of August 12, 2021.

The Strong and Harris project is also subject to a Metal Stream Agreement with Triple Flag Mining Finance Bermuda Ltd. ("Triple Flag") that is applicable to all oxide minerals production from the parts of the project located in the "Stream Area". The "Stream Area" includes the parts of the project located in Sections 23, 24 and 26 only. The Metal Stream Agreement is summarized in Table 4.2.

4.6 Environmental Liabilities

As of the Effective Date, there are no known environmental liabilities for the Strong and Harris project.

4.7 Environmental Permitting

The Strong and Harris project operations will require a number of permits that are identified and discussed in Section 0.

Table 4.2 Triple Flag Metal Stream Agreement for Strong and Harris

<table>
<tr><td>Stream Deliveries</td><td>Excelsior Mining Arizona Inc. (“Seller”) is required to deliver Grade A Copper Cathodes in an amount equal to the “Payable Copper”. The amount of Payable Copper is calculated based on a percentage of the amount of copper that is sold and delivered to Offtakers under the terms of Offtake Agreements (for percentages see heading – Payable Copper).</td></tr>
<tr><td>Payment</td><td>The Buyer pays to the Seller a price for copper equal to 25% of the daily official LME Grade A Settlement quotation for copper quoted in U.S. Dollars, as published in the Metal Bulletin.</td></tr>
<tr><td>Payable Copper</td><td>“Payable Copper” means a percentage of the Reference Copper equal to:
<table>
<tr><th>Scenario</th><th>Stage 1 (25 Mlbspa)</th><th>Stage 2 (75 Mlbspa)</th><th>Stage 3 (125 Mlbspa)</th></tr>
<tr><td>Upfront Deposit</td><td>16.5%</td><td>5.75%</td><td>3.5%</td></tr>
<tr><td>Upfront Deposit + Expansion Option</td><td>16.5%</td><td>11.0%</td><td>6.0%</td></tr>
<tr><td>Upfront Deposit + Expansion Option + Buy-Down Right</td><td>16.5%</td><td>5.5%</td><td>3.3%</td></tr>
<tr><td>Upfront Deposit + Buy-Down Right</td><td>16.5%</td><td>2.875%</td><td>1.75%</td></tr>
</table>
At the current stage of the project, the Buyer has made the initial Upfront Deposit ($65 million) and the Seller is ramping up to 25 Mlbspa.
The “Expansion Option” provides Buyer the option to invest an additional $65 million in the event Seller approves an expansion to at least 50 Mlbspa.
The “Buy-Down Right” provides the Seller an option to reduce the amount of the stream by 50% through the payment of the “Buy-Down Amount” which is equal to an aggregate amount that would need to be paid to the Buyer, after taking into account 50% of all other payments made by the Seller to Buyer (including the value of Deliveries net of payments made by the Buyer to the Seller) to yield an internal rate of return of 15% on 50% of the Upfront Deposit (assumed to occur on the Closing Date) and 15% on 50% of the Expansion Upfront Deposit.</td></tr>
</table>

4.8 Other Significant Risk Factors

There are no other known significant factors or risks that may affect access, title, or the right or ability to perform work on the property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY (ITEM 5)

The information summarized in this section is derived from publicly available sources, as cited. The authors have reviewed this information and believe this summary is materially accurate.

5.1 Access to Property

The Strong and Harris project is located in a sparsely populated, flat to slightly undulating ranching and mining area about 65 road miles east of Tucson, Arizona. Access is via the Interstate10 (I-10) freeway from Tucson and Benson in the west, or Wilcox in the east (Figure 4.1). From I-10 the property is reached via the improved unpaved Johnson Road travelling approximately 3.5 miles north from I-10.

The surface rights as described in Section 4 are sufficient for the mining and exploration activities proposed in this report.

5.2 Climate

The climate varies with elevation, but in general the summers are hot and dry and winters are mild. The area experiences two rain seasons in general, one during the winter months of December to March and a second summer season from July through mid-September. The summer rains are typical afternoon thunderstorms that can be locally heavy. Average annual rainfall for Dragoon is 13.2 inches and the average highs range from 58ºF in January to 94º F in June. Occasional light snow falls at higher elevations in the winter months. Mining and exploration can be conducted year round.

5.3 Physiography and Vegetation

Elevations range from 4,750 to 5,100 feet above mean sea level. Vegetation is typical of the upper Sonoran Desert and includes bunchgrasses, yucca, mesquite and cacti (Figure 5.1).

Figure 5.1 View East Across Strong and Harris Project

5.4 Local Resources and Infrastructure

The Tucson metropolitan area is a major population center (approximately 1,000,000 persons) with a major airport and transportation hub including well developed infrastructure (highways and rail) and services that support the surrounding copper mining industry. The towns of Benson and Wilcox are nearby and combined with Tucson can supply sufficient skilled labor for the project. There are sufficient areas for potential mine waste rock and tailings storage facilities, as well as areas for potential process facilities, mine offices and mine maintenance buildings. The Strong and Harris copper-zinc-silver mineralized material will require separate and much different process infrastructure from that available at Excelsior's Johnson Camp open-pit and solvent extraction – electrowinning ("SX-EW") facilities. There is a nearby 69 kV electric powerline and multiple water wells.

6.0 HISTORY (ITEM 6)

The information summarized in this section has been extracted and modified to a significant extent from Zimmerman et al. (2016), sources therein, unpublished company files, as well as other sources as cited. The authors have reviewed this information and believe this summary is materially accurate.

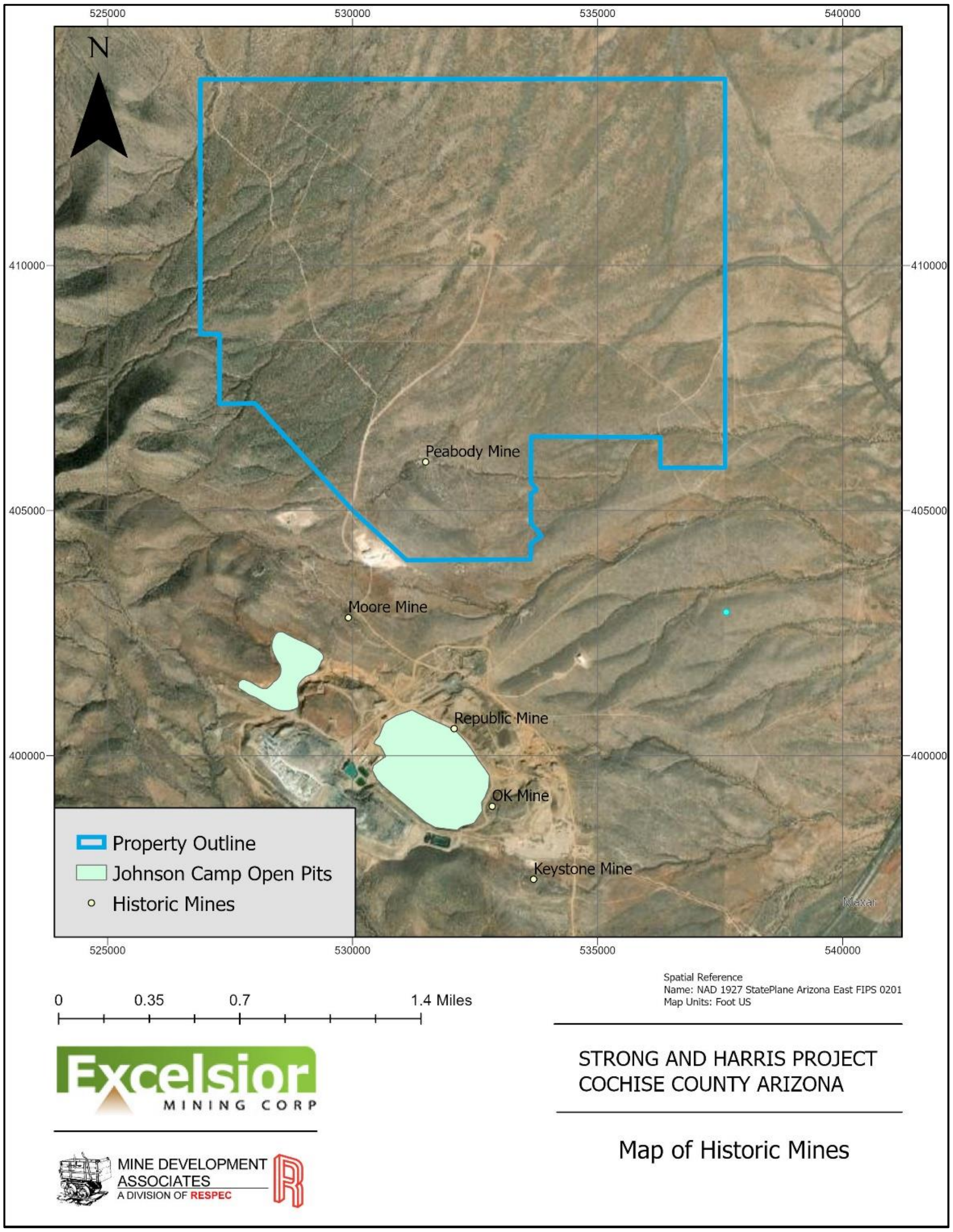
6.1 District Exploration History

The Cochise district has seen considerable copper, zinc, silver and tungsten mining beginning in the 1880s and extending to the present day. Prior to the 1880s, Mexican miners are said to have worked copper deposits cropping out south of the Strong and Harris project. Between 1882 and 1981, the district produced 12 million tons of material containing 146 million pounds of copper, 94 million pounds of zinc, 1.3 million pounds of lead, 720 thousand ounces of silver, and minor quantities of gold (Keith et. al., 1983). Much of the historical production came from small-scale underground copper-zinc mines located on what is now the Johnson Camp property controlled by Excelsior. The most significant of these producers were the Republic and Moore mines (Figure 6.1). From 1904-1940, material from these mines reportedly contained 4 to 4.5 percent copper and 0.5-0.75 ounces of silver per ton (Cooper and Silver, 1964). The zinc content for this period was not reported. After 1940, the material contained 1.5 to 3 percent copper, 5 to 10 percent zinc, and about 0.3 ounces of silver per ton. The Republic mine was the site of the historical concentrating plant in the district. Smaller underground mines in the area, such as the Peabody, reportedly yielded very high grade mineralized material which averaged 7.5 percent copper, 4 ounces of silver per ton, and contained as much as 44 percent zinc (Cooper and Silver, 1964).

Copper-oxide mineralization has been mined at the Johnson Camp open-pit operation since 1975, most recently by Nord Resources Corporation from 2008 until 2010. This property is now controlled by Excelsior. Overall, approximately 39 million tons of ore and 187 million pounds of copper have been produced out of the Johnson Camp open pits.

A major portion of the district's historical exploration work has taken place about 1.5 to 2 miles southeast of the Johnson Camp mine. In the 1960s, it was recognized that potentially economic copper-skarn mineralization could be identified remotely by magnetic highs related to the magnetite content of these mineralized bodies. As a result, a magnetic high located southeast of the now nonexistent town of Johnson was drilled in the 1960s and the North Star deposit was discovered in the valley east of the mountain flank, concealed under alluvial under cover. Since then, several companies have explored the area with extensive drilling and assaying, magnetic and induce polarization/resistivity ("IP/Res") surveys, metallurgical testing, hydrological studies and In-situ Recovery ("ISR") tests. Eventually, the North Star copper deposit became known as the Gunnison deposit.

Figure 6.1 Historical Mines of the Northern Cochise Mining District
(from Excelsior, September 2021)



6.2 Strong and Harris Project Exploration History

Modern-era exploration of the Strong and Harris project commenced in 1964. More than 100,000 feet of rotary and core drilling were done by various operators from the mid-1960s through 1992, including the Superior Oil Company ("Superior"), Cyprus Mining ("Cyprus"), Continental Exploration ("Continental"), Continental Materials, Beard Mining Company ("Beard"), AZCO and Manazanita Hills Inc. ("Manzanita"). Information on the historical drilling is summarized, to the extent known, in Section 10. The information in this section has been extracted and summarized from unpublished reports by the Ralph M. Parsons Company (Parsons, 1974), and Manzanita Hills Inc. (Manzanita, 1991).

According to Parsons (1974), oxide copper mineralization was discovered at what is now the Strong and Harris deposit in drill cuttings "while a water well was being drilled, perhaps in the early 1960s." A Mr. Strong and a Mr. Harris subsequently located mining claims on the present property.

6.2.1 1954-1957 Coronado Copper and Zinc Co.

Coronado Copper and Zinc Co. was involved in district production and exploration since 1942 and at least until 1957 (Cooper and Silver, 1964). District-wide exploration in the 1950's was focused around the underground operations in the Cochise district including the Republic Mine, the Moore Mine, and the Peabody and Black Prince Mine. A few of these drill holes were in the vicinity of the Strong and Harris deposit. They were drilled from surface to test the continuity of mineralization of the Black Prince and Peabody Mines.

6.2.2 1964 - 1972 Cyprus Mining

Cyprus optioned the property from Mr. Strong and Mr. Harris and drilled 36 holes during 1964 through 1968. Significant copper mineralization was encountered in 13 of the Cyprus drill holes. Additionally, Cyprus drilled 2 holes in 1972 in the "Peabody Sill" area of the deposit. These were likely on separate claims from the main Strong and Harris project at the time.

An induced polarization and resistivity ("IP/Res") survey was carried out for "Congdon and Carey" by McPhar Geophysics during the time of the Cyprus drilling. The survey consisted of seven east-west lines using electrodes at 400 and 600 foot spacings (Hallof and Bell, 1967).

6.2.3 1967 - 1971 Continental Exploration

Continental optioned the property from Strong and Harris in 1967 and drilled 31 core holes during 1968 through 1971. Some of the holes were used for down-hole induced potential and resistivity (IP/Res) surveys. Copper mineralization was encountered in 25 of these drill holes. Continental assigned their option to Superior via a lease agreement in 1971.

6.2.4 1971 – 1976 Superior Oil Company

Superior leased the Continental option in 1971 and drilled 63 core holes. By 1974, 51 of the 63 total holes had been drilled and at least 43 had intersected significant copper-zinc mineralization (Parsons, 1974). Superior commissioned an economic assessment of the Strong and Harris deposit by Parsons in 1974 that included an estimate of reserves for a small underground operation and documented the exploration and assay procedures used by Superior. At some point in the 1970s, Superior terminated their interest in the property.

6.2.5 1980 New Beginnings Resources

In 1980, a company known as New Beginnings Resources leased the property and carried out an IP/Res survey. New Beginnings drilled four holes with negative results (Manzanita, 1991). The authors are not aware of any further information concerning the New Beginnings exploration work and are unaware of when the New Beginnings lease of the property was terminated.

The authors have no information on the property history during most of the 1980s but believe the property ownership remained with Mr. Strong and Mr. Harris through the 1980s. In 1983, a magnetic survey was conducted by Robert L. Clayton but maps and cross sections made from the survey have been lost, and no information is available on the methods and procedures used for the survey, or the anomalies identified.

6.2.6 1985-1988 Robert Durham

Between 1985-1988 at least two exploration holes were drilled by Robert Durham who maintained ownership in the claims. Details regarding these holes are limited and largely based on public record.

6.2.7 1988 – 1989 Arizona Copper Company (“AZCO”)

In 1988 AZCO optioned the Strong and Harris project and, in 1989, entered into a joint venture with Granges Inc. to jointly explore the property. Granges drilled one hole and terminated their joint venture participation (Manzanita, 1991).

6.2.8 1991-1992 AZCO and Manzanita Hills Inc.

Manzanita entered into a joint venture with AZCO for the property in 1991. In 1992, drill hole SH-140 was drilled by AZCO. Otherwise, very little is known about any work done by Manzanita.

6.2.9 2019 – 2021 Excelsior Mining

The Strong and Harris project was idle and no work was done from 1992 into 2019. Excelsior purchased the property in 2019. Excelsior’s exploration work is summarized in Section 9.0.

6.3 Historical Mineral Resource Estimates

Historical estimates of mineralized materials for the Strong and Harris project were initially calculated by Parsons (1974) for Superior. “In-place reserves” of 53.165 million tons with average grades of 0.63% copper, 0.77% zinc and 0.22 ounces of silver per ton were stated at a cutoff grade of 0.3% total copper (Parsons, 1974). Higher grade subsets of these “reserves” were estimated in two areas. The estimates were calculated based on cross sections spaced at 200-foot intervals and then taken to level plans at 100-foot vertical intervals. These historical estimates are relevant only for historical completeness, are not considered reliable, and use categories other than those of the CIM Definitions Standards and therefore NI 43-101. Mr. Bickel is unaware of the differences with those categories and a qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. Excelsior is not treating these historical estimates as current mineral resources or mineral reserves and these estimates should not be relied on.

In 1991, Manzanita estimated “oxide reserves” of 2.213 million tons at an average of 2.78% copper, and 3.69% zinc, as well as “sulfide reserves” of 1.6 million tons averaging 2.32% copper and 3.51% zinc. Manzanita also estimated a larger, “low grade reserve” of 79 million tons with an average of 0.62% copper and 0.72% zinc. These historical estimates are relevant only for historical completeness, are not considered reliable, and use categories other than those of the CIM Definitions Standards and therefore NI 43-101. Mr. Bickel is unaware of the differences with those categories and a qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. Excelsior is not treating these historical estimates as current mineral resources or mineral reserves and these estimates should not be relied on.

6.4 Cochise District Past Production

There has been no historical production from the Strong and Harris project. Production from the surrounding Cochise mining district is summarized in Table 6.1 with data from Cooper (1964) and Zimmerman (2016) for the years 1902-2010.

Table 6.1 Historical Copper and Zinc Production, Cochise Mining District

Operation Name	Production Period	KTons of Ore	Commodity
Johnson Camp Mine	1975-2010	39,000	Copper
Moore Mine	1951-1954	250	Copper, Zinc
Republic/Mammoth Mine	1882-1952	550	Copper, Zinc
Copper Chief Mine	1905-1919	24.1	Copper, Silver
Peabody Mine	1907-1918	14.2	Copper, Silver
Black Prince Mine	1902-1918	1.4	Copper, Silver
Keystone Mine	1916-1937	1.8	Copper
Centurion Mine	1908-1944	1.5	Copper, Silver, Gold
Texas Arizona Mine	1910-1928	0.7	Copper, Lead, Silver, Gold
Total	**1902-2010**	**39,844**	

Note: data for 1902 through 2010 compiled from Cooper and Silver (1964) and Zimmerman (2016).

In addition to the operations listed in Table 6.1, several small-scale production operations with poorly preserved production records existed in the district in the late 1800s to early 1900s. This included tungsten production from vein systems in the Texas Canyon Quartz Monzonite (Cooper and Silver, 1964).

7.0 GEOLOGIC SETTING AND MINERALIZATION (ITEM 7)

The information presented in this section of the report is derived from multiple sources, as cited. Mr. Bickel has reviewed this information and believe this summary accurately represents the Strong and Harris project geology and mineralization as it is presently understood.

7.1 Regional Geologic Setting

The Strong and Harris deposit is located within the Mexican Highland region of the Basin and Range province. The region is characterized by fault-bounded mountain ranges, typically with large intrusions forming the cores of the ranges. The ranges are separated by extensional grabens containing thick sequences of Tertiary and Quaternary volcanic and alluvial deposits that overlie a basement of Precambrian through Mesozoic rocks.

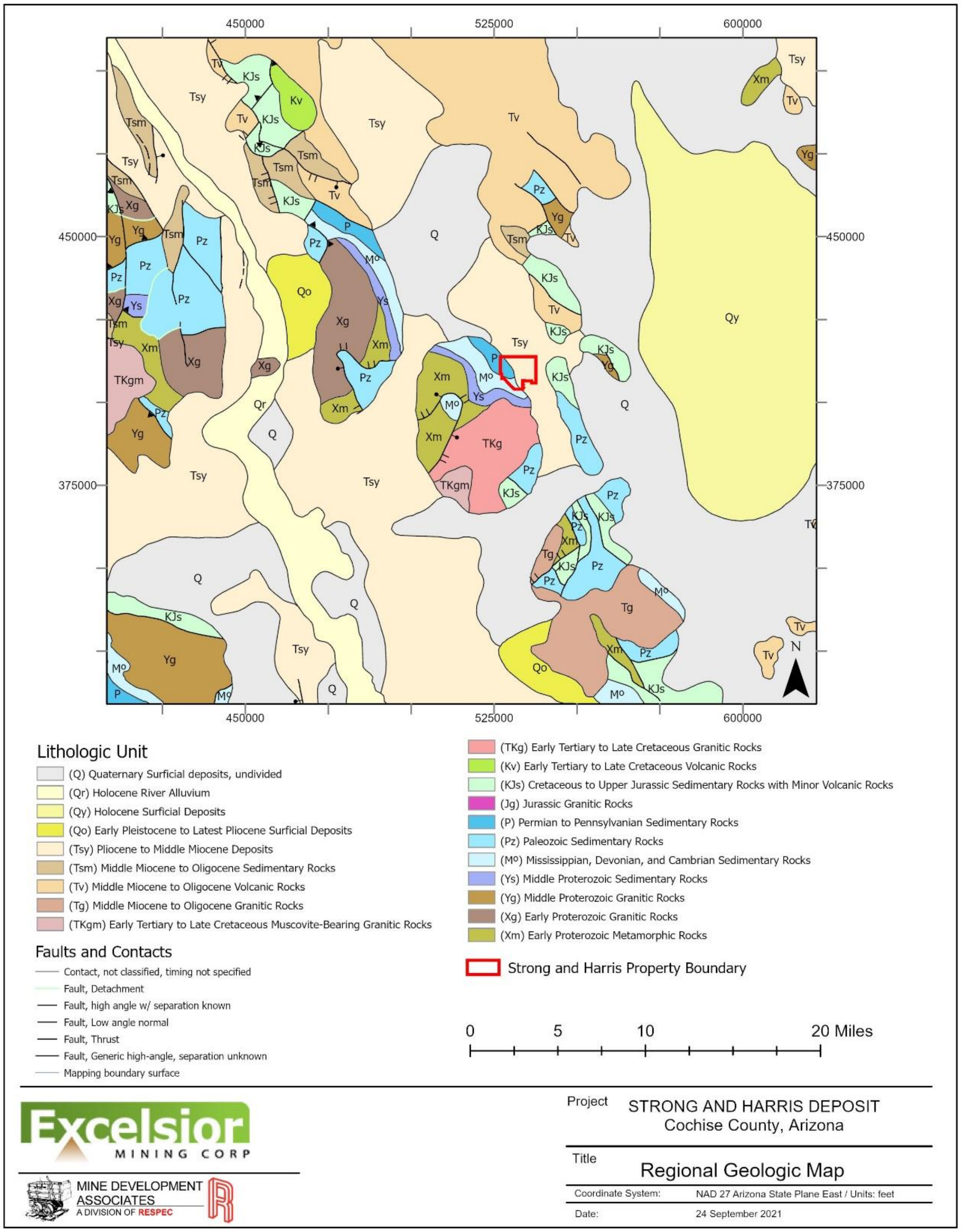
The project lies on the eastern edge of the Little Dragoon Mountains (Figure 7.1) within the Cochise mining district. The Little Dragoon Mountains are an isolated, fault bounded horst block comprised of . rocks spanning from 1.4 billion years ago (Ga) Pinal Group schists to Holocene sediments. The southern portion of the Little Dragoon Mountains consists predominately of the Texas Canyon Quartz Monzonite of Tertiary age, whereas the Pinal Group schists and a sequence of Paleozoic sedimentary units dominate the northern half of the range.

The oldest rocks in the area, the Pinal Group schists, are composed of sandstones, shales and volcanic flows that have been metamorphosed to greenschist and amphibolite facies. The Precambrian Apache Group unconformably overlies the Pinal Group schists and is composed of conglomerates, shales and quartzite that were subsequently intruded by diabase sills. The Apache Group is then unconformably overlain by Paleozoic rocks that host most of the mineralization in the district. At Johnson Camp and Gunnison, the important Paleozoic hosts include the Cambrian Abrigo and Devonian Martin Formations. At the Strong and Harris project, the relevant Paleozoic units include successively, from oldest to youngest, the Escabrosa Limestone (Carboniferous), the Horquilla Limestone (Carboniferous to Early Permian), Earp Formation (Carboniferous and Permian) and the Colina Limestone (Permian). A diabase sill has locally intruded the Paleozoic units which is thought to be correlative to Tertiary mafic intrusions that occur regionally.

The Texas Canyon Quartz Monzonite is porphyritic with large potassium feldspar phenocrysts from 1 to 10 cm in length. Livingston et. al. (1967) determined the age to be 50.3 ± 2.5 Ma (not recalculated to current decay constants). Reynolds et. al. (1986) listed eight determinations ranging from 49.5 to 55.0 Ma. The intrusion crops out to the southwest of the Strong and Harris deposit.

Several deformations have occurred in the area with the most recent being the latest Cretaceous-Paleocene Laramide Orogeny compression, followed by Miocene and younger Basin and Range extension that has modified the topography to its current appearance. Proterozoic, pre-Apache Group deformation of the Pinal Schist Group included isoclinal folding with steep to overturned fold axes with a general northeastern structural trend. Minor deformations took place in late Precambrian and post-Paleozoic but pre-Cretaceous times. The post Paleozoic-pre-Cretaceous deformation is characterized by steep northeast to easterly striking faults with displacements up to hundreds of feet.

Figure 7.1 Regional Geology Little Dragoon Mountains
(modified from Richard et al., 2000)



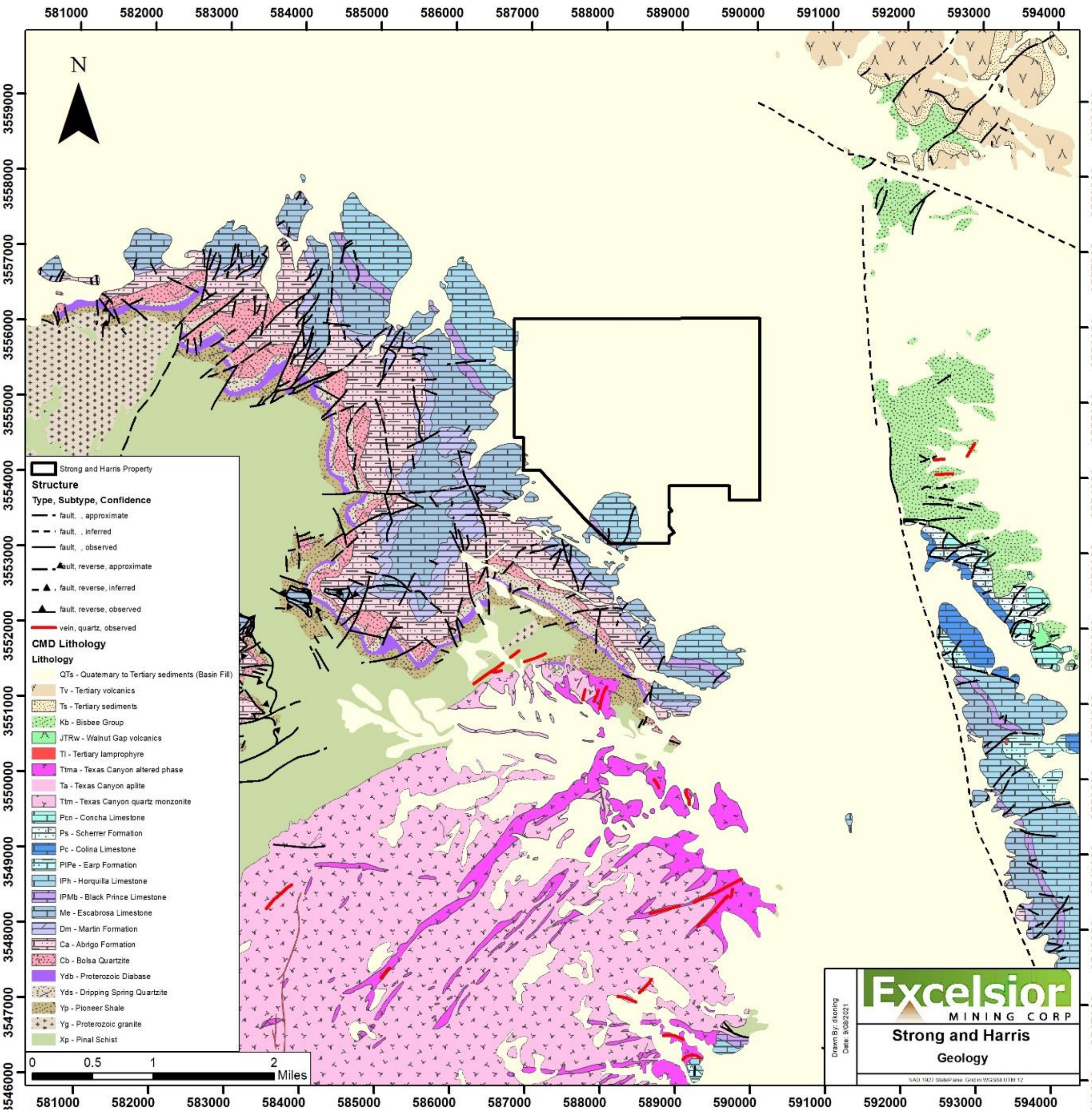
The Laramide deformation produced structures striking in a northwesterly direction and was more or less perpendicular to the Pre-Apache Group deformation. Pre-late Cretaceous faults were reactivated and modified, and folds and thrust faults are common features of the Laramide.

Two episodes of block faulting have created the Basin and Range topography that dominates the current landscape and postdates the mineralization in the region.

7.2 Property and Deposit Geology

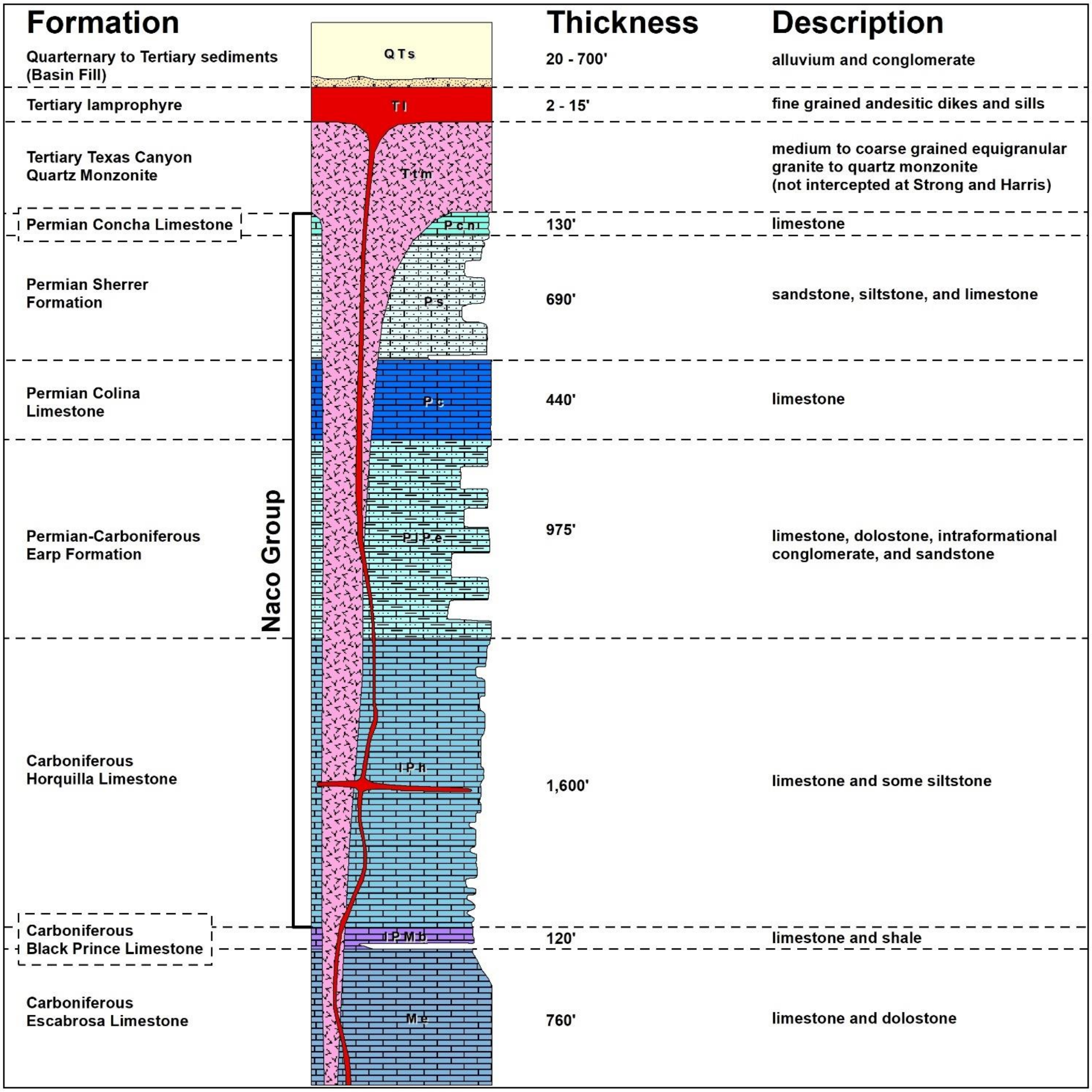
The Strong and Harris deposit is hosted in altered Paleozoic sedimentary rocks which are covered by an average of 425 feet of post-mineral and mostly unconsolidated valley fill near the northeast flank of the Little Dragoon Mountains and about three northwest of the Gunnison oxide copper deposit (Figure 7.2).

Figure 7.2 Property Geologic Setting for the Strong and Harris Project
(from Excelsior, September 2021)



A diagrammatic stratigraphic column showing the known stratigraphy of the property area is shown in Figure 7.3.

Figure 7.3 Stratigraphic Column for the Strong and Harris Project and Vicinity
(from Excelsior, September 2021)



The valley fill consists of largely unconsolidated sand, gravel, and conglomerate that dips shallowly to the east at about 15^{o} with some local variability. Near its base, the valley fill is consolidated fanglomerate for approximately 10 to 20 feet above the contact with the Paleozoic rocks. This consolidated basal layer may be correlative to the regionally extensive Gila Conglomerate. The mineralized Paleozoic rocks below the valley fill sediments strike approximately 315° azimuth and dip 30° to 45° northeast. These Paleozoic rocks include the Carboniferous Escabrosa Limestone, the Carboniferous Black Prince Limestone, the Carboniferous to Permian Horquilla Limestone, the Permian Earp Formation, and the Permian Colina Limestone. The Horquilla Limestone is intruded by a thin mafic sill in the area of Strong and Harris known locally as the "Peabody Sill" for its presence in the historic Peabody Mine south of Strong and Harris. It is likely correlative to Tertiary lamprophyre sills identified by Cooper and Silver (1964).

The stratigraphy at Strong and Harris is cut by east to northeast-striking faults with apparent right-lateral displacement and near-vertical dips. Displacements along the faults are typically between 50 to 200 feet. There appears to be a proximity relationship between these structures and the most mineralized portions of the deposit. They are likely pre- or syn-mineral in age, but may have post-mineral movement as well since some of the mineralization is apparently offset by them. Folds occur locally in the proximity of faults. However, some shallow, gentle folds are common throughout the property with fold axes approximately parallel to the northeasterly dip of the sedimentary units.

The Escabrosa Limestone is generally thick-bedded to massive, white to light grey limestone with dolomitic interbeds which are more abundant at the base of the formation. Regionally, it is 750 feet thick and forms prominent topographic ridges. In the Cochise mining district, it is often recrystallized. Skarn and calc-silicate alteration are typically limited to narrow seams along fractures or thin beds near the base of the formation. It has not been intercepted in the drill holes at Strong and Harris and does not host any known mineralization at the property. It presumably lies below the deposit.

The Black Prince Limestone is a pinkish-grey limestone with a distinct maroon shale at its base. It generally resembles the underlying Escabrosa Limestone above the basal shale and is approximately 120 feet thick. In the Cochise mining district, it is often marbelized and the shale is altered to a distinct brown hornfels. It has not been intercepted in the drill holes at Strong and Harris and does not host any known mineralization at the property. It presumably lies below the deposit.

The Horquilla Limestone consists of thick- to medium-bedded grey limestone with minor silty or shaley interbeds. The formation is typically 1,500 feet in thickness in the region, although the basal contact has not been intercepted in any of the Strong and Harris drill holes. The Horquilla is strongly marbleized in the deposit area and locally altered to various calc-silicate assemblages. Typical alteration minerals include wollastonite, diopside, tremolite, serpentine, and more rarely garnet. The Horquilla is intruded by the Peabody Sill, a fine-grained mafic igneous rock typically 10to 15 feet thick, although thinner intersections have been encountered. The sill intrudes the Horquilla consistently roughly 800 feet below the contact between the Horquilla and Earp Formation. Where mineralogy can be observed, the rock contains pyroxene and/or horneblende, biotite, and plagioclase. It is commonly altered to chlorite. Quartz-orthoclase-plagioclase(?) veins occasionally occur within 5 feet of the contact of the sill. The sill is likely correlative to Tertiary lamprophyre sills identified by Cooper and Silver (1964). Copper-zinc-silver mineralization in the Horquilla is commonly associated with or proximal to the Peabody Sill. The mineralization in the Horquilla along the diabase sill has been historically distinct from that in the Earp Formation and was historically referred to as the "Peabody Sill" mineralization.

The Earp Formation is the most significant geologic unit at Strong and Harris because it is the principal host for mineralization. The lithology is heterogeneous compared to adjacent formations, containing many interbeds (usually 2 to 8 feet in thickness) of limestone, sandstone, siltstone, and local conglomerates. The Earp Formation also exhibits relatively low competency, likely owing to the interbedded nature of the sequence. It is roughly 975 feet thick at Strong and Harris. The Earp Formation is commonly altered to various assemblages of calc-silicates, historically described as tactites, typically containing wollastonite, pyroxene, serpentine, and amphiboles. Rarely, green garnet is observed in the tactites. Limestone beds have been intensely marbleized and locally silicified. Silicification is commonly more abundant toward the base of the formation.

The Colina Limestone is a medium- to thick-bedded, dark grey to black limestone which overlies the Earp Formation. It has rare thin-bedded sandstone units near the base. It is at least 440 feet thick in the Cochise mining district (Cooper and Silver, 1964). It is only a minor host to mineralization at Strong and Harris.

7.3 Mineralization

Primary copper-zinc-silver mineralization at Strong and Harris is characterized by lenses of sulfide minerals emplaced more-or-less parallel to layering in favorable lithologic units, usually along bedding planes or in disseminated masses and blebs. Some mineralization is disseminated in certain lithologies. Less frequently, the mineralization is hosted in quartz +/- calcite +/- feldspar veins. The mineralization is typically accompanied by calc-silicate alteration of the carbonate host-rock (described as "tactite" in the logs). In some local areas or sub-units, the mineralization completely replaced the host rock with massive lenses or patches of sulfide minerals, some of which are now oxidized. The sulfide minerals include pyrite, pyrrhotite, chalcopyrite, chalcocite, and sphalerite. Minor tetrahedrite group minerals have also been reported in the historical drill logs.

Sub-units of the Earp Formation, particularly those immediately below its upper contact with the Colina Limestone, were the most favorable sites for deposition of the copper, zinc and silver minerals. However, mineralization is also present in the Colina Limestone above the Earp, as well as in the Horquilla Limestone below the Earp. Historical reports often referred to mineralization in the Horquilla as the "Peabody Sill", as such mineralization and its host rock were termed at the historical Peabody Mine southwest of the Strong and Harris deposit. The contact between the Horquilla and this sill at the Peabody Mine was reportedly favorable, at the mine although the sill itself is thin and represents only a volumetrically minor portion of that deposit. The same relationship is observed on the western side of the Strong and Harris property where the diabase sill has been logged in several holes and is often mineralized. The thickness of the sill is typically less than 10 feet. Mineralization in tactites of the Horquilla Limestone, either stratigraphically above or below the sill, is equally if not more important than the sill itself at Strong and Harris. However, the sill is a favorable host where present.

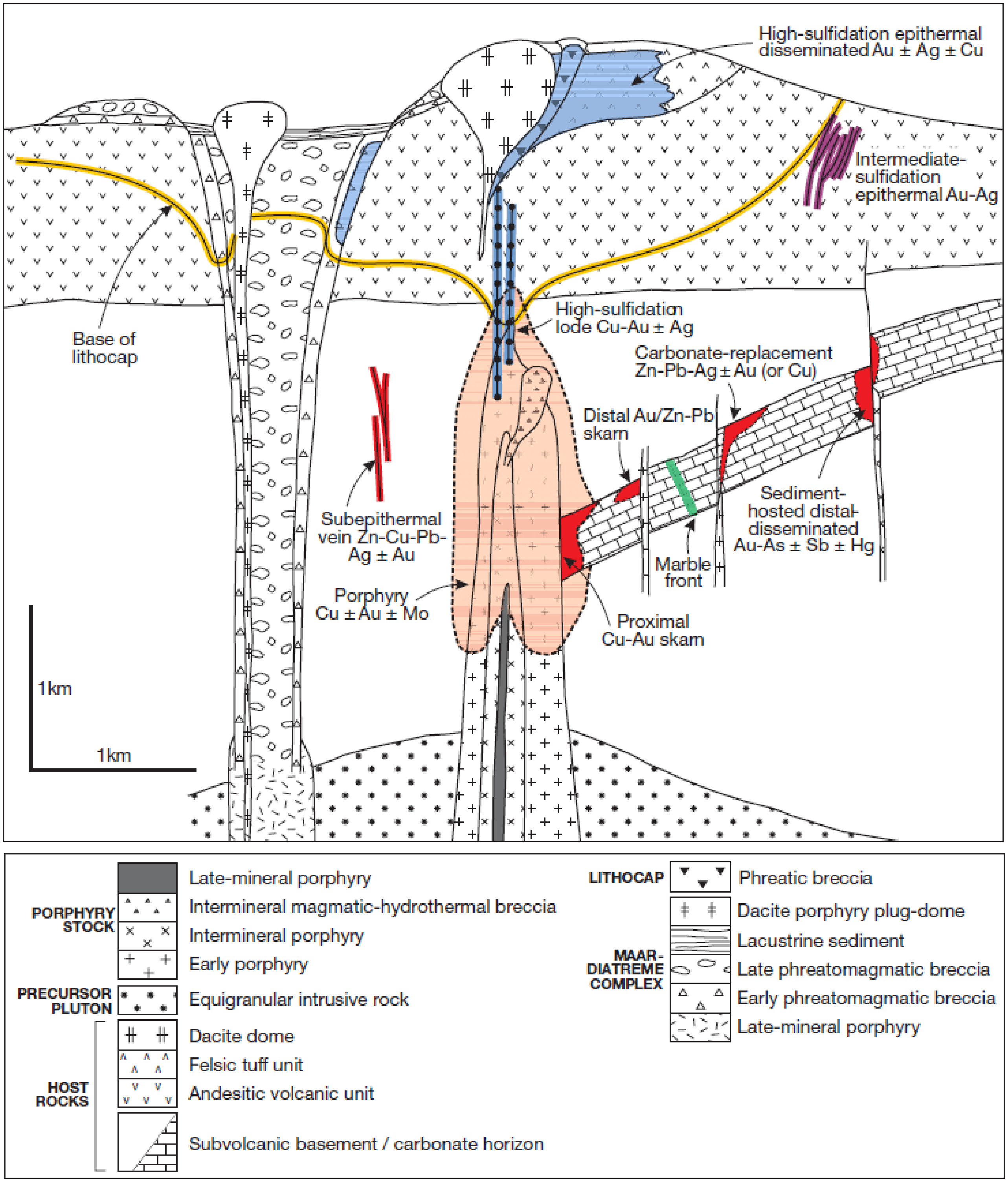
The Strong and Harris deposit has been oxidized to varying degrees that generally decrease with depth. Three oxidation zones are currently recognized in the deposit: the oxide zone, the transition (or mixed) zone, and the sulfide zone. In the oxide zone, copper is dominantly hosted in chrysocolla with minor azurite, malachite, and tenorite. Zinc minerals noted in the oxide zone include rosasite, aurichalcite, and willemite. Sulfide zone mineralogy is dominated by chalcopyrite and sphalerite with associated pyrite and pyrrhotite. In the transition (mixed zone), the mineralogy consists of secondary sulfides (namely chalcocite) mixed with a combination of the above oxide and sulfide zone mineralogy.

8.0 DEPOSIT TYPES (ITEM 8)

The Strong and Harris copper-zinc-silver deposit is a sub-type of or related to a classic copper skarn (Einaudi and Burt, 1982; and Meinert et al, 2005). Skarn deposits range in size from a few million to 500 million tonnes and are globally significant, particularly in the southwestern US. They can be stand-alone copper skarns, which are generally small, or can be spatially and temporally closely associated with porphyry copper deposits, in which case they tend to be very large. The skarn at Strong and Harris and collectively in the Cochise mining district is presumably related to the Texas Canyon Quartz Monzonite, despite the intrusive itself hosting very little known economic mineralization. Mineralization in the quartz monzonite would require more specialized conditions involving the metal and volatile content of the magma, depth of emplacement, or other factors (Burt, 1977).

Copper skarns generally form in calcareous shales, dolomites and limestones peripheral or adjacent to the margins of diorite to granite intrusions that range from dikes and sills, to large stocks or phases of batholithic intrusions, and frequently are associated with mineralized intrusions. Copper mineralizing hydrothermal fluids are focused along structurally complex and fractured rocks and convert the calcareous shales and limestones to andradite-rich garnet assemblages near the intrusive body, and to pyroxene and wollastonite rich assemblages at areas more distal to the intrusive. Retrograde evolution of the hydrothermal fluids produces actinolite-tremolite-talc-quartz-epidote-chlorite assemblages that overprint earlier garnet and pyroxene. Strong and Harris occurs approximately two miles north of any known occurrences of the Texas Canyon Quartz Monzonite intrusion in the Cochise mining district, which is thought to be the source of mineralizing hydrothermal fluids. Therefore, Strong and Harris can be sub-categorized as distal skarn related to a porphyry copper system. This assumption is supported by the high abundance of wollastonite alteration in the mineralized zones. The anatomy of a telescoped porphyry copper system model (Figure 8.1) by Sillitoe (2010) can be used as a conceptual model to understand the spatial relationship of the Strong and Harris distal skarn and associated proximal skarns in the district.

Figure 8.1 Schematic Model
(after Sillitoe, 2010)



9.0 EXPLORATION (ITEM 9)

This section summarizes the exploration work carried out by Excelsior. Mr. Bickel has reviewed the information provided by Excelsior and believes it is an accurate representation of the work done by Excelsior.

Excelsior has not conducted drilling at the Strong and Harris project. Drilling by previous operators is summarized in Section 10.

9.1 Historical Data Compilation

Excelsior inherited a data package upon purchase of the project. It consisted of well-organized boxes of paper records, drill logs, assay certificates, and technical reports from Robert Durham, who previously controlled the property.

In 2019, Excelsior began a comprehensive technical review of the reports and project drill data. In 2020-2021, Excelsior completed a data compilation program to digitize and validate the Strong and Harris data. Excelsior contracted MDA to assist with some of this program.

As of the Effective Date of this report, Excelsior's compilation of historical data efforts include:

- Scanning of all historical reports, drill logs, assays, and miscellaneous technical information from the paper files. Excelsior gathered all paper records for the Strong and Harris project and scanned them.

- Conversion of drill hole collar coordinates from historical grid to UTM NAD 27, State Plane Arizona East coordinate system. Drill hole collar coordinates were provided in the historical data records, along with maps of the collar locations. The grid used in the historical data was not recognized. Excelsior contracted MDA to use these data, along with data derived from handheld GPS measurements of the existing collars, to create a two-point transformation of the historical collar coordinates to UTM NDAD 27, State Plane Arizona East coordinate system to match Excelsior's existing data formats. These collar locations were further verified and adjusted based on field suveys performed in 2021 (as described in Sections 10.9 and 12.2.2).

- Construction of digital drill hole database. Excelsior contracted MDA to construct a digital drill hole database in 2020 based on the historical paper records and scans thereof. This included a comprehensive compilation of all assays, lithologies, collar, survey, and other relevant data into digital format. During this process, MDA verified the data compiled into the database described in section 12.2.

- Digitization of geologic surfaces. At Excelsior's request, MDA created preliminary 3D geologic surfaces of the geologic units relevant to the Strong and Harris deposit. These surfaces were based on cross-sections and maps contained within the historical paper records. In 2021, the geologic surfaces were further refined by collaborative edits between Excelsior staff geologists and MDA.

- Inventory of historical drill core. Excelsior contracted technical staff, including MDA, to move the historical Strong and Harris drill core from its location in Dragoon, AZ, to Excelsior's core processing facility in Casa Grande, AZ. As a part of this process and the ensuing re-sampling of the core, a detailed inventory of the remaining available drill core was generated. The inventory recorded 125 unique historical drill holes, all corresponding to those in the data records and database. In some cases, core for certain sections of the holes were missing boxes and/or intervals. In total, approximately 35,000 feet of core remains intact.

9.2 Geologic Mapping

In 2020 and 2021, Excelsior conducted geologic mapping over selected areas within the Cochise mining district west and south of the Strong and Harris project. Traverse mapping at a 1:10,000 scale focused on alteration assemblages, veins orientation, and confirmation of published USGS geologic maps. The mapping was conducted to identify alteration assemblage's indicative of potential deposits and to characterize known mineralization. Mapping in the Johnson Camp area extended north to the Peabody mine and the exposed lithologies that could be relevant to the areas of the property covered by Cenozoic basin-fill units.

9.3 Excelsior Re-Sampling of Historical Drill Core

Excelsior carried out a re-sampling program in February and March of 2021 based on MDA's recommendations. The program was executed by a collaboration between Excelsior staff and contractors, and MDA. The purpose of the program was to 1) verify historical data, and 2) increase the amount of silver assays in the database for the purposes of estimating resources. In total, 1089 samples were selected for re-assay (not including standards, blanks, and duplicates). The criteria for samples to be re-assayed generally included spatial and geologic distribution, as well as core availability. 20% of the samples were intended for verification specifically. Spot-checking of lithology logging and mineralization were included as a part of the program. The processes employed in the re-sampling program are described in Section 11, and the results are discussed in Section 12.

10.0 DRILLING (ITEM 10)

All of the drilling summarized in this section was conducted by historical operators from the 1960s through 1992. Excelsior has not conducted drilling at the Strong and Harris project. This section summarizes the historical drilling and the information presented in this section of the report is derived from multiple sources, as cited. The author has reviewed this information and believe this summary accurately represents drilling done at the Strong and Harris project.

10.1 Summary

The authors are aware of records for a total of 152 holes drilled within the Strong and Harris project, for a total of approximately 130,679 feet drilled. The author believes these holes were drilled in 1965 through 1992 as summarized in Table 10.1. Of these, at least 125 holes were drilled with rotary methods from surface through the valley fill sediments to an average depth of about 425 feet where the top of the Paleozoic sequence was encountered. From that contact the holes were drilled to their final depths with diamond-core ("core") methods. The drill hole locations are shown in Figure 10.1. Cross sections with representative drill results are provided in Section 14 of this report.

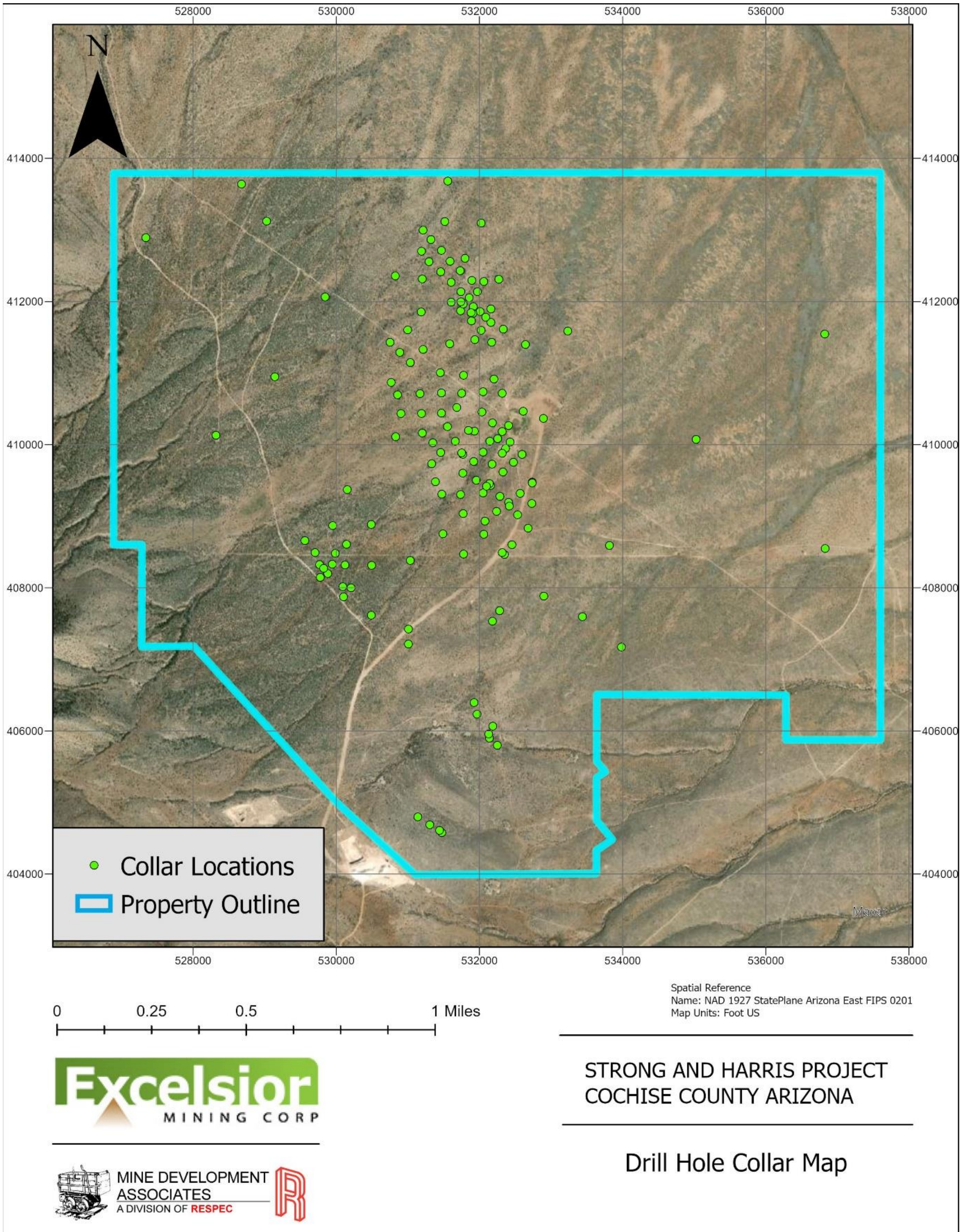
Records of the historical drilling are fragmentary and incomplete. Much of the original information on the methods and procedures used for the historical drilling has been lost. This section is partly based on the summary information provided by Parsons (1974) for the Superior drilling.

Table 10.1 Summary of Strong and Harris Historical Drilling

Operator	Year	Holes	Feet
Coronado Copper and Zinc Co.	1954 – 1957	10	7,173
Cyprus Mining	1965 - 1972	38	32,952
Continental	1968 - 1970	31	22,597
Superior Oil	1971 - 1976	70	64,304
Beard Mining	1980	deepened SH-83	1,501
Robert C. Durham	1985-1986	2	1,094
AZCO/Granges	1992	1	1,058
Totals		***152***	***130,679***

Three of the ten holes drilled by Coronado Copper and Zinc Co. were angled at -40˚ and the rest were vertical. All 10 were core holes drilled with BX and smaller diameters but no information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole surveys, if any were conducted.

Figure 10.1 Map of Strong and Harris Drill Holes



10.2 1954 – 1957 Historical Drilling by Coronado Copper and Zinc Co.

Coronado Copper and Zinc Co. drilled a total of 7,173 feet in 1954 and 1957. According to the records, most of the core size was EX with the exception of BX and AX pre-collars. No other information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole surveys, if any were conducted.

10.3 1965 - 1968 Historical Drilling by Cyprus Mining

Cyprus drilled a total of 32,952 feet in 38 vertical holes in 1965 to 1968. According to Parsons (1974), the core size was NX with the exception of a few feet of BX size. No information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole surveys, if any were conducted.

10.4 1968 – 1970 or 1971 Historical Drilling by Continental Exploration

Continental drilled a total of 22,597 feet in 31 vertical holes in 1968 through 1970, and possibly into 1971. The core size was NX with the exception of a few feet of BX size (Parsons (1974). No information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole deviation surveys, if any were conducted.

10.5 1971 – 1976 Historical Drilling by Superior Oil Company

The author has records indicating that Superior drilled a total of 64,304 feet in 70 holes during 1971 to possibly as late as 1975. All of the holes were vertical. The core size was NX with the exception of a few feet of BX size (Parsons (1974). No information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole deviation surveys, if any were conducted.

10.6 Historical Drilling by New Beginnings Resources

New Beginnings drilled four holes at the Strong and Harris project according to a 1991 report by Manzanita Mining. Records of this drilling have been lost and the author is unaware of the locations of these drill holes, the methods and procedures used for the drilling, and the results of this drilling.

10.7 Historical Drilling by Robert C. Durham

In 1985 and 1986, Robert C. Durham drilled two holes at Strong and Harris. The drill contractor was Longyear Company. No information is available regarding the rig type(s) or methods and procedures for collar and down-hole deviation surveys, if any were conducted.

10.8 Historical Drilling by AZCO/Granges Inc.

In 1992, AZCO Mining (through a Joint Venture with Granges, Inc), drilled one hole at Strong and Harris. The drill contractor was Longyear Company. No information is available regarding the rig type(s) or methods and procedures for collar and down-hole deviation surveys, if any were conducted.

10.9 Drill-Hole Collar Surveys

Excelsior has located 97 historical drill hole collars through a survey from Darling Environmental & Surveying, Ltd. of Tucson, Arizona. The survey was conducted using a Trimble Global Positioning System ("GPS"), which can be accurate to 0.05ft horizontally and 0.2ft vertically.

10.10 Down-Hole Surveys

Only two of the holes at the Strong and Harris project are known to have been surveyed for down-hole deviation. Both holes were drilled by Superior. The surveys were conducted in 1974 by Parsons Survey Co. of Tucson, Arizona (Parsons, 1974). The author is not aware of the methods, procedures or type of instruments used for these surveys.

10.11 Summary Statement

Mr. Bickel believes that the drilling sampling procedures provided samples that are representative and of sufficient quality for use in the resource estimations discussed in Section 14.0. The author is unaware of any sampling or recovery factors that materially impact the mineral resources discussed in Section 14.0.

There is a general lack of down-hole deviation survey data for the historical holes in the Strong and Harris database in all but two drill holes. While the paucity of such data is not unusual for drilling done prior to the 1990s, the lack of deviation data contributes a level of uncertainty as to the exact locations of drill samples at depth. However, in the Strong and Harris area these uncertainties are mitigated to a significant extent by the vertical orientation of nearly all drill holes, the fact that the two surveys that do exist show very little deviation, and the likely open-pit nature of any potential future mining operation that is based in part on data derived from the historical holes.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY (ITEM 11)

This section summarizes all information known to Mr. Bickel relating to sample preparation, analysis, and security, as well as quality assurance/quality control procedures and results, that pertain to the Strong and Harris project. The information has either been compiled by Mr. Bickel from historical records as cited, or provided by Excelsior.

11.1 Historical Sample Preparation and Analysis

Mr. Bickel is unaware of any information on the methods and procedures used by Cyprus and Continental for the preparation of their drilling samples. Samples from the Cyprus and Continental drilling were originally analysed at Southwestern Assayers and Chemists (Parsons, 1974). Incomplete records indicate that some samples were analysed for various combinations of copper ± gold, ± silver ± lead ± zinc ± molybdenum, but the analytical methods are not known.

According to Parsons (1974), for both old core and new core acquired by Superior, the core was split using a guillotine-type splitter. Half was stored and half was placed in cloth sample bags and sent to American Analytical Research Laboratories ("AARL") in Tucson, Arizona. Each sample was reportly crushed to minus ¼ inch and split to yield about a two pound fraction. The two pound split was pulverized and dried, then composites of the individual samples were prepared to make 100ft intervals that were analysed at AARL for total copper, total zinc, and oxide copper. In some cases, gold and silver were determined. Copies of assay certificates are preserved in the historical records and the author has reviewed and audited the certificates against the database. The author has no information on the analytical methods and procedures used, or the certifications that AARL may have held. The author infers that AARL was independent of Superior.

Samples from the AZCO/Granges hole of 1992 were analyzed by Skyline Labs for copper, lead and zinc. Copies of assay certificates are preserved in the historical records and the author has reviewed and audited them against the database. No information is available on the methods and procedures used for sample preparation and analysis.

11.2 Excelsior 2021 Sample Preparation and Analysis

Drill core remaining from historical drilling was inspected and selected intervals were re-sampled in 2021 under the supervision of Mr. Bickel. Before sampling, the core boxes were inventoried, photographed, and inspected by Mr. Bickel and Excelsior staff. Samples were selected based on criteria agreed upon by Excelsior and Mr. Bickel, and core availability. A vast majority of the samples existed as half core (originally split by historical operators). These samples were split to ¼ core. In some rare cases, the samples were taken on full core that had not been sampled previously. These samples were split to ½ core. All samples were mechanically split and placed in bags. Internal QA/QC samples (standards, blanks, and ¼ core duplicates) were inserted approximately every tenth sample in the sequence.

The Excelsior samples were prepared and analyzed at Skyline Laboratories ("Skyline") in Tucson, Arizona. Skyline is an independent commercial laboratory that holds ISO 9001:2015 and ISO/IEC 17025:2017 accreditations.

The samples were crushed to plus 75% passing -10 mesh, then split and pulverized with standard steel to plus 95% passing -150 mesh.

The analytical methods for the assays are as follows:

Total Cu (Cu) and Zinc (Zn) analyses: Samples are digested in a mixture of hydrochloric, nitric and perchloric acids. This solution is heated and taken to dryness. The contents are treated with concentrated hydrochloric acid and the solution is brought to a final volume of 200 mL with de-ionized water. This solution is read by Atomic Absorption using Standard Reference Materials made up in 5% hydrochloric acid.

Sequential Analysis of Acid-Soluble Cu (ASCu) and Cyanide-Soluble Cu (CNCu) analyses: Samples are digested in 5% sulfuric acid and supernatant solution is diluted to 100 mL with de-ionized water. The residue is digested in 10% sodium-cyanide solution and diluted to 100 mL. The ASCu samples are read on Atomic Absorption units using 0.5% H2SO4 calibration standards. The CNCu samples are read on Atomic Absorption units using 1% NaCN calibration standards.

Silver Fire Assay analyses: Silver was determined by fire-assay fusion of a 50g aliquot of the pulp, followed by a gravimetric finish.

31 element analyses: A total of 31 major, minor and trace elements, including silver, were determined by inductively-coupled plasma optical-emission spectrometry ("ICP-OES") after aqua-regia digestion.

220 of the samples were analyzed by Skyline for bulk density using the water-displacement method.

11.3 Sample Security

The authors have no information on the sample security methods and procedures used by historical operators. Drill core remaining from the historical drill campaigns has been stored at the Excelsior core facility in Casa Grande, AZ Excelsior's samples were selected and stored in plastic bags at the Excelsior core facility. The plastic bags were placed into large mobile bins and made available for direct pickup by Skyline labs. Upon pickup by Skyline, Chain of Custody sheets were filled out and signed by Excelsior and Skyline.

11.4 Quality Assurance/Quality Control

11.4.1 Historical QA/QC Results

Little information is provided in the historical records pertaining to the results of historical QA/QC programs. According to Parsons (1974), a spot check of assay values generated by American Analytical vs. those from Southwestern Assayers and Chemists was performed on samples from Continental's drill holes. Parsons (1974) indicates that the reproducibility results showed a slight low bias in American Analytical's results compared to the results of Southwestern, but the differences were "well within" 10% of the original value.

11.4.2 Excelsior QA/QC Methods and Results

CRMs. In 2021, Excelsior purchased commercial certified reference materials ("CRMs") for use in the 2021 re-sampling program. The CRMs were inserted into the re-sample stream and analyzed with the core samples. The results were used to evaluate the analytical accuracy and precision of the analyses in Excelsior's samples.

In the case of normally distributed data, 95% of the CRM analyses are expected to lie within the two standard-deviation limits of the certified value, while only 0.3% of the analyses are expected to lie outside of the three standard-deviation limits. Note, however, that most assay datasets from metal deposits are positively skewed. Samples outside of the three standard-deviation limits are typically considered to be failures. As it is statistically unlikely that two consecutive analyses of CRMs would lie between the two and three standard-deviation limits, such samples are also considered to be failures unless further investigations suggest otherwise. All potential failures should trigger investigation, possible laboratory notification of potential problems, and possible reanalysis of all samples included with the failed standard result.

Table 11.1 lists the CRMs used by Excelsior. Note that the CRM "CRM Oxide Au" is not a gold standard, but is certified for silver.

Table 11.1 Certified Reference Materials for 2021 Assays

Reference Material	Certified Value (%Cu)	2 Std Dev (%Cu)	Certified Value (Zn ppm)	2 Std Dev (Zn ppm)	Certified Value (Ag ppm)	2 Std Dev (Ag ppm)	No of Skyline Analyses
AMIS 0200	1.06	0.09					6
A106013X	0.57						2
CRM Oxide Au					47.6	4.8	20
MEG-GB4			708	78			7

The Skyline copper analyses of the Excelsior CRMs returned excellent results, with generally good precision and accuracy and no 'failures' for both AMIS 0200, shown in Figure 11.1, and A106013X, shown in Figure 11.2.

Figure 11.1 AMIS 0200 Copper Analyses

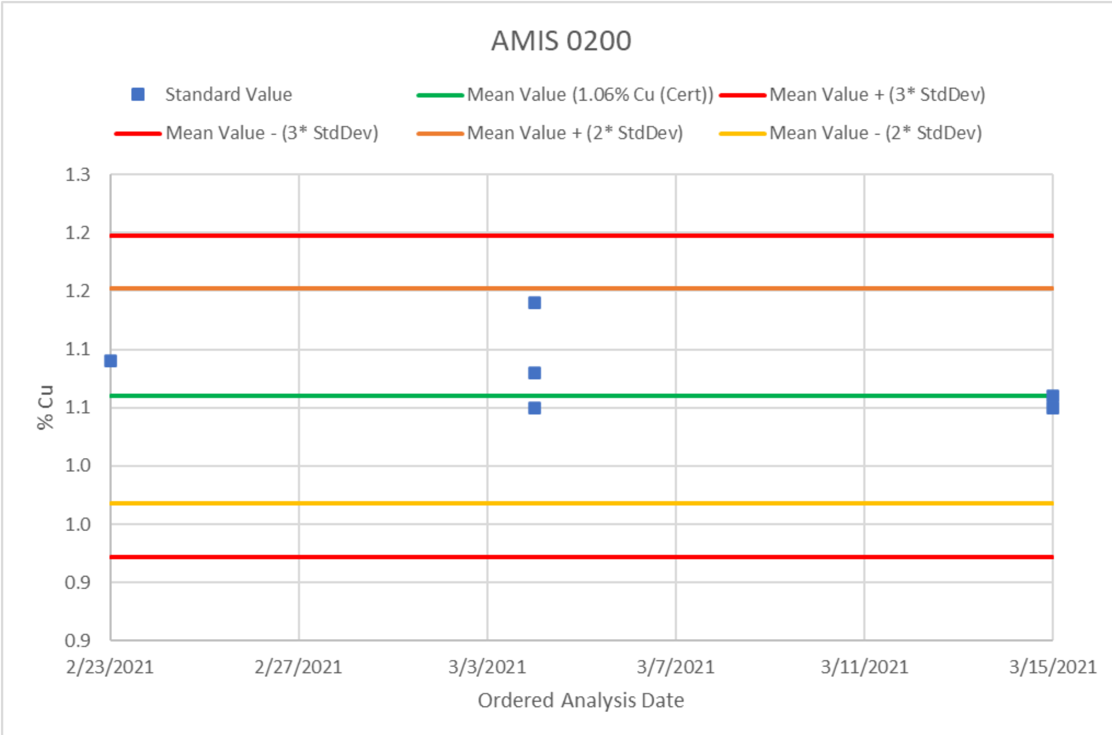
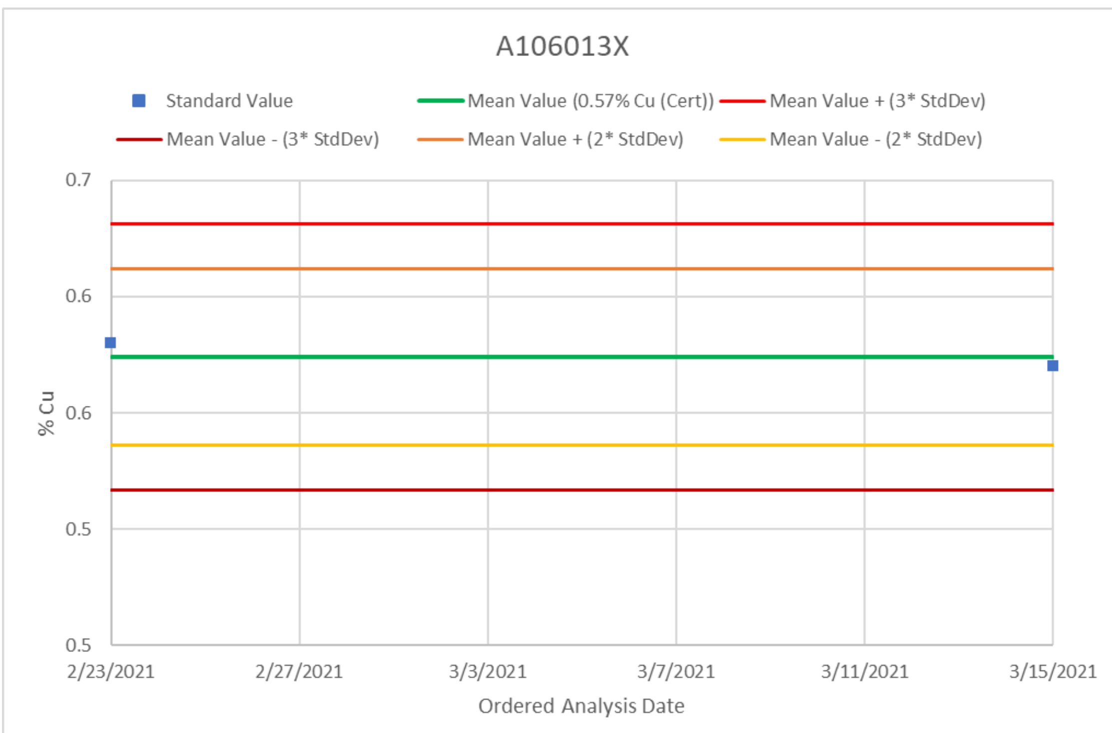
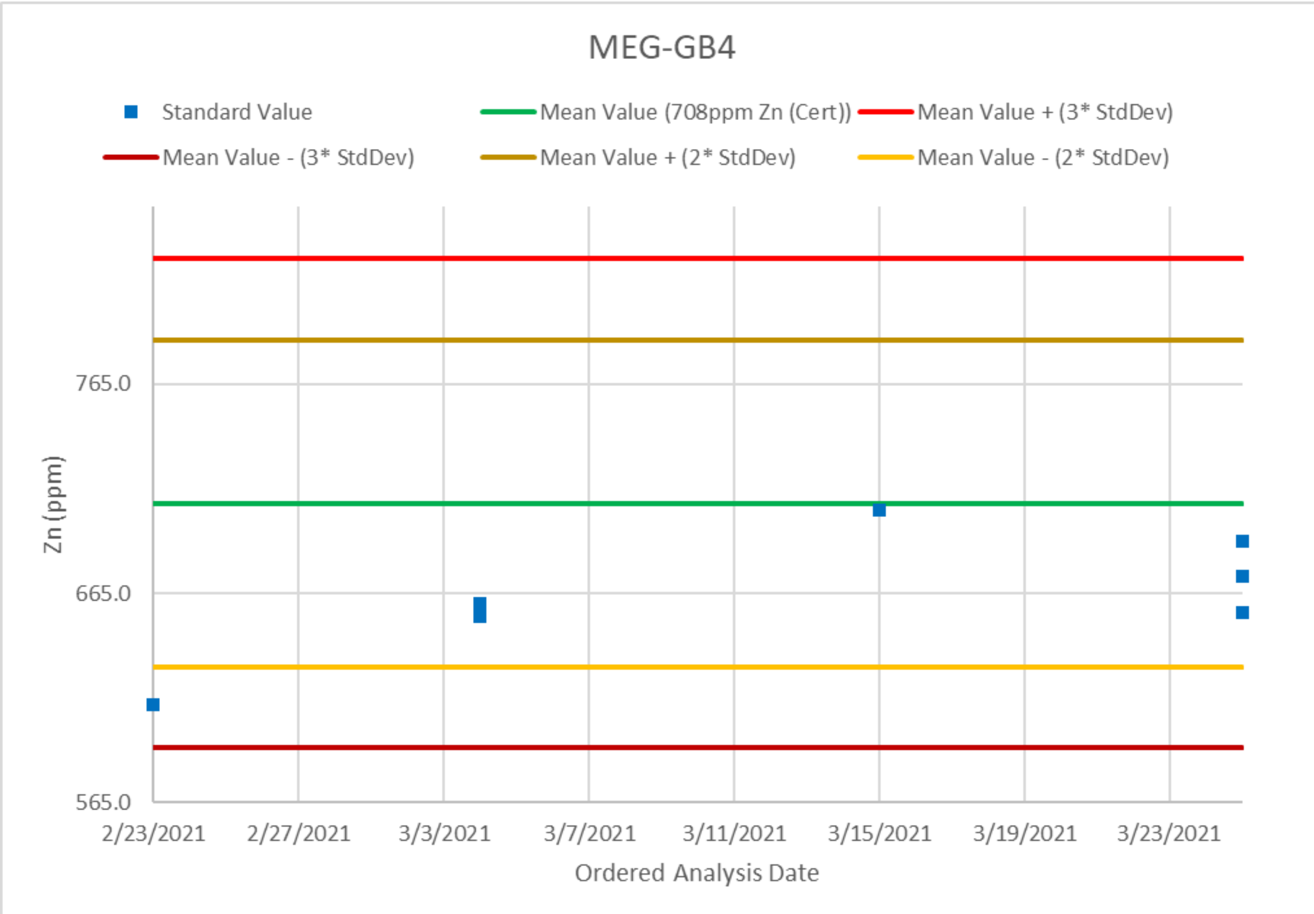


Figure 11.2 A106013X Copper Analyses



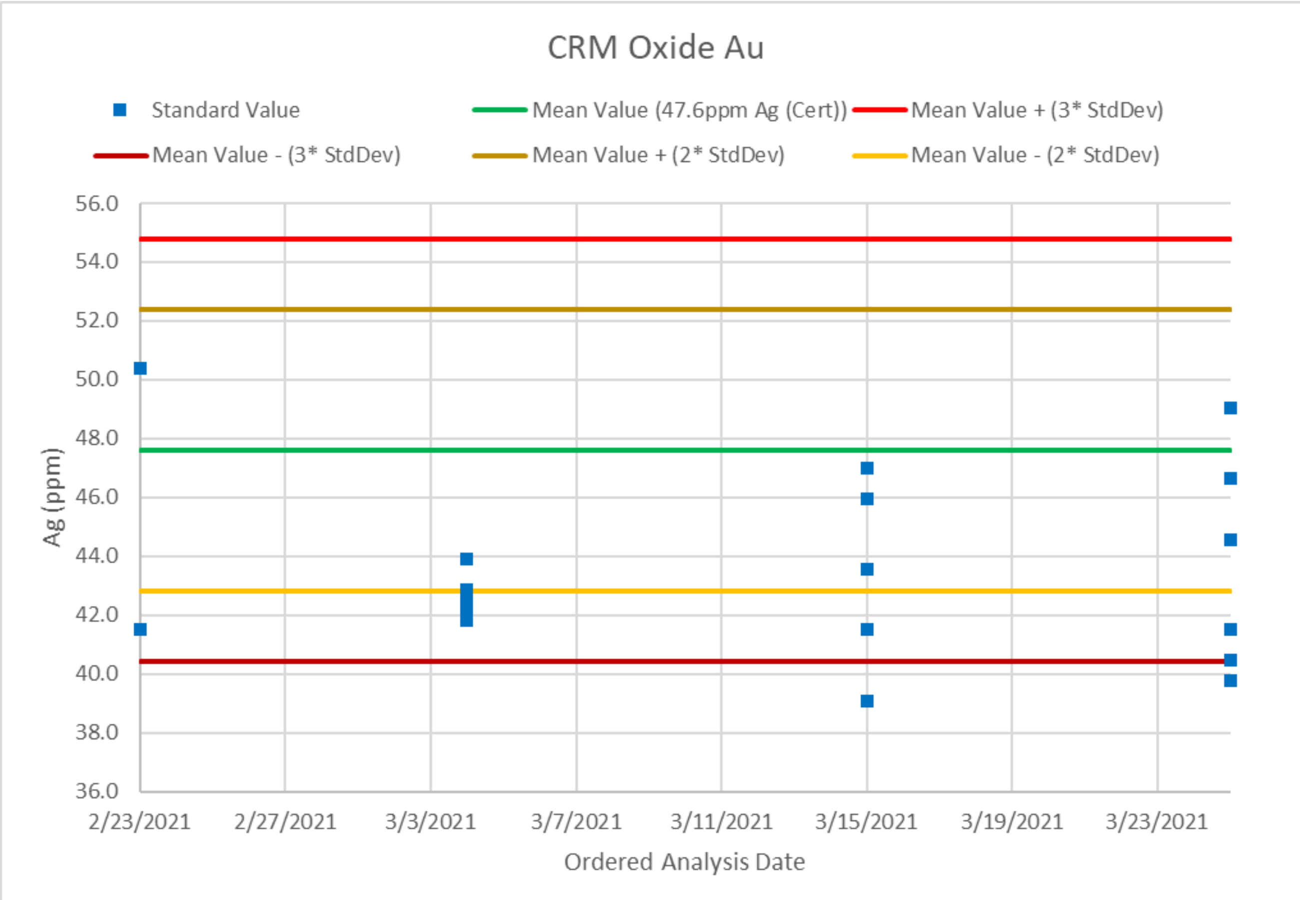
Skyline zinc analyses of the Excelsior CRMs met normal performance thresholds with few ‘failures’. The results are shown in Figure 11.3. The zinc analyses clearly showed a low bias.

Figure 11.3 MEG-GB4 Zinc Analyses



Skyline silver analyses of the Excelsior CRMs met normal performance thresholds with a moderate amount ‘failures’. The results are shown in Figure 11.3. The silver analyses clearly showed a low bias.

Figure 11.4 CRM Oxide Au Silver Analyses



Coarse Blanks. Coarse blanks are samples of barren material that are used to detect possible contamination in the laboratory, which is most common during sample preparation stages. In order for analyses of blanks to be meaningful, they must be sufficiently coarse to require the same crushing and pulverizing stages as the drill samples. It is also important for a significant number of the blanks to be placed in the sample stream within, or immediately following, a set of mineralized samples, which would be the source of most contamination issues. In practice, this is much easier to accomplish with core samples than RC.

Blank results that are greater than five times the lower detection limit of the relevant analyses are typically considered failures that require further investigation and possible re-assaying of associated drill samples. The detection limit of the Skyline analyses was 0.01 % for copper and zinc, and 0.1 oz/ton for silver, so blank samples assaying in excess of these detection limits are considered to be failures. Plots of the Skyline analyses of the coarse blanks (y-axis) versus the values of the previous samples, which would be the likely source of any in-lab contamination, are shown in Figure 11.5, Figure 11.6, and Figure 11.7. There were no coarse blank failures among the samples analyzed.

Figure 11.5 Coarse Blank Silver Values vs. Silver Values of Previous Samples

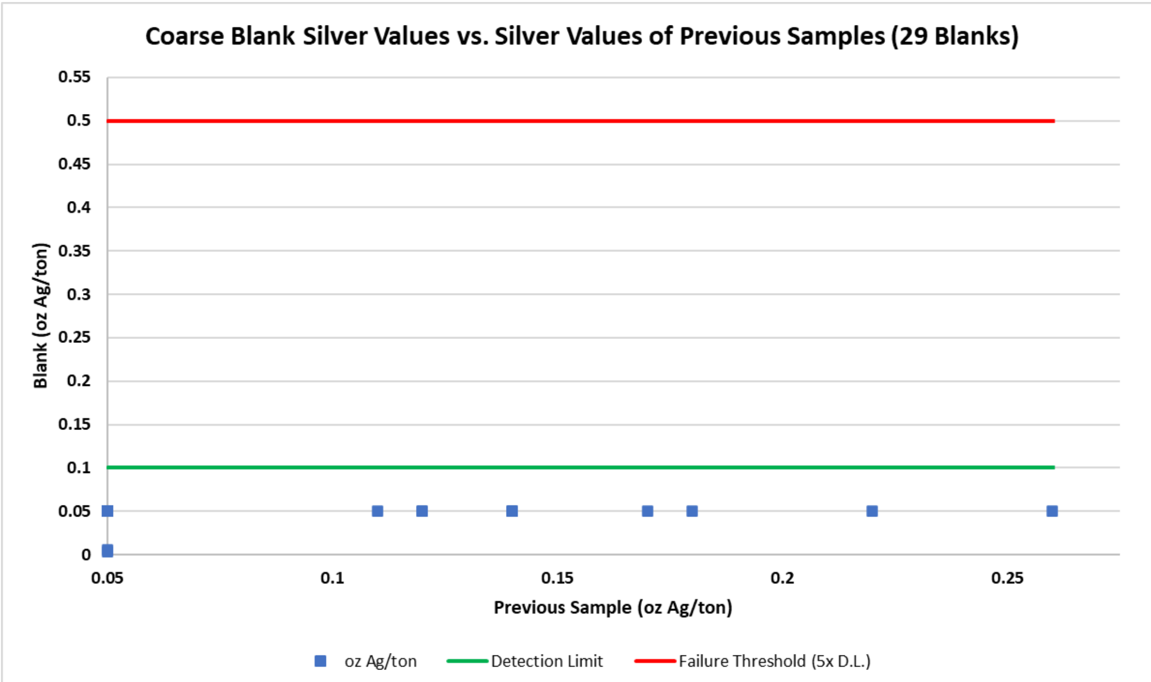


Figure 11.6 Coarse Blank Copper Values vs. Copper Values of Previous Samples

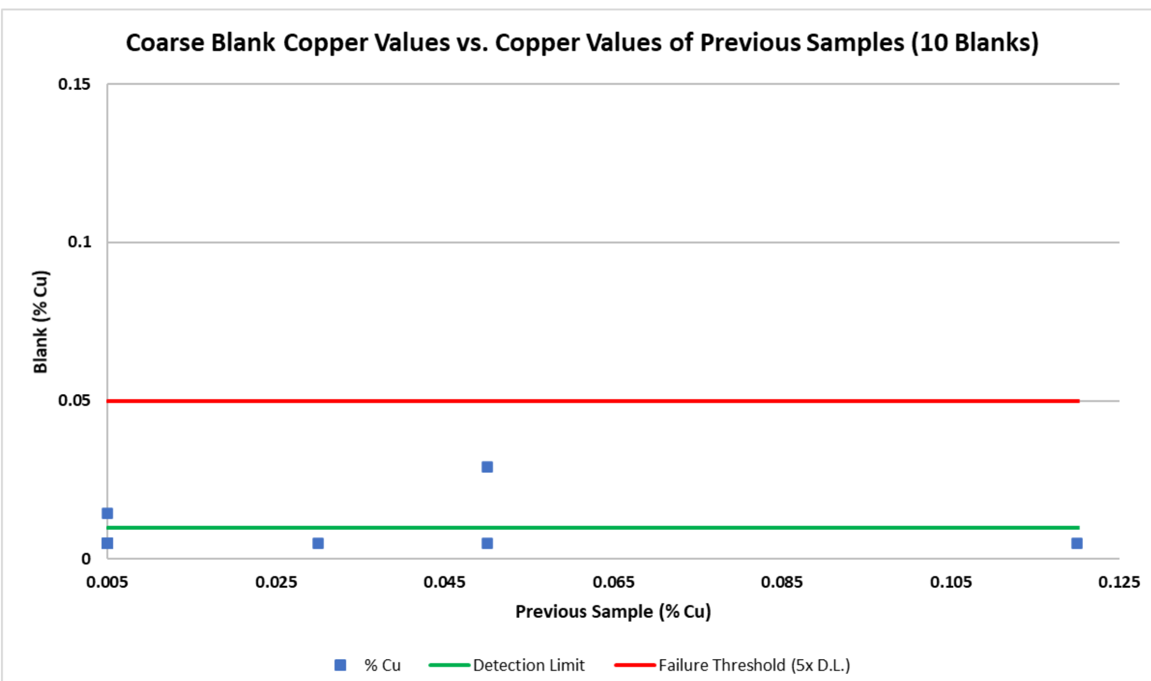
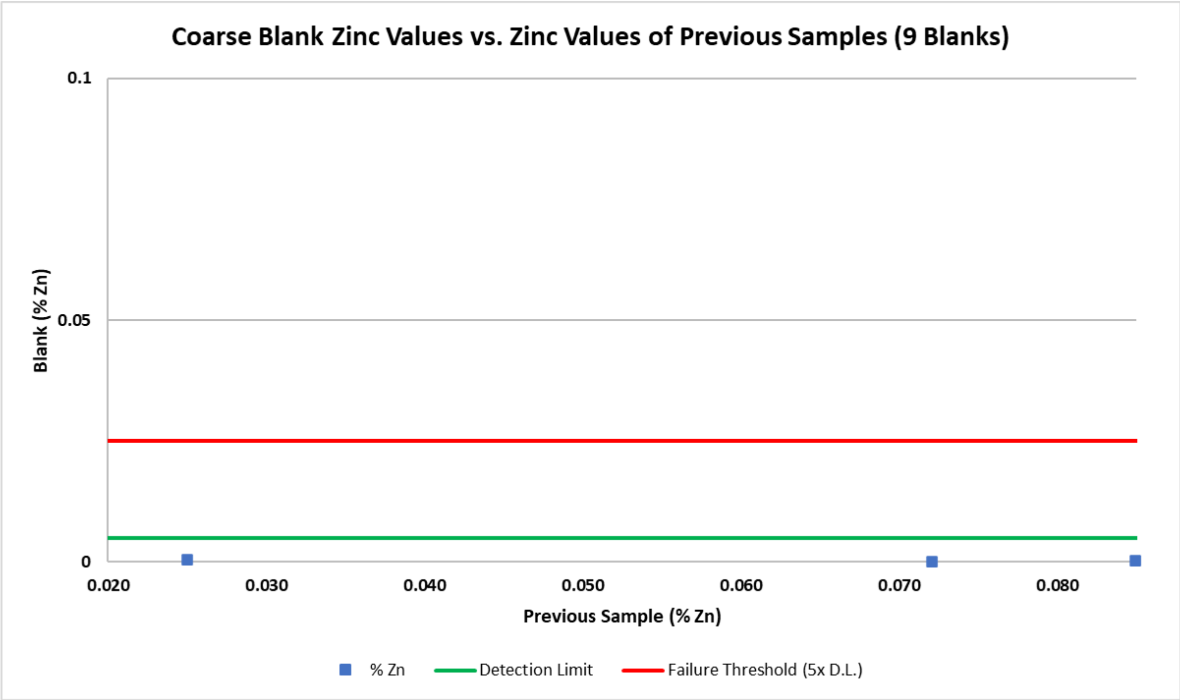


Figure 11.7 Coarse Blank Zinc Values vs. Zinc Values of Previous Samples



Core-Duplicates. Core field duplicates are secondary splits of original core samples collected simultaneously with the primary sample splits. One half split core is quartered to create the duplicate. Core duplicates are used to evaluate the total variability introduced by subsampling, including in the laboratory as well as the variability in the analyses. Core-duplicates should therefore be analyzed by the primary analytical laboratory.

Excelsior's resampling program included a total of 14 pairs of copper analyses, 14 pairs of zinc analyses, and 27 pairs of silver analyses. Figure 11.8 is a relative-difference graph that compares the RC duplicate data to the primary samples.

Figure 11.8 Core-Duplicate Copper Results Relative to Primary Sample Assays

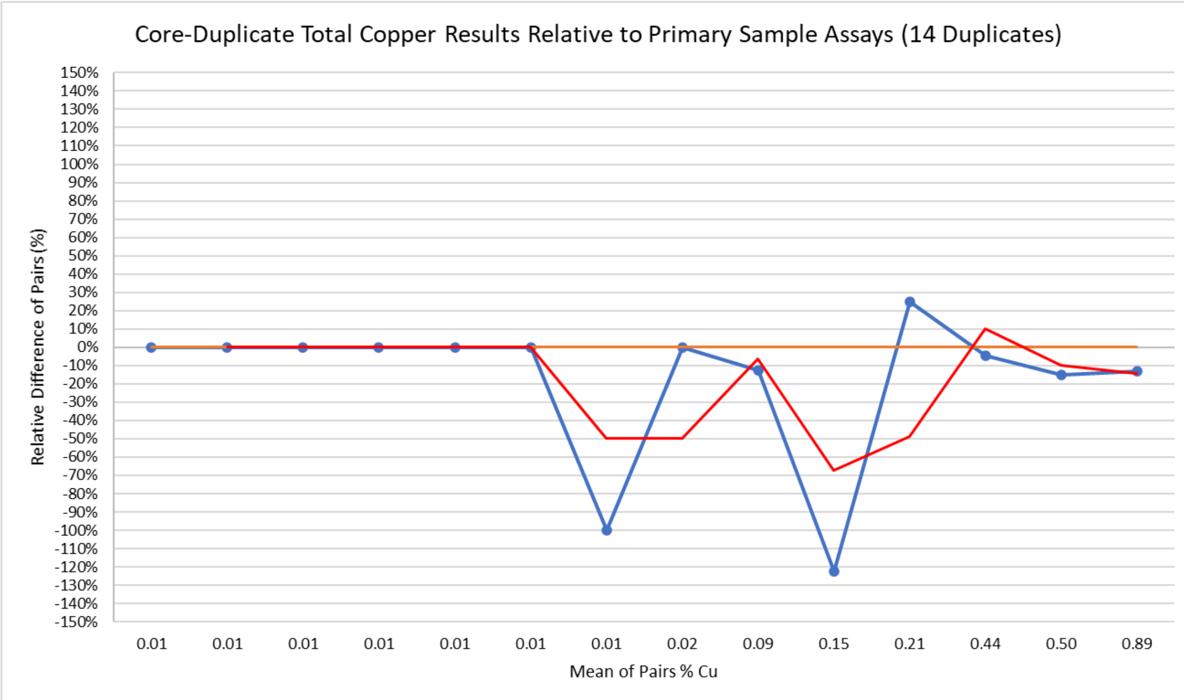


Figure 11.9 Core-Duplicate Zinc Results Relative to Primary Sample Assays

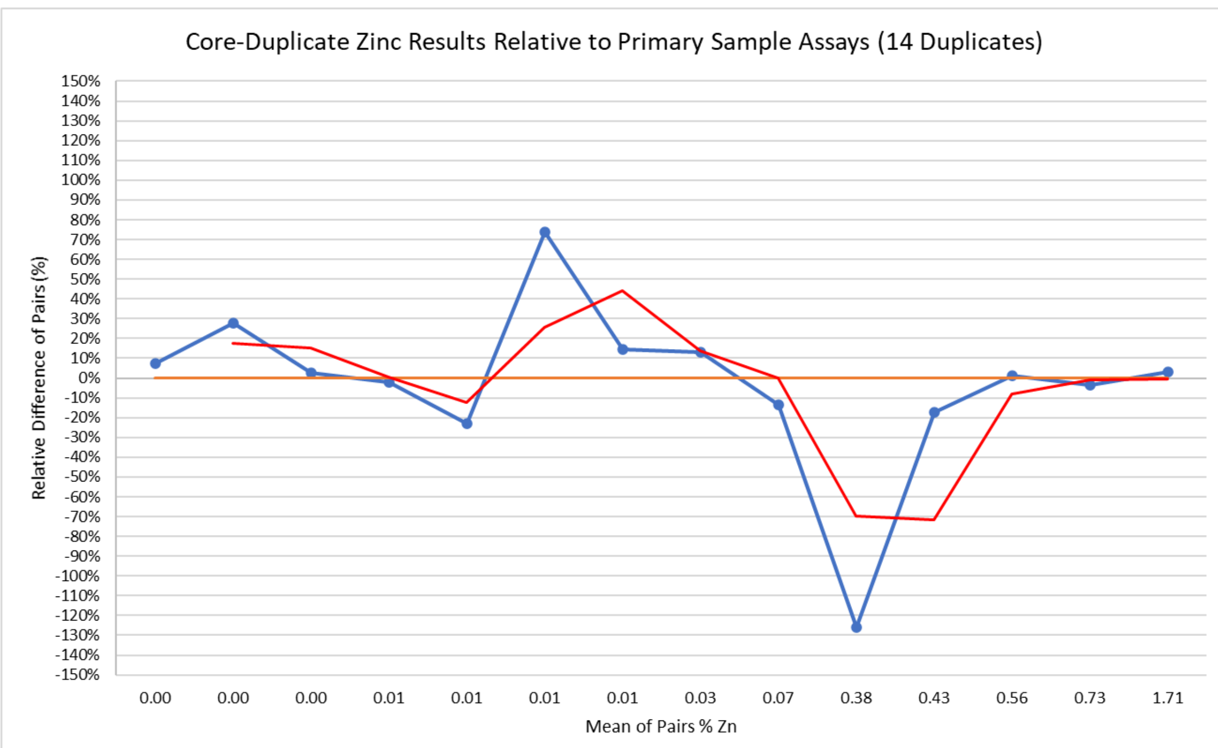
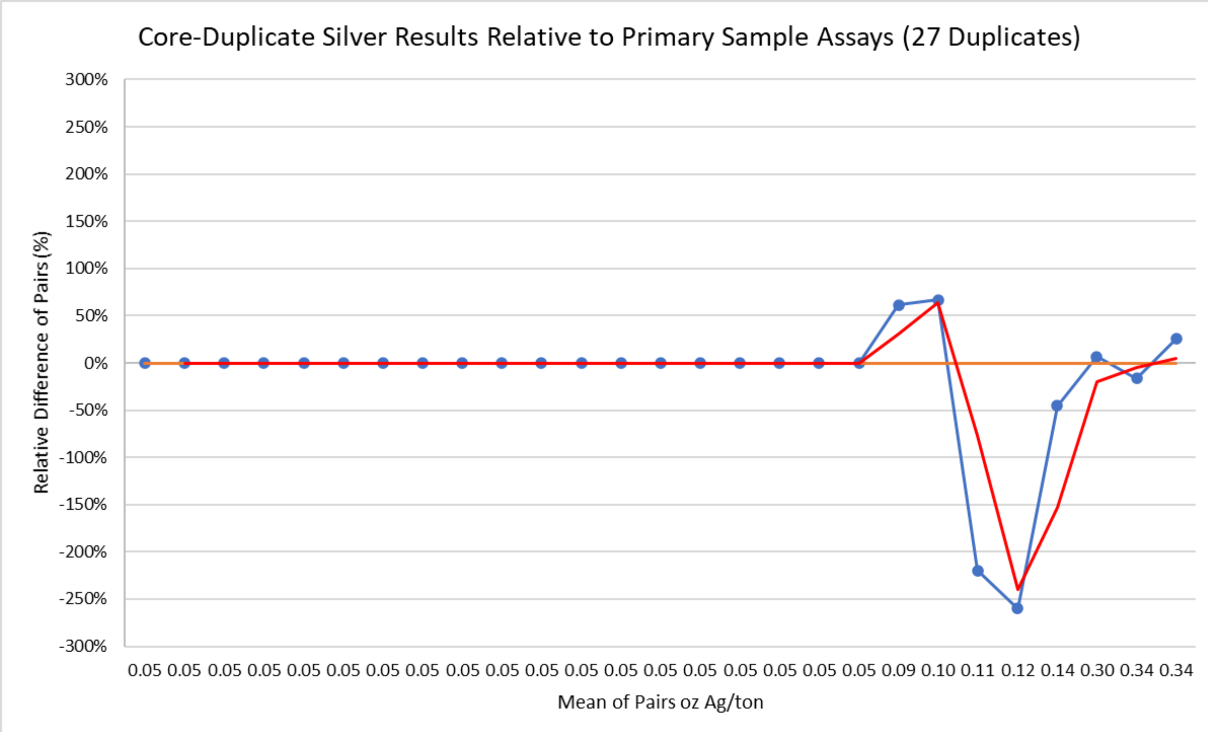


Figure 11.10 Core-Duplicate Silver Results Relative to Primary Sample Assays



There is no obvious bias in the duplicate sample results. The average assay duplicate assay values for copper, zinc, and silver, are all within 10% of the average original values. No outliers were removed.

11.5 Summary Statement

The certification status of the analytical laboratories is not known. Mr. Bickel is not familiar with AARL. Southwestern Assayers and Chemists is the predecessor to Skyline. Mr. Bickel believes these were independent commercial laboratories that were widely recognized and used by the mining industry at that time.

Documentation of the methods and procedures used for historical sample preparation, analyses, and sample security, as well as for quality assurance/quality control procedures and results, is incomplete and in many cases not available. Despite this, a majority of the historical assay certificates have been preserved and Excelsior was able to reasonably duplicate the original results (described in 12.2.4). Mr. Bickel is therefore satisfied that the historical analytical data are adequate to support the current resources, interpretations, conclusions, and recommendations summarized in this report.

Excelsior's sample preparation and analyses were performed at a well-known certified laboratory, and the sample security and QA/QC procedures are adequate to support the current resources, interpretations, conclusions, and recommendations summarized in this report.

12.0 DATA VERIFICATION (ITEM 12)

Mr. Bickel has verified the Strong and Harris project database and compiled and analyzed available quality QA/QC data collected by Excelsior. Data verification, as defined in NI 43-101, is the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used. There were no limitations on, or failure to conduct, the data verification for this report other than those discussed later in this section. Additional confirmation on the drill data's suitability for use are the analyses of the Strong and Harris project QA/QC procedures and results as described in Section 11.4.

12.1 Site Visit

Mr. Bickel visited the Strong and Harris project site on multiple occasions in early 2021. Initially, Mr. Bickel visited the project site on January 28-29, 2021, and then again on multiple occasions between February 12 and March 26, 2021. The latter dates coincided with MDA's work to assist Excelsior in a re-sampling campaign. During the site visits Mr. Bickel inspected the surface geology of the Strong and Harris deposit area; reviewed historical drill core and the methods and procedures used for Excelsior's sampling process; and carried out discussions of the current geologic interpretations with Excelsior personnel. Mr. Bickel also independently verified drill hole collar locations by inspecting drill sites and obtaining collar coordinates with a hand-held GPS receiver (see below).

Drill site and mineralization verification procedures were conducted, and sampling procedures were appraised. Mr. Bickel has also maintained a relatively continual line of communication through telephone calls and emails with Excelsior personnel in which the project status, procedures, and geologic ideas and concepts have been discussed. The result of the site visits and communications is that the author has no significant concerns with the project procedures.

12.2 Database Verification

The current drill-hole database, which supports the resource estimation of the Strong and Harris project area, was created by MDA using the drill-hole collar coordinates, hole orientations, and analytical information, including laboratory reports of analyses, in the original historical paper records in the possession of Excelsior. This drill-hole information was then supplemented with Excelsior's surveying and sampling data, and results through July 1, 2021. The historical information was subjected to various verification measures, the primary one consisting of the core re-sampling campaign conducted by Excelsior and MDA personnel under Mr. Bickel's supervision in 2021.

12.2.1 Drill-Collar Verification

In February of 2021, MDA directly received drill-hole collar location data from Darling Environmental & Surveying, Ltd. ("Darling") of Tucson, Arizona. These data were collected for Excelsior during a survey of the property using digital GPS equipment. Locations of 97 drill-hole collars were provided and MDA added them to the Excelsior database. During his site visits, Mr. Bickel independently checked a number of these locations with a hand held GPS and found them to reasonably match the collar coordinates received from Darling.

Prior to 2021, collar information was found in the historical documentation for 127 of the Strong and Harris drill holes. Collar coordinates were given in a local grid system, which were then converted to a UTM projection with NAD27 datum using a two-point transformation derived from handheld GPS measurements of the existing collars. The remaining 25 holes without historical coordinates, and which were otherwise not located in the field, were assigned coordinates from historical maps of the collars.

Excelsior updated the collar coordinates of these holes directly in the database to reflect the new survey data. Additionally, a new two-point transformation was created based on the data from the Darling survey and used to update collar locations which could not be located in the field.

12.2.2 Down-Hole Survey Verification

Down-hole deviation data exists for only two drill holes in the historical records (SH-109 and SH-118). These data were verified to match the original paper records to the drill hole database by Mr. Bickel. Historical logs also indicated the planned deviation of the Strong and Harris drill holes, all of which were planned vertically. Based on the data from SH-109 and SH-118, which indicated that both had minimal deviation from their planned vertical orientation, it is reasonable to assume that most of the drill holes are generally vertical. However, the database lacks the spatial precision associated with a more complete set of deviation data.

12.2.3 Assay Database Verification

Historical Assays: Historical paper records, including copies of original assay certificates, and to a lesser extent, handwritten assay values included on geologic logs, were reviewed, transcribed into the digital database and audited under the supervision of Mr. Bickel. Assay data from the original lab certificates represents 92% of the historical assay information in the Strong and Harris database. The remaining 8% of historical assays were transcribed from geologic logs where no data from the original assay certificates existed. During the audit, Mr. Bickel compared the transcribed assays in the database to the certificate and log copies. Some discrepancies were found between the original assay certificates and the hand-written values in the logs, where both existed. These discrepancies were determined to be either transcription errors or, in some cases, the values on the logs appeared to be from re-assay values but the matching re-assay certificates were not found. It is possible that historical assays in the Strong and Harris database taken from hand-written values in the geologic logs are subject to the same transcription errors noted in audit. However, Mr. Bickel does not consider this risk to be material.

Excelsior Assays: MDA received electronic records from the assay lab with the results from Excelsior's 2021 re-sampling program. These data were added to the database by MDA for any drill hole intersection that did not already have a historical assay value. The remainder, which were duplicate assays of historical intervals, were compared to the historical analyses for verification, but did not otherwise replace the historical values in the database. The results of this comparison are summarized below.

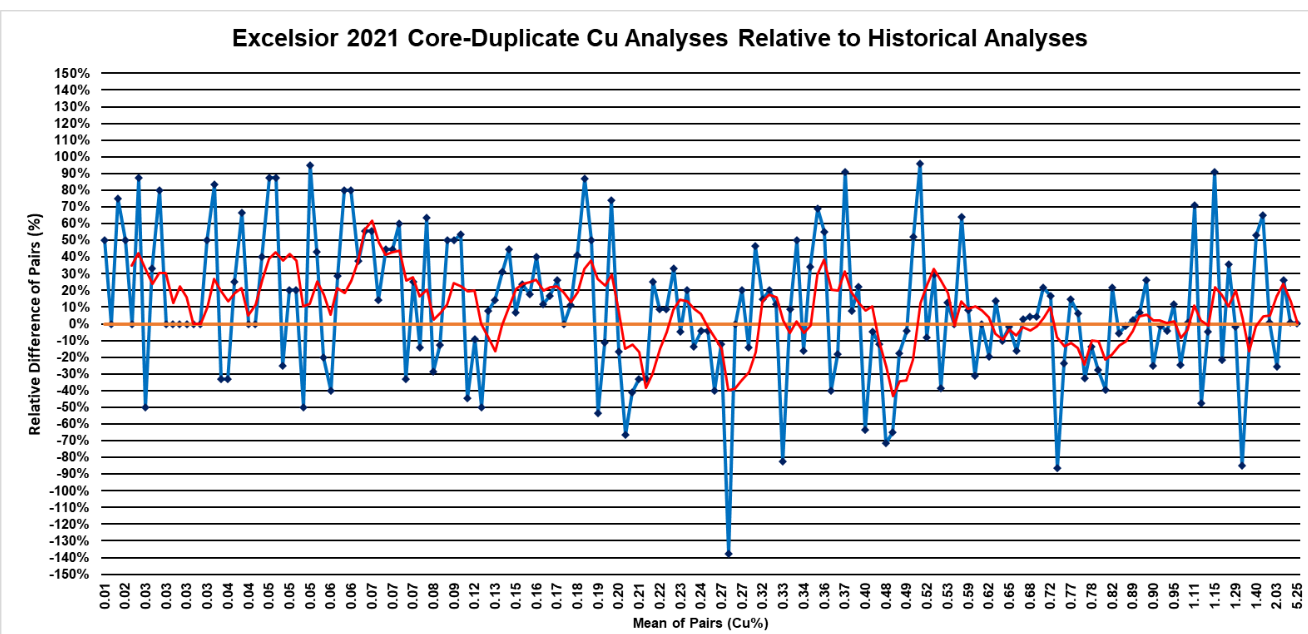
12.2.4 Excelsior 2021 Re-Samples

Excelsior re-sampled selected intervals of historical drill core based on MDA's recommendations and submitted them to Skyline for analysis. The samples were selected from a spatial distribution of drill holes throughout the deposit, as well as a distribution of drill holes from the various historical operators who originally drilled and explored the property.

Results from the re-sampled intervals of ¼ core represent core-duplicate analyses. Mr. Bickel compared the 2021 core-duplicate analyses with the historical analyses in the MDA database and conducted a mean of pair ("MOP") analysis.

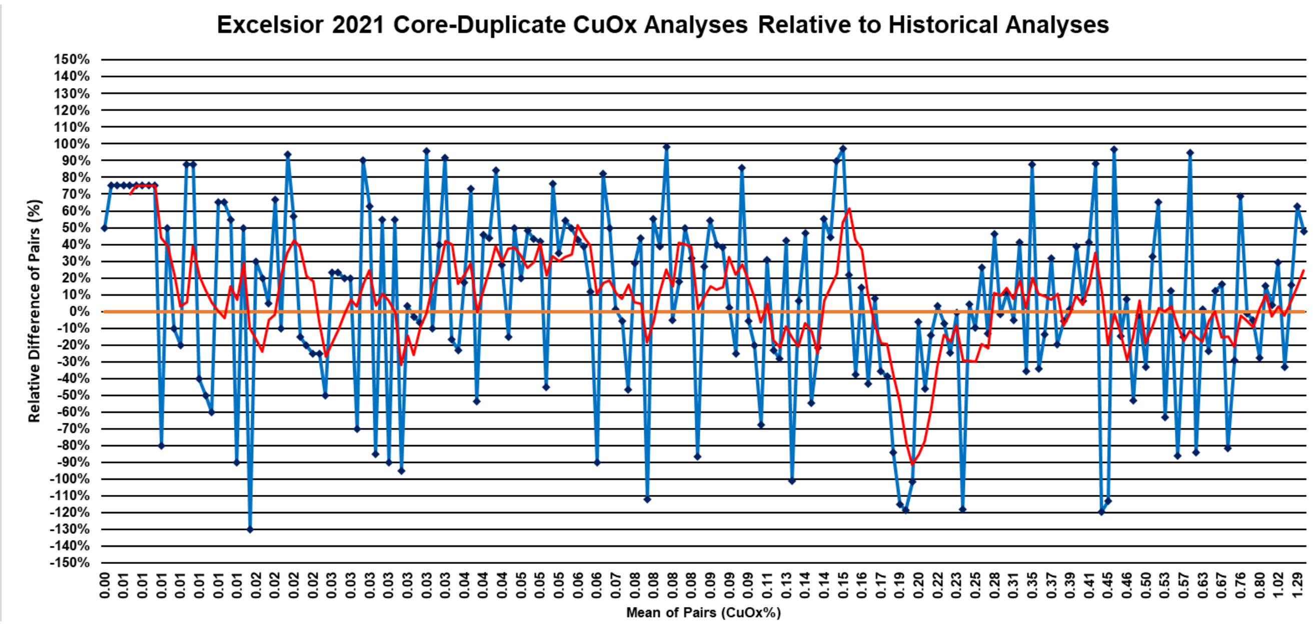
The MOP analysis for total copper ("Cu") samples is provided in Figure 4.1Figure 12.1. A total of 185 samples were submitted to Skyline for analysis. Ten of these samples (5%), were considered outliers and have been excluded from the results. The average relative difference between the new data and historical data is 9%.

Figure 12.1 Total Copper ("Cu") Core-Duplicate Analyses Relative to Historical Analyses



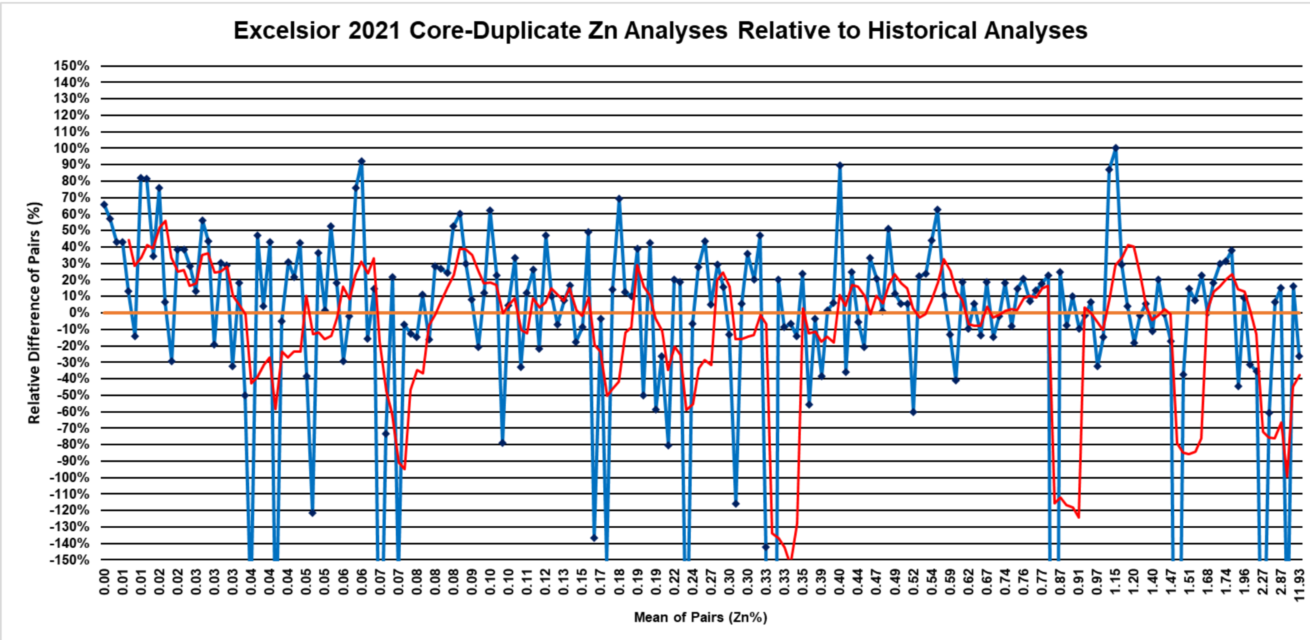
The MOP analysis for soluble copper ("CuOx") samples is provided in Figure 12.2. A total of 211 samples were submitted to Skyline for analysis. Twenty of these samples (9%), were considered outliers and have been excluded from the results. Increased outlier results from the soluble copper assays is expected relative to total copper due to the tendency of soluble copper minerals to be hosted in fine material which is easily shaken, mobilized, or otherwise lost from the core boxes in the historical handling and sampling of core. The average relative difference between the new data and historical data is 8%.

Figure 12.2 Soluble Copper ("CuOx") Core-Duplicate Analyses Relative to Historical Analyses



The MOP analysis for zinc ("Zn") samples is provided in Figure 4.1Figure 12.3. A total of 207 samples were submitted to Skyline for analysis. Eleven of these samples (5%), were considered outliers and have been excluded from the results. The average relative difference between the new data and historical data is 10%.

Figure 12.3 Zinc ("Zn") Core-Duplicate Analyses Relative to Historical Analyses



12.3 Independent Verification of Mineralization

Verification of mineralization was conducted during Mr. Bickel's participation in Excelsior's 2021 sampling campaign. In this period, Mr. Bickel was able to extensively investigate and verify the mineralization in the deposit and its relationship to relevant geology by comparing the 2021 analytical results to notes directly to the mineralized drill core. During several site visits in 2021, outcrops with visible copper and zinc mineralization were observed a short distance west from the Strong and Harris deposit. The existence of the Strong and Harris deposit has been widely known in the industry for many years prior to Excelsior's involvement, based on the results of drilling programs conducted by major exploration companies (Cyprus, Superior, and Continental) that were well-known and reputable operators.

12.4 Summary Statement on Data Verification

Mr. Bickel has undertaken extensive verification of the historical data. This work has identified very few errors in the transcription of assay data into the mine-site drill-hole databases. In addition, the core-duplicate analyses performed in 2021 allowed Mr. Bickel to verify that the historical assay data in the Strong and Harris database is of sufficient quality for use in the estimations of the current resources.

Explicit modeling of the copper, zinc, and silver mineralization was the most critical component to the estimation of the project mineral resources. This 'hands-on' approach provided meaningful verification of the historical data, whereby continuity and sensibility of meaningful geological variables, and the assays in the context of those variables, were carefully evaluated and considered.

Mr. Bickel experienced no limitations with respect to data verification activities related to the Strong and Harris project other than the limited availability of original-source assays discussed previously. In consideration of the information summarized in this and other sections of this report, Mr. Bickel has verified that the project data are adequate as used in this report, most significantly to support the estimation and classification of the mineral resources reported herein.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING (ITEM 13)

This section has been prepared by Mr. Robert Bowell of SRK Consulting at the request of Excelsior. Mr. Bowell has reviewed the information summarized below and believes it is correct as presently understood.

13.1 Introduction

To date, limited metallurgical testwork has been undertaken on the mineralized materials from the Strong and Harris deposit. These studies have focused on the amenability of the material to acid leaching and bulk flotation work. The recognition of oxide, sulfide and a transitional zone between the two prompts consideration of different process methods. Sulfide material is planned to be processed through a two-stage flotation process whilst transitional and oxide material is planned to be processed on a heap leach. Historically, the nearby Johnson Camp deposit has shown amenability to both options for respective sulfide and oxide mineralized material types (Argall, 1976).

In this section reference will be made to several studies that were initiated on the mineralization at Strong and Harris, as well as on the adjacent Johnson Camp mine and the Gunnison project. These are:

- Mountain States Research & Development ("MSRD"), 1974 (March), Preliminary Metallurgical Tests on Samples of Sulfide, Mixed and Oxide Copper-Zinc Ores: report to Superior Oil Company, Minerals Division. Project 2086, March 20,1974, 53p.
- Patel, R., 1993 (December), Bottle Roll Leach Test, Johnson Camp Samples, ML-2093: internal report, Cyprus Minerals, December 8, 1993, 4p.
- Patel, R., 1996 (January), Small Column Leaching of I-10 Drill Core Samples, Single Pass Leaching using Acidified Solution, ML2515: internal report, Cyprus Minerals, January 23, 1996, 13p.
- Hazen Research, Inc. ("HRI"), 2011 (September), Draft, HRI for Excelsior Mining Arizona, Inc., Project No 11245, "Copper Ore Column Leach Experiments."
- Roman, R.J., 2013 (July), Ore Leach Tests: report to Excelsior, July 31, 2013, 157p.

13.2 Geometallurgy

The mineralized zones at Strong and Harris have a strong control by oxidation. Although some logical overlap exists, there is no close correlation of zinc and copper oxidation. The mineralized material is inferred to have undergone 30% enrichment by supergene processes such as oxidation.

In the upper, oxide zone, copper mineralogy is strongly dominated by chrysocolla with reports of azurite and malachite in drill logs. Minor phases include; antlerite, brochantite, libethenite, tenorite, cuprite, copper and spangolite. Zinc mineralogy in the oxide zone is less well documented but aurichalcite, smithsonite and willemite have been described from the locality.

In the sulfide zone, the dominant copper sulfides are bornite, chalcopyrite and chalcocite. Zinc is present as sphalerite. Tetrahedrite group minerals as well as pyrite and pyrrhotite are also reported from the sulfide zone.

In between these two zones is a gradational transition zone that includes minerals from both zones and in addition has variable amounts of chalcocite and covellite. Broadly described, the zones occur vertically stacked within the deposit.

Typical copper and zinc grades, and acid-leachable proportions are shown in Table 13.1 (MSRD, 1974).

Table 13.1 Representative Head Grade for Strong and Harris Mineralization
(from MSRD, 1974)

Material Type	Copper %		Zinc %	
	Total	Acid Soluble	Total	Acid Soluble
Sulfide	1.09	0.29	2.15	0.43
Mixed	1.96	0.52	2.65	0.77
Oxide	0.93	0.79	1.35	0.73

It should be noted that acid-soluble phases in the sulfide zone are most likely for copper, chalcocite and covellite, and for zinc as Fe-rich sphalerite.

13.3 Sulfide Mineralized Material Testwork

Grinding of the sulfide material was done using 6.0 and 9.0 minute runs at 62.5% solids to determine p_{80} 212 µm grind (65 mesh). Screen analyses are shown in Table 13.2.

The 6-minute grind produced 99% passing a 212 µm screen. Initial flotation testwork using a conventional selective copper-zinc flotation approach was negative due to a high degree of zinc activation. Thus, no acceptable concentrates were initially produced. A bulk copper-zinc concentrate was later produced that developed bulk concentrates. From the initial 1974 testwork, test 18 gave the best overall recoveries at 9.4% copper and 8.2% zinc, with a ratio of 10.3:1 in the bulk rougher concentrate to give recoveries of 79.68% Cu and 66.9% zinc. Reworking of the tails including leaching by acid and ammonia, improved recoveries such that final copper and zinc recoveries from this approach were calculated as 94.8% copper and 86.7% zinc (MSRD, 1974).

Table 13.2 Sulfide Flotation Testwork, Strong and Harris Material
(from MSRD, 1974)

Test	Objective	Variable	Products	Grade		Recovery %		Concentration ratio
				Cu %	Zn%	Cu	Zn	
3	Cu-Zn separation	Na CaO No NaCN	No separation					
4	Cu-Zn separation	Zinc hydrosulfide	Concentrates combined with Test 5					
5	Cu-Zn separation	Sodium Sulfide	Concentrates combined with Test 5					
6	Grade + Separation	Cleaner	Cu Re cleaner Con	22.2	19.3			
			Cu Re cleaner tails	21.6	19.9			
			Zn clean con	15.8	18			
			Scavenger con	6.1	10.8			
9	Cu-Zn separation	NaF/SO_2	Cu Rough Con	18.1	20.8	50.16	53.95	
			Zn Rough Con	4.4	1.9	7.83	3.16	
			Bulk Rough Con	12.7	13.4	57.99	57.11	21.9
10	Cu-Zn separation	SO_2/Fe^0	Zn Con	15.2	22	8.25	10.68	
			Cu Con	16.5	24	40	52.18	
			Bulk Rough Con	13.9	20	50.45	64.86	22.5
18	Float common to all 3 ore types		Bulk Rough Con	9.4	8.2	79.68	66.89	10.3

**6 is a combined product of 4 and 5*

Con= concentrate

Rough= Rougher concentrate, initial stage in concentration to get more complete liberation of the valuable minerals. The primary objective of roughing is to recover as much of the valuable minerals as possible, with less emphasis on the quality of the concentrate produced

Clean= Cleaner concentrate, product of cleaning the rougher concentrate to remove undesirable minerals that also may have reported to the froth

Scav = Xcavenger flotation concentrate that is applied to the rougher tailings. The objective is to recover any of the target minerals that were not recovered during the initial roughing stage.

Additional 1974 work on the tailings sample indicated an additional copper recovery of 3.7% and zinc recovery of 1.9% extracted from the rougher flotation tailings by ammonia leaching. Overall, the estimated sulfide material recovery is shown in Table 13.3.

Table 13.3 Overall Recovery, Copper and Zinc Sulfides (MSRD, 1974)

Step	Cu, % of total	Zn, % of total
Floatable	79.8	67.7
Ammonia soluble from tails	3.7	1.9
Acid soluble from ammonia residue	13.3	28.9
Not recovered	3.2	1.5
Total	100.0	100.0

Based on mineralogy work at the time, the non-recovered copper and zinc appeared to be bi-modal with copper and zinc associated with clays and iron oxides, as well quartz encapsulated fine grained sulfides. Testwork on the composite of oxide and sulfide mineralized materials generally gave similar recoveries into the bulk concentrate as were obtained for the sulfide materials.

Recent advancement in sulfide flotation has demonstrated significantly better recoveries than those observed in the 1970s. Finer grinding coupled with sulfide flotation using MIC and SEX reagents, followed by a rougher oxide flotation using NaSH as a promoter, can provide significant upgrades into rougher concentrates. Recoveries of 85-95% for copper and zinc sulfides, even in partially mixed materials are expected.

13.4 Analogue Studies

Testwork performed on mineralized materials from the Tres Mares project in northern Mexico, with similar geology and mineralogy, demonstrated recoveries that averaged 84% for copper and 89% for zinc into concentrates using flotation methods proposed in this study. Even transitional or mixed type mineralization with more complex copper and zinc mineralogy has demonstrated recoveries that averaged 80.1% for copper and 69.7% for zinc (SRK, 2009). These recoveries were observed on a 2t per day pilot plant. Consequently, these numbers are thought more applicable than recoveries from 1974, as it is likely that the concentrates will not be leached on site with ammonia or sulfuric acid, but rather sold to a smelter where a silver credit can be gained that would equate to 4.2 oz/ton or more.

13.5 Oxide Mineralized Material Testwork 1970s

Initial testwork on the mixed and oxide material types from Strong and Harris involved flotation testwork coupled with ammonia and acid leaching. Leaching produced estimates of 93% copper and 82% zinc recovery with high acid consumption of 100 lbs/ton of material (MSRD, 1974).

Later, assessment of the I-10 deposit by bottle-roll tests gave predictions of copper extraction from four samples of 48% to 78% with an average of 63% (Patel, 1993). These samples were ground to a pulp and were considered a diagnostic test to determine if the copper is leachable. However, high acid consumption was also observed in the I-10 samples (Table 13.4).

Table 13.4 Summary of Bottle Roll Test Results, I-10 Deposit
(from Patel, 1993)

Mineralization Group	Copper Extraction %	Acid Consumption, lbs/ton material	Acid Consumption lbs/lb Cu
1	78.1	914	91.1
2	47.6	80	10.3
3	66.2	942	123
4	59.2	104	10

The high acid consumption was reflected in the high pH (between pH 8 and pH 9) observed for the samples and initial acidification required 100 g/L sulfuric acid to be added to obtain pH 1.5. During the test pH was measured every 24 hours and additional acid added as required to maintain pH below 2.0. In addition to copper and iron, high magnesium and manganese were reported and in tests 1 and 3 gypsum had precipitated.

In 1996 Cyprus undertook further internal testwork on the oxide material in the “I-10” deposit. The small-column testing was not particularly successful and reported 28.6% copper recovery despite reporting total copper assays of 0.48% and acid soluble copper of 0.28% (Patel, 1996). Acid consumption was relatively high at 52.7 lb/ton, or 19.2 lb/lb copper. Few details are given of the material and it is likely that the material included oxide, mixed and sulfide mineralization collected from various intervals from 780 ft to 870 ft in depth from one hole (MCC-7).

Excelsior is currently using ISR for mining copper at the Gunnison project south of the Strong and Harris project. Extensive leach testwork designed to assess ISR potential at Gunnison has been undertaken that is applicable to evaluating heap leaching of copper and zinc from the Strong and Harris deposit. The mineralization at Strong and Harris is hosted in the same lithologies as Gunnison and occurs as skarn type deposits. The most applicable work has been on crushed material in columns both saturated and unsaturated similar to conventional column tests (Roman, 2013). The purpose of the test program was to determine how the response of the mineralization is affected by changes in irrigation rate and acid concentration of the leach solution.

In the 2013 column test work, 24 column tests were run with the purpose of assessing variability in the deposit and as such are the most applicable tests on oxide and mixed material. The column tests were run at Mineral Advisory Group Research & Development, LLC (“MAG”) in Tucson, Arizona (Roman, 2013).

After filling each column with a sample of mineralization, the columns were filled with leach solution made from raffinate from the Johnson Camp mine that had been adjusted with sulfuric acid to 15 g/L free acid. The irrigation rate was initially set to 0.26 gpd for the first 15 days then increased to 1.85 gpd. PLS samples were collected daily. The pH and oxidation-reduction potential (“ORP”) of each PLS sample were measured, and the solution was then assayed for free acid and copper. Initially there were 24 columns to be leached.

Six of the columns became impermeable due to clay precipitation and were discontinued leaving 18 columns. It is believed that the six samples in these columns had been crushed to minus 1.0 inch, contained excessive fines which restricted their permeability. A seventh column was added containing a duplicate of the Column 6 sample material. This column was designated Column 6B. Column 6B was irrigated at a rate of 5.3 gpd. Sample materials are summarized in Table 13.5.

Table 13.5 Summary of 2013 Column Tests

MAG Column #	Formation	Fracture Intensity	Cu(tot), %	Cu(AS), %	SI	Sample Source
2	Upper Abrigo	Low	0.431	0.312	0.724	NSM-001 (156 lbs),NSM-002 (175 lbs),NSM-004 (19 lbs)
3	Abrigo Undivided	High	0.581	0.326	0.562	NSM-001 (71 lbs),NSM-002 (112 lbs),NSM-003 (27 lbs),NSM-004 (19 lbs),NSM-006 (126 lbs)
6 & 6B	Martin/Escabrosa	High	1.701	1.163	0.684	NSM-001 (66 lbs),NSM-002 (23 lbs),NSM-004 (50 lbs),NSM-006 (98 lbs),NSM-007 (131 lbs),NSM-010 (17 lbs),NSM-005A (11.5 lbs)
9	Martin/Escabrosa	Low	0.316	0.259	0.818	NSM-005A (350 lbs)
10	Martin/Escabrosa	High	0.895	0.683	0.763	NSM-005A (61 lbs), NSM-010A (142 lbs), NSM-011 (159 lbs)
11	Upper Abrigo	Low	0.409	0.320	0.784	NSM-005A (350lbs)
12	Abrigo Undivided	High	0.451	0.288	0.639	NSM-006 (58 lbs),NSM-007 (35 lbs),NSM-005A (257 lbs)
13	Abrigo Undivided	High	1.081	0.142	0.132	NSM-005A (118 lbs), NSM-011 (282 lbs)
14	Upper Abrigo	Low	0.601	0.363	0.605	NSM-005A (350 lbs)
15	Upper Abrigo	Low	0.525	0.365	0.695	NSM-005A (84 lbs), NSM-011 (267 lbs)
16	Middle Abrigo	Low	0.759	0.556	0.732	NSM-003 (32 lbs), NSM-006 (46 lbs), NSM-007 (17 lbs), NSM-008 (256lbs)
17	Middle Abrigo	Low	0.748	0.539	0.720	NSM-008 (114 lbs), NSM-011 (237.5 lbs)
18	Lower Abrigo	Low	0.634	0.450	0.710	NSM-008 (349.5 lbs)
19	Lower Abrigo	Low	0.826	0.480	0.581	NSM-008 (91 lbs), NSM-013 (265 lbs)
21	Middle Abrigo	Low	0.397	0.273	0.687	NSM-011 (97 lbs), NSM-013 (253 lbs)
22	Abrgo Undivided Transition	Low	0.510	0.353	0.693	NSM-011 (291 lbs), NSD-022 (59 lbs)
23	Abrgo Undivided Transition	Low	0.279	0.142	0.509	NSD-022 (350 lbs)
24	Middle Abrigo	Low	0.393	0.285	0.723	NSD-013 (356 lbs)

Figure 13.1 Results of Column Leaching, Upper Abrigo Formation
(from Roman, 2013)

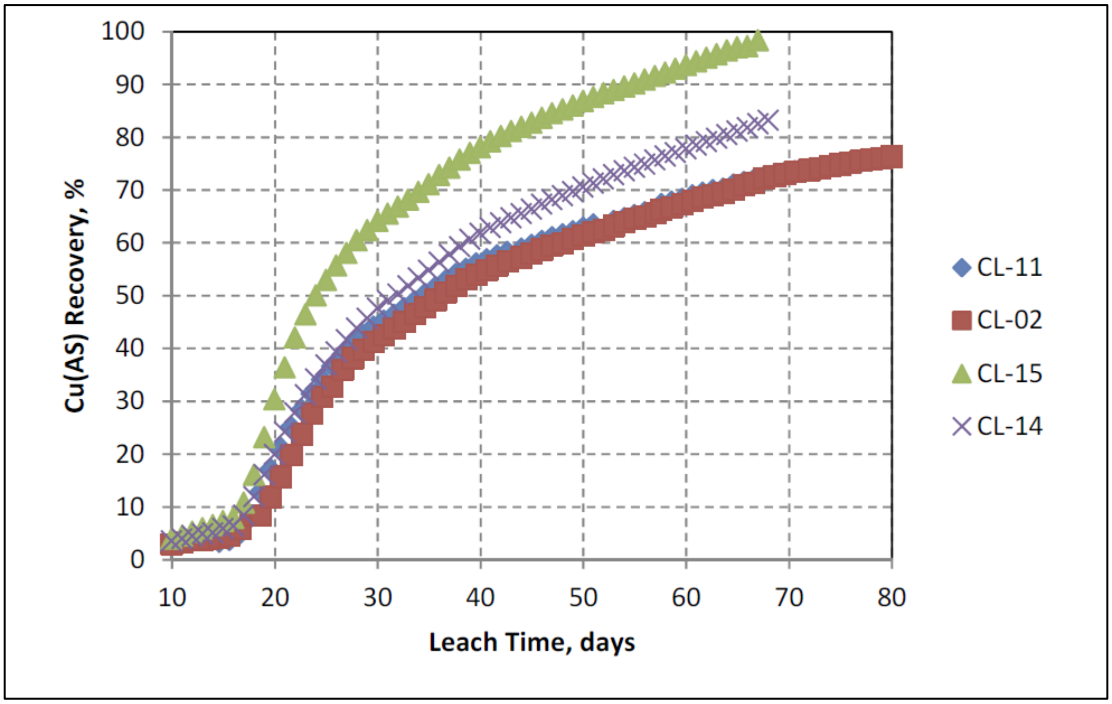


Figure 13.2 Results of Column Leaching, Middle Abrigo Formation
(from Roman, 2013)

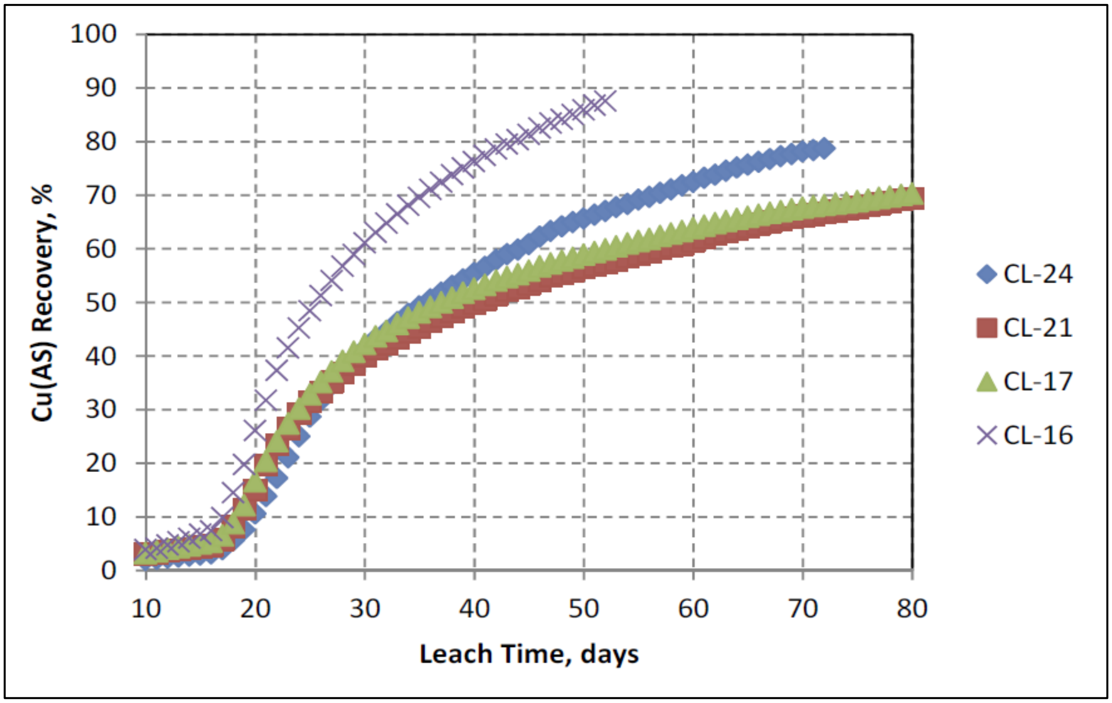
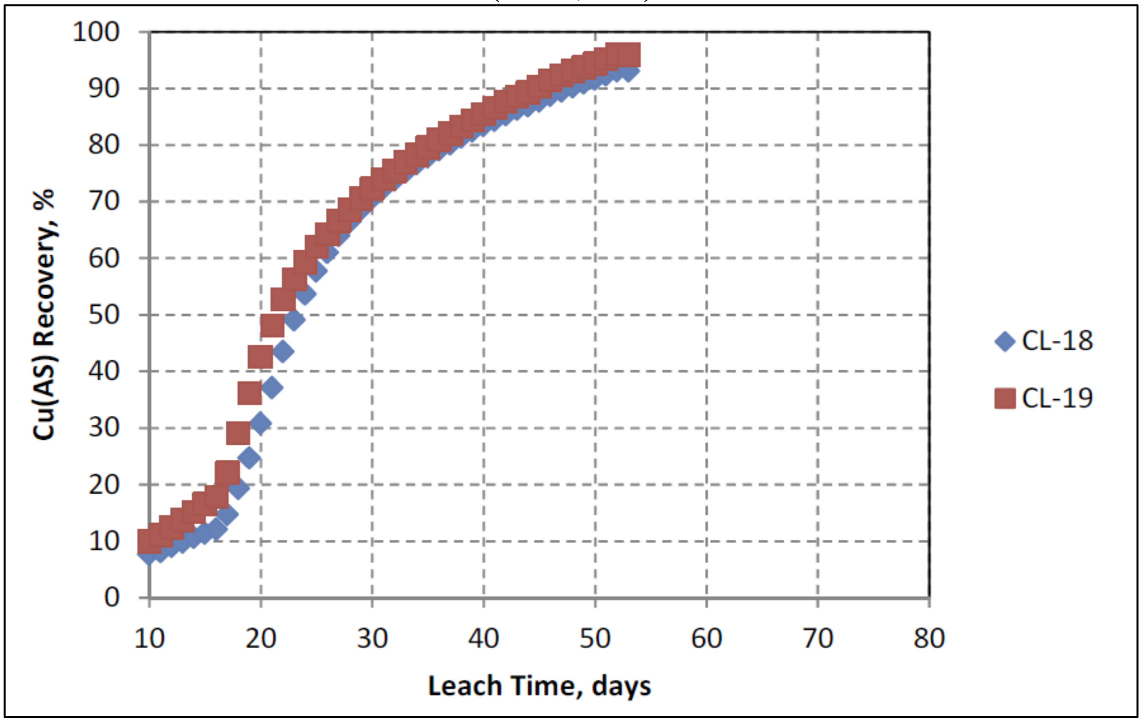


Figure 13.3 Results of Column Leaching, Lower Abrigo Formation
(Roman, 2013)



Summary operating parameters and results for the 2013 column tests are given in Table 13.6. Overall, the columns performed in a typical manner of copper oxide mineralization from similar deposits in Arizona. Acid-soluble copper recovery ranged from 88% to 112%. Total copper recoveries ranged from 60% to 80% depending on the initial solubility index, rock type, acid strength and leach solution flow rate of the columns. Higher acid concentrations and/or higher flow rates generated faster recovery rates. The above total acid soluble copper in the core samples reflect copper in forms other than acid soluble copper as defined by the standard sequential copper assay procedure. This is most likely secondary copper sulfides and some slow leached insoluble copper oxide, as well as a contribution from some leaching of chalcopyrite.

Table 13.6 Summary of 2013 Column Parameters and Results
(from Roman, 2013)

Sample Description	Column #	Irrigation rate, lph/m^2	Leach Solution, gpl acid	Acid, gms/hr/m^2	Days Leached	Cum PLS grade, gpl Cu	Estimated Recovery, %	
							Cu(total)	Cu(AS)
Martin	1	22	10	220	179	0.19	60.8	89.2
	2	22	5 then 15 then to 20	110 then 330 then 440	179	0.21	70.0	98.5
	3	11	5 then 10	55 then 110	179	0.31	59.9	87.8
Upper/Middle Abrigo	4	22	15	330	119	0.24	79.8	102.1
	5	37	10	370	119	0.15	76.5	97.9
	6	22	10	220	119	0.21	74.3	95.0
	7	22	5	110	119	0.17	62.5	79.9
	8	11	5 then 10	55 then 110	179	0.24	67.6	98.7
Lower Abrigo	9	22	15	330	99	0.16	69.7	112.3
	10	37	10	370	76	0.11	67.9	109.3
	11	22	10	220	81	0.13	65.9	106.1
	12	22	5	110	99	0.10	57.3	92.2
	13	11	5	55	99	0.15	60.8	97.8

The 2013 column testwork demonstrated how the acid consumption was influenced by irrigation rate, acid concentration of the leach solution, rock type, and percentage of copper leached. However, the absolute values of acid consumption are significantly over estimated due to the test procedure and the nature of the sample preparation. This reflects the higher surface area in the drill core and crushing of sample to allow for column testing.

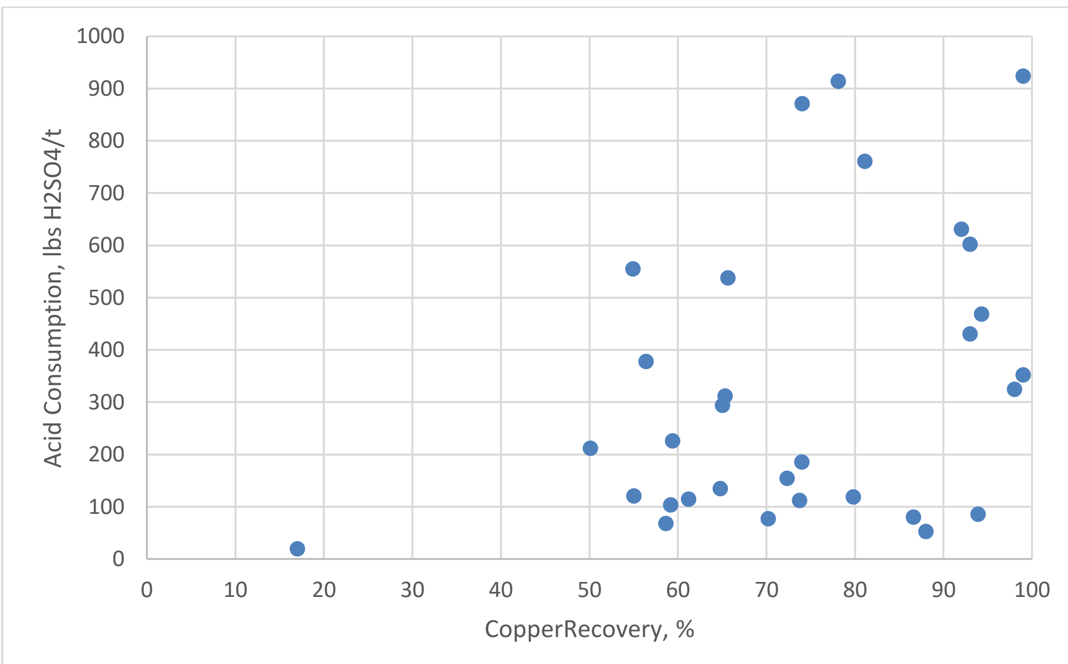
In addition, gangue acid consumption testwork determined that net acid consumption was predicted to be on the order of 11.6 to 28.2 lbs/lb Cu (Huss et al, 2014) as summarized in Table 13.7.

Table 13.7 Assessment of Acid Consumption
(from Huss et al, 2014)

Sample		Acid Consumption, g	Cu leached, g	Net Acid Consumption, lbs/lb Cu	Cu(AS) Recovery, %
C1	Uncoated	155.48	12.22	11.59	61.5
C2	Coated	26.95	8.20	1.75	55.0
F4	Uncoated	139.99	4.71	28.21	63.4
F3	Coated	72.72	5.75	11.10	55.8

For the heap leach assessment, C1 and F4 were considered the more reliable estimates of acid generation and pointed towards a higher acid consumption. Acid consumption in the column testing was extremely variable from estimates as low as 50 lbs/t (equivalent in those tests to ~ 19 lbs/lb Cu) to over 900 lbs/t (equivalent in that test to 123 lbs/lb Cu). The recovery of copper is positively correlated to acid consumption (Figure 13.4).

Figure 13.4 Acid Consumption for Copper-Zinc Mineralization in the Abrigo Formation
(source: compiled from all studies)



Apart from the work by MSRD (1974), extraction of zinc in column testwork is very limited. The few extraction tests undertaken using material from the Cochise district (citation for zinc extractions other than MSRD) indicate a similar range to that reported by MSRD.

13.6 Conclusions

Metallurgical testwork for the Strong and Harris project is at a conceptual level of understanding and currently relies on historical data and analogue work undertaken by Excelsior and others in the same mining district. Limited comminution work indicates that flotation has been challenged with this mineralized material and production of a bulk concentrate generated recoveries of 79.7% copper and 69.7% zinc for sulfide materials. Arguably better results have been obtained at other projects using a more modern scheme, and liberation and separation of the sulfides should be possible with better efficiency although this needs testing on material from site. Based on an analysis of all applicable column data, and assuming optimum acid consumption being around 100 lbs/ton of material to be processed, the overall estimate at the PEA stage is that copper and zinc extraction by acid leaching should be around 92.3% and 82.3% respectively.

13.7 Recommendations

The most pressing need is to generate metallurgical composites for the sulfide, mixed and oxide mineralized materials from Strong and Harris. These need to go through comminution testing to assess:

- Parameters for crushing and grinding conditions;
- Optimum grain size for separation for sulfide and mixed material types and;
- Re-assessment of flotation application using more modern approach for bulk, copper and zinc concentrates.

In addition, the use of more modern gravity methods to produce a bulk sulfide concentrate should be considered. In terms of the heap leach, it is clear copper and zinc can be leached from the mixed and oxide materials, and that a number of the economic host minerals react with sulfuric acid. However, acid consumption is potentially high in the deposit and this needs to be assessed. Options for reducing this could include;

- Comminution work to remove calcite prior to leaching and possibly tank leaching as opposed to heap leaching;
- Leaching of coarser fraction;
- Pre-treatment with organic acid; and
- Use of ammonia leaching as an alternative to sulfuric acid.

In addition, no testwork on metal production by SX-EW has yet to be undertaken and this needs to be undertaken particularly to ensure no cross-over of copper into the zinc recovery circuit. This has been applied commercially elsewhere but to define operating conditions at greater than PEA level, testwork is required. Additional tests should be considered for the other major rock types as well as for a greater variety of oxidation profiles throughout the deposit.

14.0 MINERAL RESOURCE ESTIMATES (ITEM 14)

14.1 Introduction

The mineral resource estimation for the Strong and Harris project was completed for disclosure in accordance with NI 43-101. The modeling and estimation of the copper, zinc, and silver mineral resources were completed in July, 2021 under the supervision of Jeff Bickel and Michael Gustin, both qualified persons with respect to mineral resource estimations under NI 43-101. The Effective Date of the resource estimate is September 9, 2021. Mr. Bickel, a former employee of Excelsior, is independent of Excelsior(s) by the definitions and criteria set forth in NI 43-101 as of the Effective Date of this report, as is Mr. Gustin. There is no affiliation between Mr. Bickel or Mr. Gustin and Excelsior(s) except that of independent consultant/client relationships. Mr. Bickel and Mr. Gustin are not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Strong and Harris mineral resources as of the date of this report. No mineral reserves have been estimated for the Strong and Harris project.

The Strong and Harris mineral resources are classified in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore NI 43-101. CIM mineral resource definitions are given below, with CIM's explanatory text 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.

The mineral resources are reported herein 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 and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction."

14.2 Data

The Strong and Harris copper, zinc, and silver resources were modeled and estimated using information provided by Excelsior in the database constructed by MDA under Mr. Bickel's supervision. The information is derived from historical core holes drilled by Cyprus Minerals, Superior Minerals, Continental Materials, Beard Mining, and AZCO. The drill hole database also includes analyses performed by Excelsior on the historical core. This data, as well as digital topography of the project area, were provided to MDA by Excelsior in a digital database in Arizona State Plane, East Zone coordinates in US Survey feet using the NAD27 datum.

All modeling of the Strong and Harris digital geology, mineral domains and estimation of the mineral resources were performed using GEOVIA Surpac mining software as well as proprietary software developed at MDA. The Strong and Harris resource block model extents and dimensions are provided in Table 14.1.

Table 14.1 Block Model Extents and Dimensions

In Feet	X	Y	Z
Min Coordinates	531,079	403,549	2,800
Max Coordinates	540,479	415,149	5,800
Block Size	20	20	20
Rotation	0	-45	0

14.3 Deposit Geology Pertinent to Resource Block Model

The copper-zinc-silver mineralization at Strong and Harris occurs primarily in Paleozoic sedimentary units. The primary controls on mineralization are (i) favorable stratigraphic units altered to various calc-silicate assemblages; (ii) a diabase sill (otherwise known as the "Peabody Sill") and adjacent stratigraphic units; (iii) the intersection of favorable units with important structures; and (iv) oxidation of primary mineralization. Geologic factors critical to the grade domain modeling of Strong and Harris copper-zinc-silver mineralization therefore include lithology, structure, and oxidation.

14.4 Geologic and Oxidation Models

At Excelsior's request, MDA constructed stratigraphic interpretations on a set of vertical, digital cross sections oriented at 045^o azimuth through the Strong and Harris deposit. These sections were spaced at 200-foot intervals over a strike extent of 10,000 feet, which covers the resource area. The stratigraphic units modeled on the cross sections include the Colina Limestone, Earp Formation, Horquilla Limestone, Black Prince Limestone, and the diabase sill. The sectional interpretations were then triangulated to create 3D surfaces or solids.

Fault surfaces were constructed using information from three sources: (i) Excelsior interpretations on cross sections; (ii) MDA interpretations on cross sections; and (iii) historical interpretations from Superior Minerals.

MDA also interpreted oxidation domains on the cross sections using logging data and the ratio of soluble copper assays to total copper assays. The mineralization was assigned to oxide, transition, or sulfide material types (domains). In general, if the ratio CuOx/Cu was greater than or equal to 60%, the mineralization was assigned to the oxide domain. If the ratio ranged between 25% and 60%, the mineralization was assigned to the transition material. Mineralization with a ratio of less than 25% soluble copper was assigned to the sulfide domain. These oxidation ratio rules were modified as needed with geological context. The cross-sectional oxidation domains were then triangulated into 3D surfaces.

14.5 Density

A total of 220 samples were taken from drill core by Excelsior and sent to Skyline for determinations of specific-gravity ("SG"). The samples were taken across a range of geological characteristics and spatial distribution in the deposit. The samples were analyzed using the water-displacement method.

MDA evaluated the SG results for use in the resource estimation and then assigned an average SG to each copper mineral domain as described in Section 14.6. The SG measurements were converted to tonnage factors as summarized in the Table 14.2.

Table 14.2 Average SG and Tonnage Factors by Copper Domain

Grade Domain	SG	TF
Outside Domains	2.65	12.1
Low-Grade	2.62	12.2
Mid-Grade	2.68	11.9
High-Grade	2.89	11.1

14.6 Mineral Domain Modeling

A mineral domain encompasses a volume of rock that is ideally characterized by a single, natural population of metal grades that occurs within a specific geologic environment. Mineral domains were modeled by MDA to respect the lithologic, structural, and oxidation interpretations of the deposit. Following statistical evaluation of the drillhole data, mineral domains were modeled on cross sections for each metal. Low-, mid-, and high-grade domains were modeled for copper and zinc, and were numbered 100, 200, and 300, respectively, for each of the two metals. Material outside the 100, 200, and 300 domains was assigned to the 0 domain. These grade domains were based on assay data populations. Low- and high-grade domains were modeled for silver (numbered 100 and 200). Soluble-copper domains were not explicitly modeled; instead, the soluble-copper to total-copper ratio was used in the block model to calculate the grade for soluble copper, described in detail below.

14.6.1 Copper, Zinc, and Silver Domain Modeling

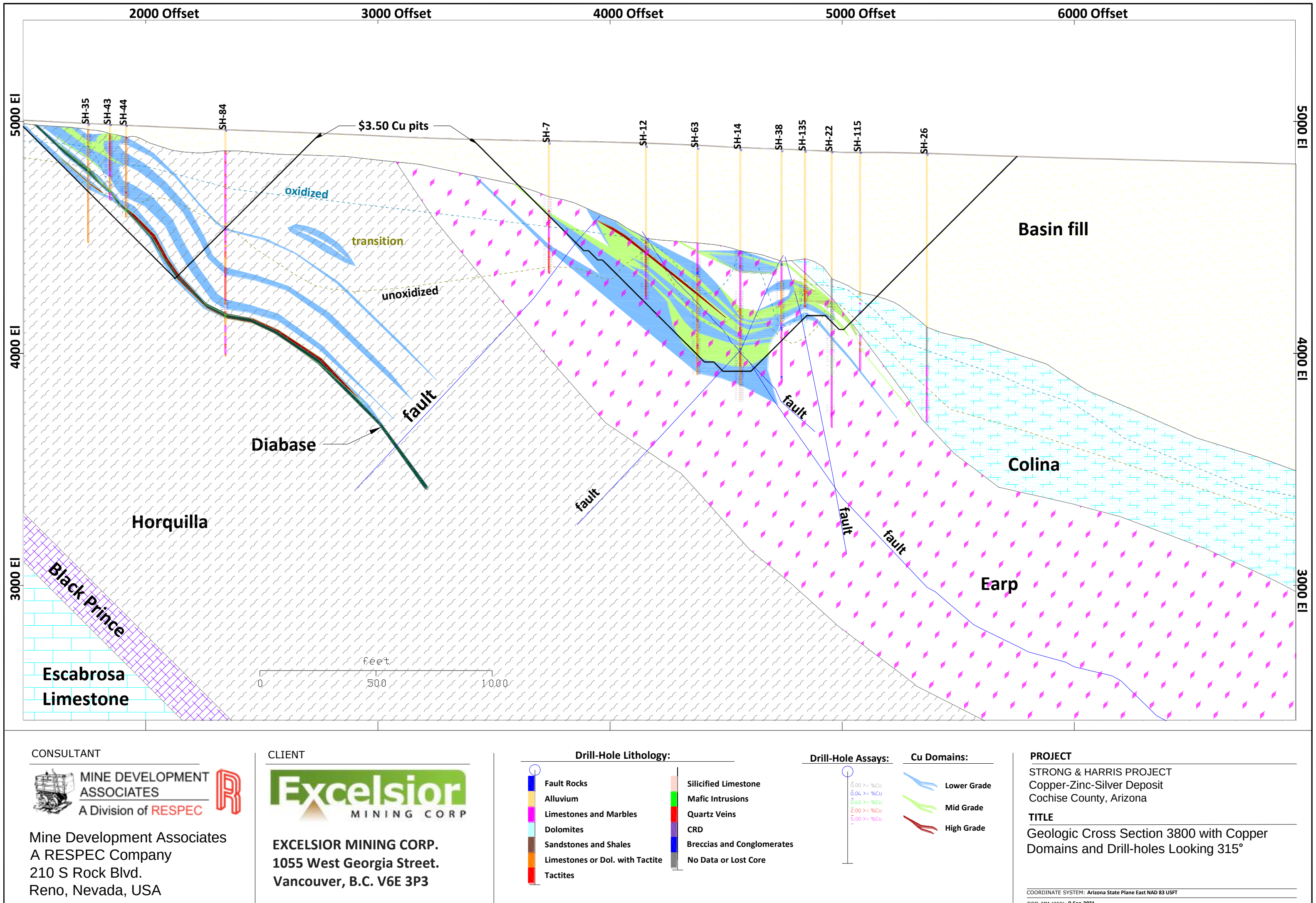
In order to define the mineral domains at Strong and Harris, the natural populations of copper, zinc, and silver grades were identified on separate population-distribution graphs for all drillhole samples in the deposit area. The analysis led to identification of distinct populations. Ideally each of these populations can be correlated with geologic characteristics which then can be used in conjunction with the grade populations to interpret the bounds of each of the mineral domains. The approximate grade ranges of the domains are listed in Table 14.3 for each metal.

Table 14.3 Grade Domain Ranges

Domain	Copper %
100	~0.04 to ~0.4
200	~0.4 to 2.0
300	> ~2.0
Domain	**Zinc %**
100	~0.04 to ~0.5
200	~0.5 to 2.5
300	> ~2.5
Domain	**Silver oz/ton**
100	~0.06 to ~0.2
200	> ~0.2

Using these grade populations in conjunction with lithologic and structural interpretations, grade domains were independently modeled for each metal within the Strong and Harris deposit by interpreting mineral domain polygons on a set of 200ft-spaced cross sections oriented along the approximate direction of dip (045° azimuth). While each metal was explicitly interpreted on every cross section, copper, zinc, and silver are generally spatially coincident throughout the deposit. Representative cross sections showing the copper, zinc, and silver mineral domains are shown in Figure 14.1, Figure 14.2, and Figure 14.3.

Figure 14.1 Geologic Cross Section with Copper Oxide Domains



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excelsiormining.com

PROJECT

STRONG & HARRIS PROJECT
Copper-Zinc-Silver Deposit
Cochise County, Arizona

TITLE

Geologic Cross Section 3800 with Copper Domains and Drill-holes Looking 315°

COORDINATE SYSTEM: Arizona State Plane East NAD 83 USFT
DDD-MM-YYYY: 9-Sep-2021
FILE NAME: H:\Excelsior\Strong_and_Harris\Reports\PEA2021_NI43-101\Figures\Cu_interp_sxn_3800.dwg

Figure 14.2 Geologic Cross Section with Zinc Domains

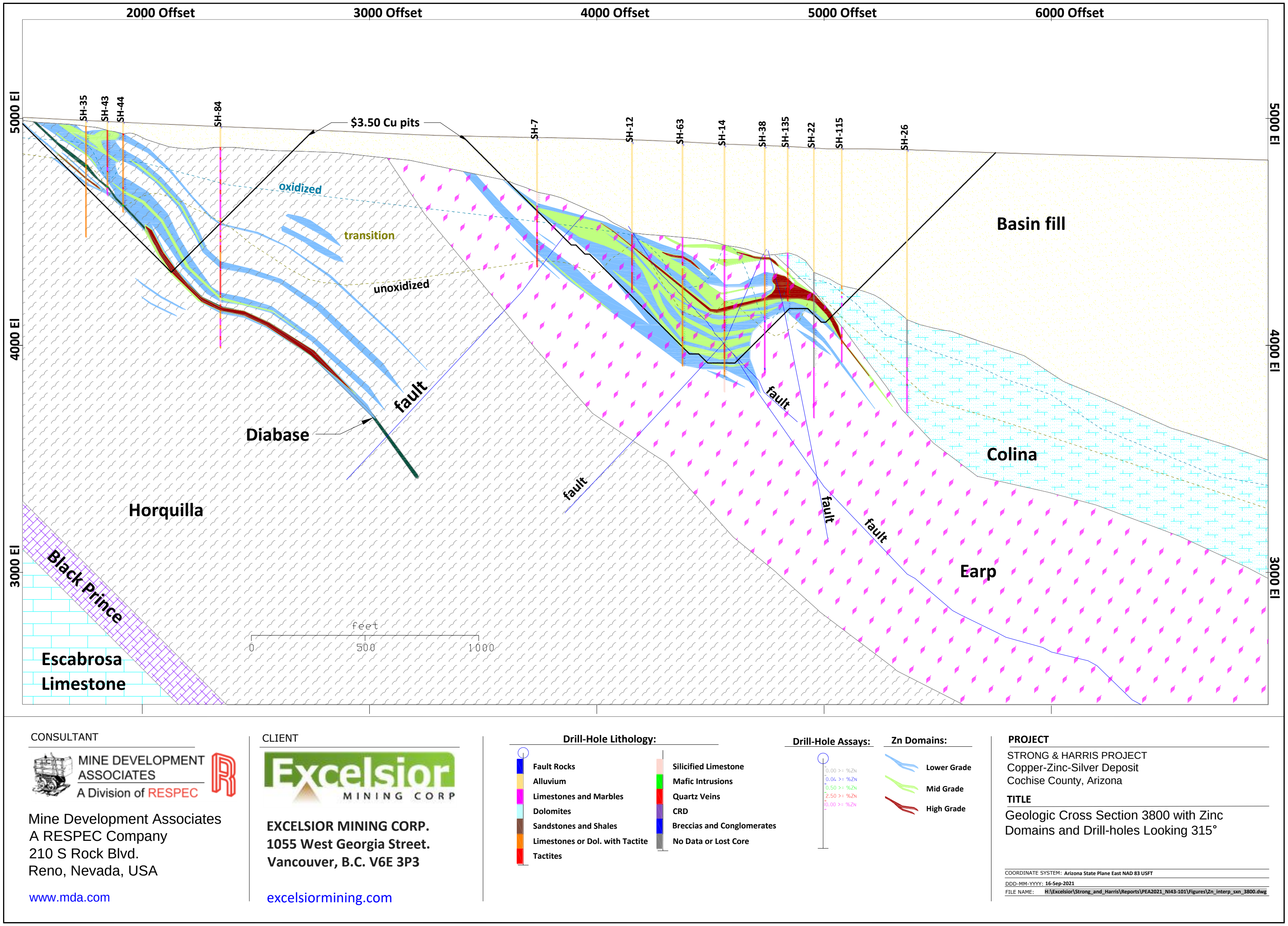
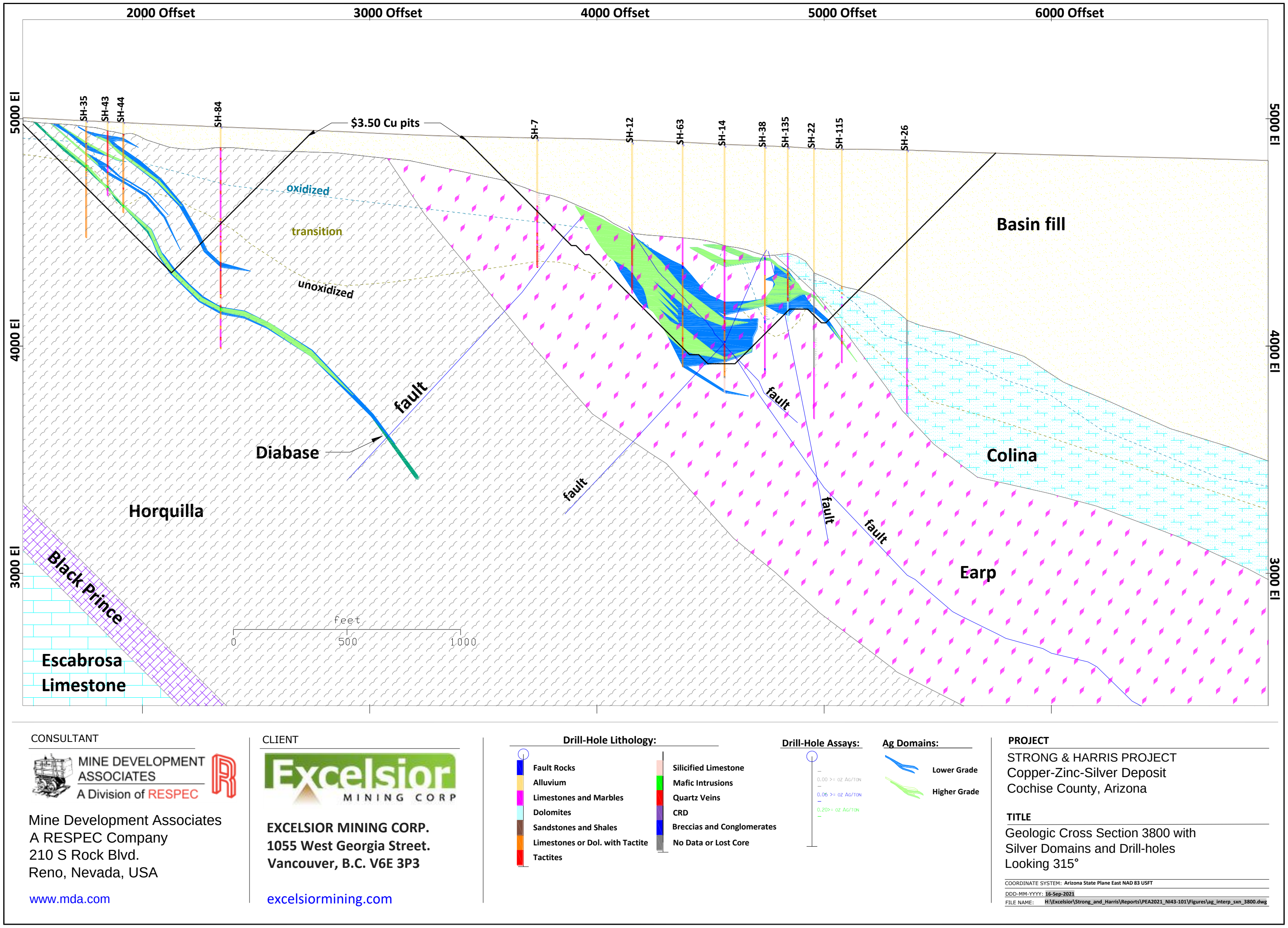


Figure 14.3 Geologic Cross Section with Silver Domains



14.6.2 Soluble-Copper Ratio

There are two methods for estimating soluble copper: directly, using composites of the soluble-copper analyses from the database; or indirectly, by estimating the soluble-copper to total-copper ratios ("Cu Ratio"). In the latter case, the ratios are determined for each drill interval that has both soluble- and total-copper analyses, and these ratios are then coded, composited, and used to estimate the ratios into the model blocks. The estimated soluble-copper model values are then derived by multiplying the estimated ratio by the estimated copper value in each block.

Remobilization of supergene copper is not evident at Strong and Harris. This is likely due to the remnant carbonate minerals in the host units that would have restricted the movement of acidic solutions during oxidation. In the Strong and Harris deposit, the ratios are generally uniform within each of the modeled oxidation zones despite some internal variability that is likely stratigraphically controlled.

The estimation of ratios for soluble copper can negate possible biases created by intervals that were selectively analyzed for total copper but not soluble copper. In the Strong and Harris database, 3% of the total copper samples do not have soluble copper analyses.

MDA used estimated ratios to code the Strong and Harris block model with soluble copper values. The ratio estimation was confined to blocks with estimated total copper values. The ratios of the blocks coded to the oxide, transitional, and sulfide zones were estimated independently.

14.7 Assay Coding, Capping, and Compositing

The cross-sectional mineral-domain polygons described in Section 14.6 were used to code drillhole assay intervals to their respective mineral domains for copper, zinc, and silver. The polygons were coded 100 feet either side of the section plane from which they were created. Soluble copper ratios were coded to the oxide, transitional, and sulfide domains using the oxidation surfaces. Assay caps were determined by domain to identify high-grade outliers that might be appropriate for capping. Visual reviews of the spatial relationships concerning possible outliers and their potential impacts during grade interpolation were also considered in the assay cap definitions. Table 14.4 provides the caps used by each domain for each metal.

Table 14.4 Grade Caps

Copper	Cap (% Cu)
0	0.2
100	1
200	4
300	20
Zinc	**Cap (% Zn)**
0	0.2
100	1.3
200	5
300	17
Silver	**Cap (oz/ton Ag)**
0	0.1
100	0.6
200	2
CuOx Ratio	**Cap (Ratio)**
0	1
100	1
200	1
300	1

Descriptive statistics of the coded assays of capped and uncapped copper, zinc, and silver analyses are provided in Table 14.5, Table 14.6, and Table 14.7. Copper ratio statistics are provided in Table 14.8, Table 14.9, and Table 14.10.

Table 14.5 Coded Copper Assay Statistics

Domain	Assays	Count	Mean (%Cu)	Median (%Cu)	Std. Dev.	CV	Min. (%Cu)	Max. (%Cu)
0	Cu	1160	0.02	0.01	0.05	2.8	0	4.45
	Cu Cap	1160	0.02	0.01	0.02	1.35	0	0.2
100	Cu	1411	0.14	0.11	0.11	0.81	0.005	2.16
	Cu Cap	1411	0.14	0.11	0.11	0.79	0.005	1
200	Cu	807	0.71	0.61	0.51	0.71	0.01	14.6
	Cu Cap	807	0.71	0.61	0.47	0.67	0.01	4
300	Cu	116	3.78	2.75	3.37	0.89	0.02	22.5
	Cu Cap	116	3.76	2.75	3.27	0.87	0.02	20
100+200+300	Cu	2334	0.46	0.19	1.01	2.19	0.005	22.5
	Cu Cap	2334	0.46	0.19	0.99	2.15	0.005	20

Table 14.6 Coded Zinc Assay Statistics

Domain	Assays	Count	Mean (%Zn)	Median (%Zn)	Std. Dev.	CV	Min. (%Zn)	Max. (%Zn)
0	Zn	1044	0.02	0.01	0.02	1.12	0.0009	0.41
	Zn Cap	1044	0.02	0.01	0.02	0.98	0.0009	0.2
100	Zn	1552	0.15	0.10	0.13	0.82	0	0.65
	Zn Cap	1552	0.15	0.10	0.13	0.82	0	0.65
200	Zn	650	0.89	0.78	0.52	0.58	0	2.49
	Zn Cap	650	0.89	0.78	0.52	0.58	0	2.49
300	Zn	123	5.66	4.05	4.27	0.75	0.09	18
	Zn Cap	123	5.65	4.05	4.23	0.75	0.09	17
100+200+300	Zn	2325	0.56	0.19	1.39	2.47	0	18
	Zn Cap	2325	0.56	0.19	1.39	2.46	0	17

Table 14.7 Coded Silver Assay Statistics

Domain	Assays	Count	Mean (oz/ton)	Median (oz/ton)	Std. Dev.	CV	Min. (oz/ton)	Max. (oz/ton)
0	Ag	674	0.05	0.05	0.01	0.16	0	0.18
	Ag Cap	674	0.05	0.05	0.01	0.14	0	0.1
100	Ag	439	0.13	0.14	0.05	0.38	0.001	0.42
	Ag Cap	439	0.13	0.14	0.05	0.38	0.001	0.42
200	Ag	271	0.34	0.26	0.38	1.14	0.001	7.6
	Ag Cap	271	0.33	0.26	0.24	0.73	0.001	2
100+200+300	Ag	710	0.21	0.17	0.26	1.25	0.001	7.6
	Ag Cap	710	0.20	0.17	0.18	0.88	0.001	2

Table 14.8 Coded Cu Ratio Statistics in Oxide Material

Domain	Assays	Count	Mean (Ratio)	Median (Ratio)	Std. Dev.	CV	Min. (Ratio)	Max. (Ratio)
0	Cu Ratio	172	0.76	1.00	0.26	0.34	0.25	1
	Cu Ratio Cap	172	0.76	1.00	0.26	0.34	0.25	1
100	Cu Ratio	410	0.79	0.82	0.18	0.23	0	1.2
	Cu Ratio Cap	410	0.79	0.82	0.18	0.23	0	1
200	Cu Ratio	246	0.86	0.91	0.18	0.21	0.02	1
	Cu Ratio Cap	246	0.86	0.91	0.18	0.21	0.02	1
300	Cu Ratio	41	0.92	0.98	0.16	0.18	0.1	1
	Cu Ratio Cap	41	0.92	0.98	0.16	0.18	0.1	1
100+200+300	Cu Ratio	697	0.82	0.87	0.18	0.22	0	1.2
	Cu Ratio Cap	697	0.81	0.87	0.18	0.22	0	1

Table 14.9 Coded Cu Ratio Statistics in Transition Material

Domain	Assays	Count	Mean (Ratio)	Median (Ratio)	Std. Dev.	CV	Min. (Ratio)	Max. (Ratio)
0	Cu Ratio	438	0.67	0.50	0.27	0.41	0	1
	Cu Ratio Cap	438	0.67	0.50	0.27	0.41	0	1
100	Cu Ratio	591	0.58	0.57	0.25	0.43	0	1
	Cu Ratio Cap	591	0.58	0.57	0.25	0.43	0	1
200	Cu Ratio	363	0.52	0.50	0.32	0.62	0	1
	Cu Ratio Cap	363	0.52	0.50	0.32	0.62	0	1
300	Cu Ratio	47	0.45	0.37	0.38	0.85	0	1
	Cu Ratio Cap	47	0.45	0.37	0.38	0.85	0	1
100+200+300	Cu Ratio	1001	0.55	0.55	0.28	0.51	0	1
	Cu Ratio Cap	1001	0.55	0.55	0.28	0.51	0	1

Table 14.10 Coded Cu Ratio Statistics in Sulfide Material

Domain	Assays	Count	Mean (Ratio)	Median (Ratio)	Std. Dev.	CV	Min. (Ratio)	Max. (Ratio)
0	Cu Ratio	510	0.65	0.50	0.32	0.49	0	1
	Cu Ratio Cap	510	0.65	0.50	0.32	0.49	0	1
100	Cu Ratio	365	0.32	0.20	0.29	0.9	0	1
	Cu Ratio Cap	365	0.32	0.20	0.29	0.9	0	1
200	Cu Ratio	166	0.17	0.09	0.22	1.23	0	0.97
	Cu Ratio Cap	166	0.17	0.09	0.22	1.23	0	0.97
300	Cu Ratio	27	0.12	0.08	0.17	1.47	0	0.96
	Cu Ratio Cap	27	0.12	0.08	0.17	1.47	0	0.96
100+200+300	Cu Ratio	558	0.27	0.17	0.28	1.01	0	1
	Cu Ratio Cap	558	0.27	0.17	0.28	1.01	0	1

The capped assays were composited at 10-foot down-hole intervals, respecting the mineral domain boundaries. Descriptive statistics of the composites for each metal are given in Table 14.11.

Table 14.11 Composite Statistics

Domain	Hole Count	Comp. Count	Mean	Median	Std. Dev.	CV	Min.	Max.
Cu Composites by Domain								
Domain	Hole Count	Comp. Count	Mean	Median	Std. Dev.	CV	Min.	Max.
0	111	1205	0.02	0.01	0.02	1.35	0.00	0.20
100	122	1416	0.14	0.11	0.10	0.73	0.01	0.82
200	109	741	0.71	0.61	0.43	0.60	0.01	4.00
300	50	99	3.76	2.90	2.68	0.71	0.37	16.03
all	126	2256	0.46	0.20	0.91	1.98	0.01	16.03
Zn Composites by Domain								
Domain	Hole Count	Comp. Count	Mean	Median	Std. Dev.	CV	Min.	Max.
0	106	1073	0.02	0.01	0.02	0.97	0.00	0.20
100	118	1578	0.15	0.11	0.12	0.79	0.00	0.65
200	111	655	0.89	0.79	0.48	0.54	0.01	2.49
300	58	111	5.65	4.00	3.95	0.70	0.09	17.00
all	123	2344	0.56	0.20	1.35	2.39	0.00	17.00
Ag Composites by Domain								
Domain	Hole Count	Comp. Count	Mean	Median	Std. Dev.	CV	Min.	Max.
0	60	662	0.05	0.05	0.01	0.14	0.00	0.10
100	83	654	0.13	0.14	0.05	0.36	0.00	0.42
200	74	385	0.33	0.26	0.21	0.64	0.05	2.00
all	99	1039	0.20	0.17	0.16	0.81	0.00	2.00
Cu Ratio Composites by Domain and Oxidation Zone								
Oxide								
Domain	Hole Count	Comp. Count	Mean	Median	Std. Dev.	CV	Min.	Max.
0	58	200	0.76	0.97	0.26	0.34	0.25	1.00
100	79	419	0.79	0.81	0.18	0.22	0.00	1.00
200	64	247	0.86	0.91	0.17	0.20	0.02	1.00
300	19	33	0.92	0.96	0.15	0.17	0.10	1.00
all	90	699	0.81	0.87	0.18	0.22	0.00	1.00
Transition								
Domain	Hole Count	Comp. Count	Mean	Median	Std. Dev.	CV	Min.	Max.
0	77	453	0.67	0.50	0.27	0.41	0.00	1.00
100	99	589	0.58	0.57	0.24	0.41	0.00	1.00
200	70	331	0.52	0.51	0.30	0.58	0.00	1.00
300	22	42	0.45	0.39	0.34	0.75	0.00	0.98
all	104	962	0.55	0.54	0.27	0.48	0.00	1.00
Sulfide								
Domain	Hole Count	Comp. Count	Mean	Median	Std. Dev.	CV	Min.	Max.
0	65	522	0.65	0.50	0.32	0.49	0.00	1.00
100	57	373	0.32	0.20	0.28	0.88	0.00	1.00
200	42	156	0.17	0.10	0.21	1.18	0.01	0.96
300	16	24	0.12	0.08	0.16	1.37	0.00	0.96
all	62	553	0.27	0.17	0.27	0.99	0.00	1.00

14.8 Block Model Coding

The 200-foot-spaced cross-sectional mineral-domain polygons were used to code 20 x 20 x 20 (x, y, z)-foot blocks that comprise a digital model rotated to a bearing of 315°. The percentage volume of each mineral domain, as coded directly by the cross-sections, is stored within each block as a "partial percentage", as is the partial percentage of the block that lies outside of the modeled metal domains (domain 0). In other words, each block stores the partial percentage of each of the four domains for each modeled metal.

The Strong and Harris lithologic surfaces and solids were used to code each block to a single lithology on a 'majority wins' basis. The Strong and Harris digital topographic surface was used to code the block model on a partial percentage basis. The specific gravity values shown in Table 14.2 were assigned to the model blocks based on the copper mineral domain codes in each model block.

The mineralization has a variety of orientations. Wireframe solids were therefore created to encompass model areas with similar mineral domain orientations, and the solids were used to code the model blocks to these areas on a block-in/block-out basis. This coding was then used to control search-ellipse orientations during copper, zinc, and silver-grade interpolations. The orientations given in Table 14.12 were applied to all domains for each metal.

Table 14.12 Estimation Area Orientations

Area	Bearing	Plunge	Tilt
1	315	0	-35
2	315	0	0
3	315	0	-20

14.9 Grade Interpolation

Copper, zinc, and silver grades, as well as soluble copper ratios, were interpolated using inverse distance, ordinary kriging, and nearest-neighbor methods. The mineral resources reported herein were estimated by inverse distance interpolation as this method led to results that most appropriately respected the drill data and geology of the deposit. This is particularly true with respect to the estimation of the lowest-grade areas in the model, where potential overestimation of volumes could materially impact the resource estimation at grades close to potential open-pit mining cutoffs. The nearest-neighbor estimation was completed for the purposes of statistical checking of the various estimation iterations. The parameters applied to the grade estimations at Strong and Harris are summarized in Table 14.13**Error! Reference source not found.**.

Table 14.13 Estimation Parameters

Estimation Pass	Search Ranges (feet)			Composite Constraints		
	Major	Semi-Major	Minor	Min	Max	Max/Hole
Pass 1	650	650	325	2	15	3
Pass 2	1000	1000	500	1	15	3
Pass 3	1000	1000	1000	1	15	3

Grade interpolations were completed using 10-foot composites. The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded to that domain. Blocks coded as having partial percentages of more than one domain had multiple grade interpolations, one for each domain coded into the block. The estimated grades for each of the metal domains 0, 100, 200, and 300 coded to a block were coupled with the coded partial percentages of those domains to enable the calculation of a single volume-weighted grade of each of the metal species for each block. These resource block grades are therefore diluted to the full block volumes using this methodology.

14.10 Mineral Resources

The Strong and Harris project mineral resources have been estimated to reflect potential open-pit extraction and potential processing by heap leaching or concentration (depending on oxidation zone of the mineralization). To meet the requirement of the resources having reasonable prospects for eventual economic extraction, a pit optimization was completed using the parameters summarized in Table 14.14.

Table 14.14 Pit Optimization Parameters

Parameter	Value	Unit
Mining	$ 2.00	$/ton Mined
Processing - Leaching	$ 5.00	$/ton Processed
Processing - Floatation	$ 9.00	$/ton Processed
Processing Rate	7,200	1,000s tons-per-year
G&A Cost per Ton	$ 0.83	$/ton Processed
Cu Price	$ 3.50	$/pound produced
Cu Refining Cost	$ 0.08	$/pound produced
Royalty	3%	NSR
Metallurgical Recoveries	**Value**	**Unit**
Copper Recovery Heap Leach Oxide and Transition Mineralization	92.3%	% Rec of CuOx
Copper Recovery Heap Leach Oxide and Transition Mineralization	82.3%	% Rec of Zn
Copper Recovery Concentrator Transition Mineralization	80.1%	% Rec of Cu
Zinc Recovery Concentrator Transition Mineralization	69.7%	% Rec of Zn
Copper Recovery Concentrator Sulfide Mineralization	84.0%	% Rec of Cu
Zinc Recovery Concentrator Sulfide Mineralization	89.0%	% Rec of Zn

The pit shells created using these optimization parameters were used to constrain the project resources. The in-pit resources were further constrained by the application of a cutoff of 0.1% Cu to all model blocks within the optimized pits.

The Strong and Harris project resources are summarized in Table 14.15. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.15 Strong and Harris Mineral Resources

(0.1% Cu cutoff)

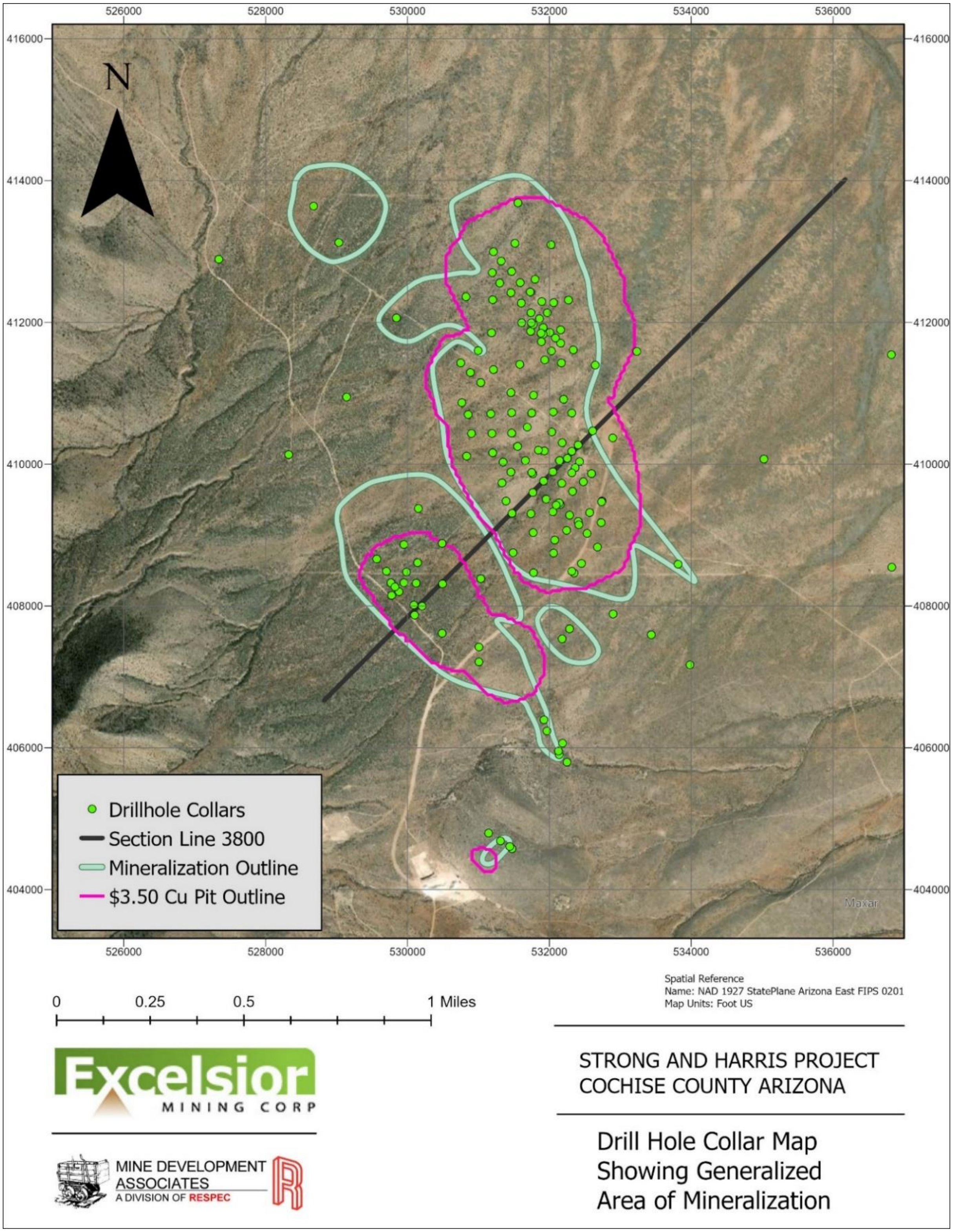
Classification	Tons	% Cu	% CuOx	% Zn	oz Ag/ton	lbs Cu	lbs CuOx	lbs Zn	oz Ag
Inferred	76,161,000	0.52	0.33	0.56	0.12	794,049,000	500,155,000	858,425,000	9,515,000

1. The Effective Date of the mineral resources is September 9, 2021.
2. The project mineral resources are shown in bold and are comprised of all model blocks at a 0.1 % Cu cutoff that lie within optimized resource pits.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. The estimate of mineral resources may be materially affected by geology, environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
5. Rounding as required by reporting guidelines may result in apparent discrepancies between tons, grade, and contained metal content.

The Strong and Harris mineral resources are entirely classified as Inferred. This classification is based on the confidence in the underlying data which are largely historical. Excelsior's 2021 sampling program verified the historical data sufficiently to warrant the Inferred classification, but additional drilling and sampling, as well as more detailed geological modeling, would be required to allow for higher classification of the project resources.

The Strong and Harris in-pit resources cover an aerial extent of over one mile along strike. Figure 14.4 shows the surface projections of the pit shells resulting from the resource-constraining pit optimization in the context of the deposit with section line 3800. Figure 14.5, Figure 14.6, Figure 14.7, and Figure 14.8 are representative cross sections through the block model along section line 3800.

Figure 14.4 Plan Map of Strong and Harris Drilling, Mineralization and $3.50/lb Cu Pit Shells
(September, 2021)



Note: see Figure 10.1 for the location of the resource foot prints relative to the property outline.

Figure 14.5 Geologic Cross Section with Copper Block Model Grades

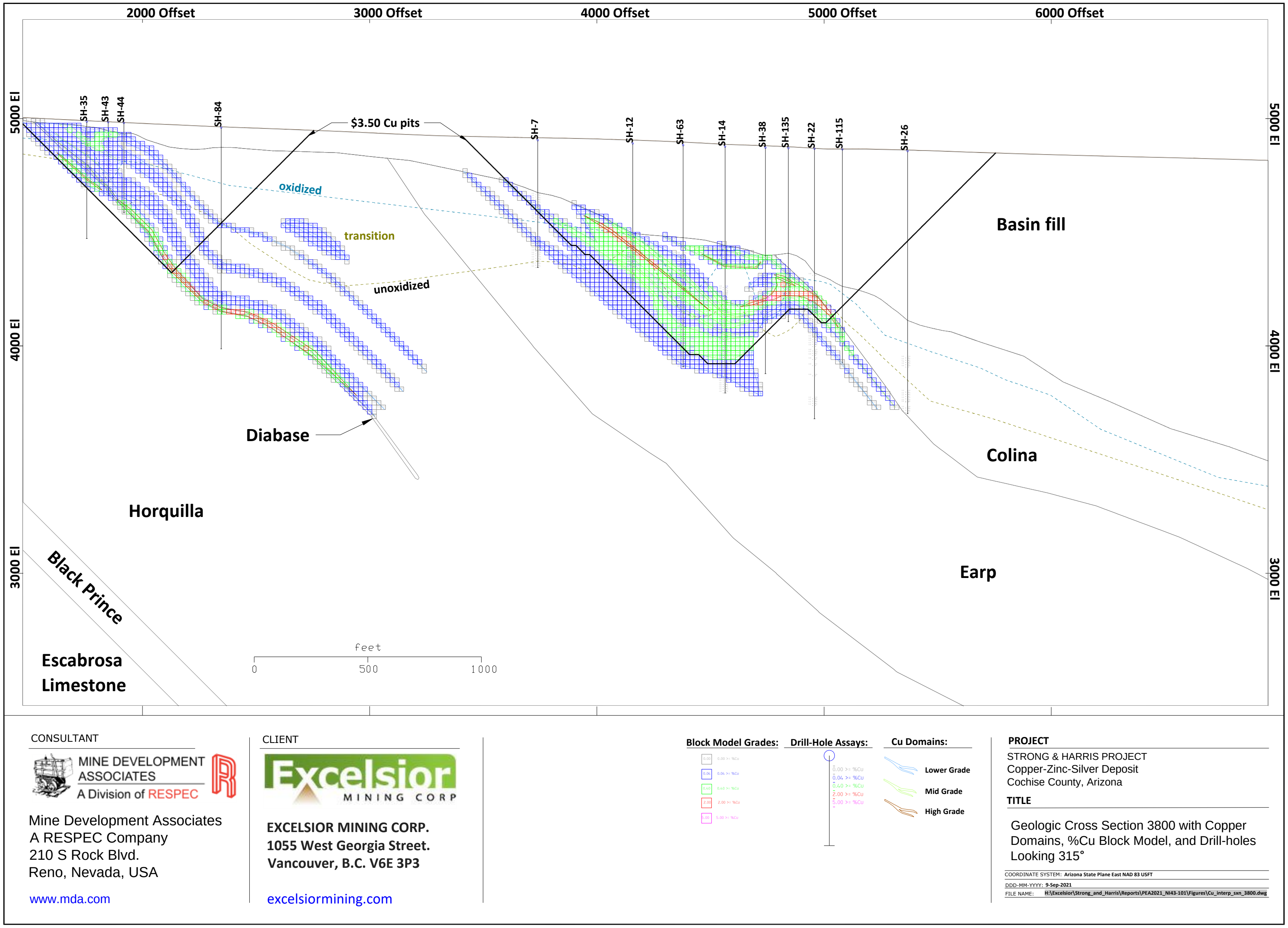


Figure 14.6 Geologic Cross Section with Soluble Copper Block Model Grades

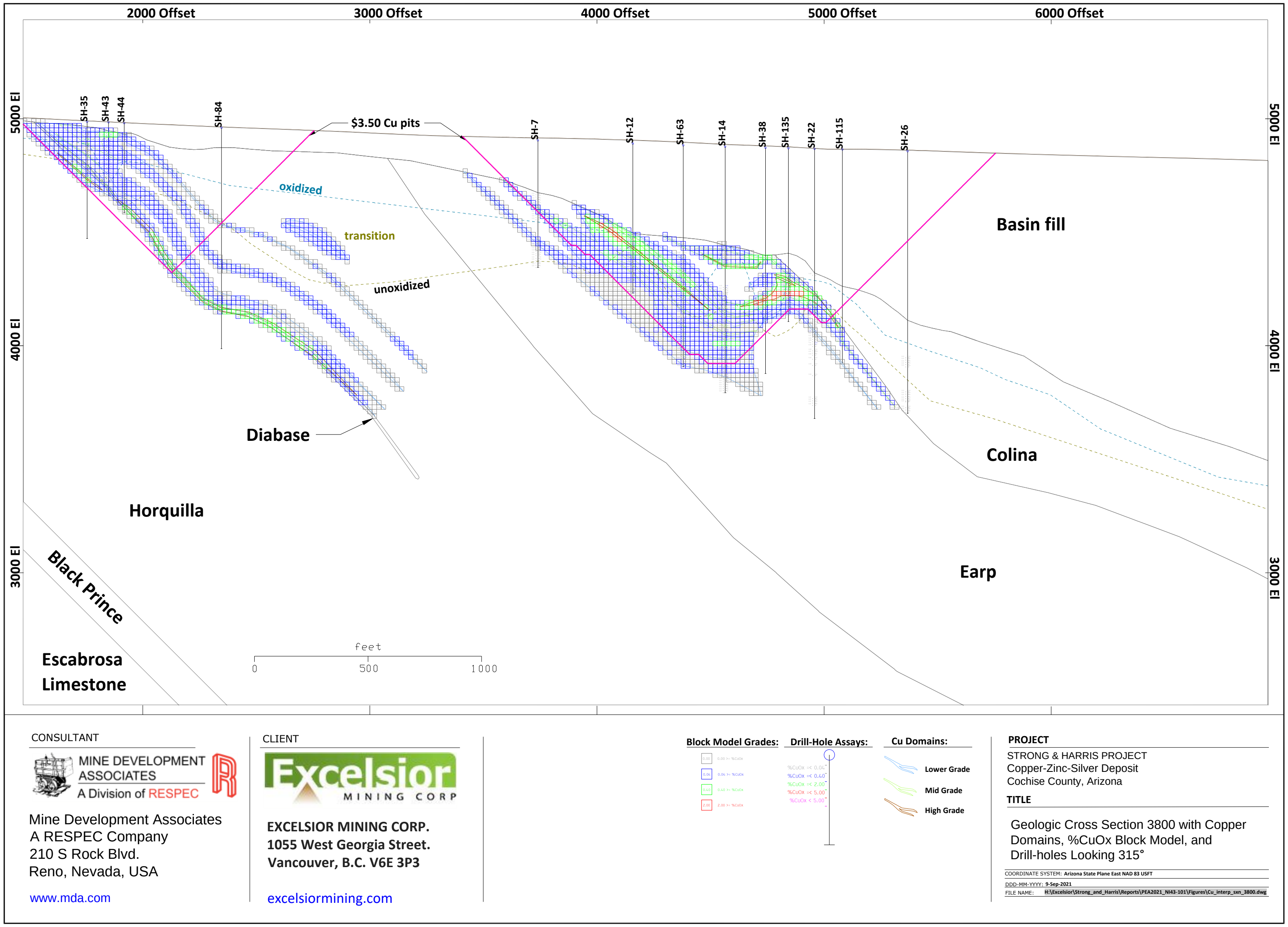


Figure 14.7 Geologic Cross Section with Zinc Block Model Grades

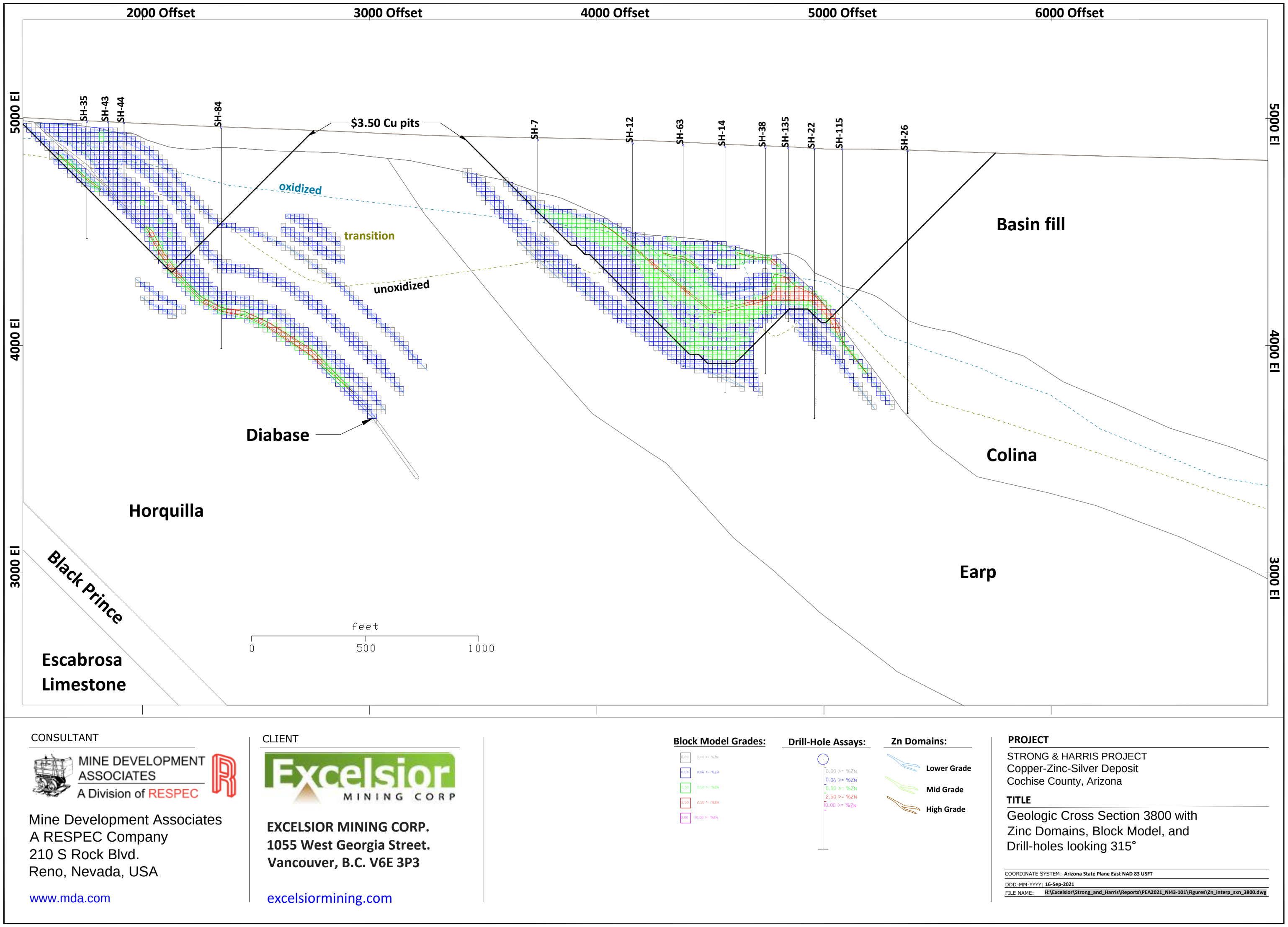
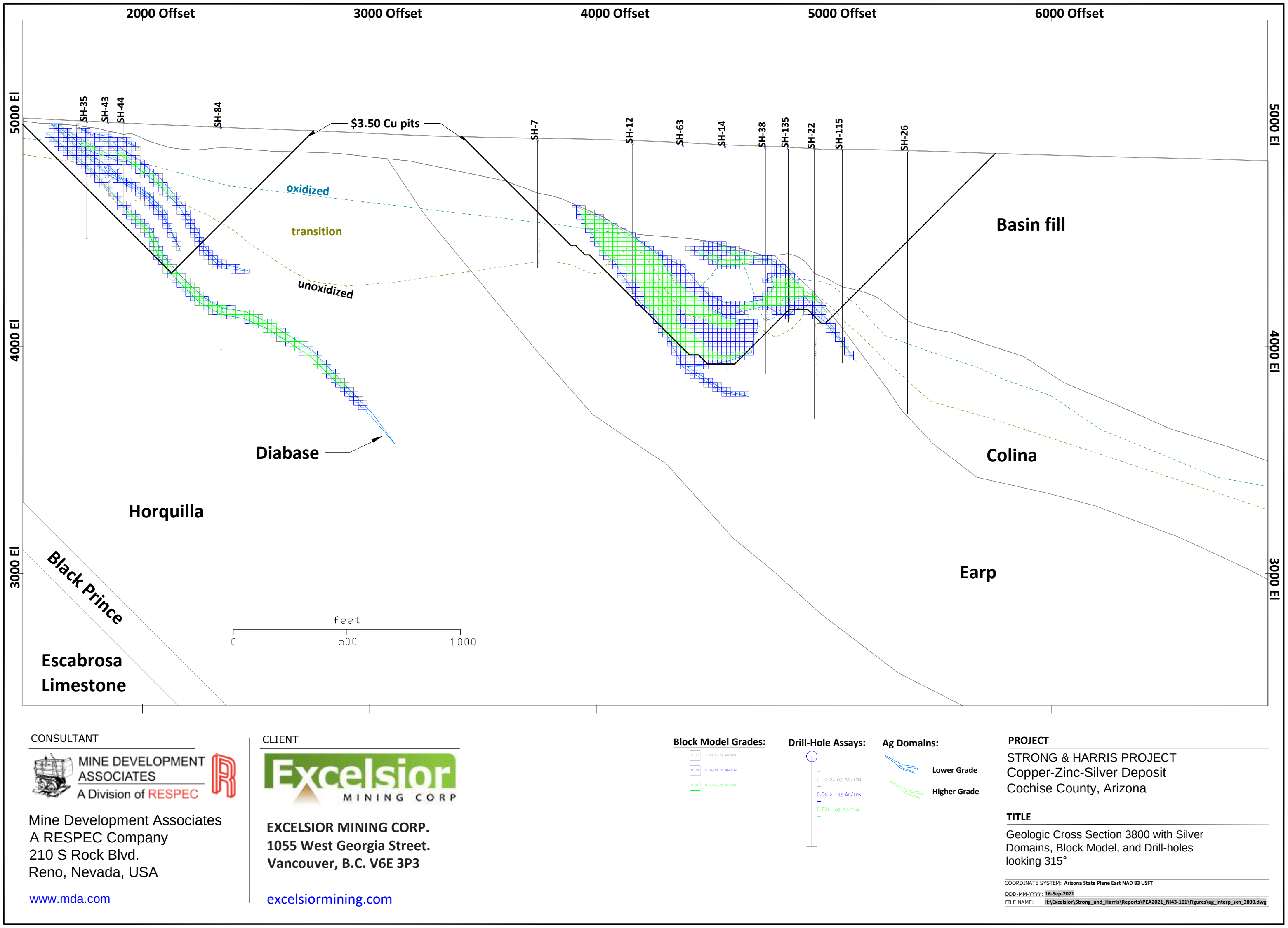


Figure 14.8 Geologic Cross Section with Silver Block Model Grades



The Strong and Harris in-pit resources are categorized geological variables in the model including oxidation zones and lithology. The in-pit resources are broken-down by oxidation zone in Table 14.16 and by lithology in Table 14.17.

Table 14.16 Strong and Harris Pit-Constrained Resources by Oxidation Zone
(0.1% Cu cutoff)

Oxidation Zone	Tons	% Cu	% CuOx	% Zn	oz Ag/ton	lbs Cu	lbs CuOx	lbs Zn	oz Ag
Oxide	30,517,000	0.52	0.44	0.56	0.11	317,503,000	269,354,000	343,129,000	3,353,000
Transition	33,057,000	0.52	0.29	0.60	0.13	344,362,000	189,932,000	395,842,000	4,291,000
Sulfide	12,587,000	0.53	0.16	0.47	0.15	132,184,000	40,868,000	119,454,000	1,871,000

Table 14.17 Strong and Harris Pit-Constrained Resources by Lithology
(0.1% Cu cutoff)

Lithology	Tons	% Cu	% CuOx	% Zn	oz Ag/ton	lbs Cu	lbs CuOx	lbs Zn	oz Ag
Horquilla	11,201,000	0.37	0.21	0.25	0.09	82,598,000	46,871,000	57,073,000	961,000
Earp	61,338,000	0.53	0.33	0.61	0.13	653,560,000	409,000,000	753,850,000	8,267,000
Colina	3,087,000	0.82	0.66	0.69	0.06	50,642,000	40,851,000	42,860,000	198,000
Diabase	535,000	0.68	0.32	0.43	0.17	7,249,000	3,433,000	4,641,000	89,000

Table 14.18 presents the Strong and Harris mineral resources compared to subsets of mineralized material tabulated with increasing cutoff grades. This is presented to provide grade-distribution data that allows for detailed assessment of the project resources. All of the tabulations are constrained as lying within the same optimized pit shells used to constrain the current mineral resources, which means the tabulations at cutoffs higher than the resource cutoff grade of 0.1% Cu represent subsets of the current resources.

Table 14.18 Strong and Harris Pit-Constrained Resources at Various Cutoffs

% Cu Cutoff	Tons	% Cu	% CuOx	% Zn	oz Ag/ton	lbs Cu	lbs CuOx	lbs Zn	oz Ag
0.1	***76,161,000***	***0.52***	***0.33***	***0.56***	***0.12***	***794,049,000***	***500,155,000***	***858,425,000***	***9,515,000***
0.2	54,187,000	0.67	0.42	0.70	0.15	731,493,000	458,808,000	757,677,000	7,900,000
0.4	34,848,000	0.90	0.56	0.87	0.17	624,078,000	390,701,000	605,666,000	5,768,000
0.6	22,176,000	1.12	0.71	1.05	0.18	498,599,000	314,910,000	463,692,000	4,050,000
0.8	12,280,000	1.48	0.94	1.35	0.20	362,913,000	231,657,000	330,633,000	2,455,000
1	7,077,000	1.91	1.25	1.77	0.23	271,046,000	176,599,000	250,717,000	1,645,000

1. The project mineral resources are shown in bold and are comprised of all model blocks at a 0.1% Cu cutoff that lie within optimized resource pits.
2. Tabulations at higher cutoffs than used to define the mineral resources represent subsets of the mineral resource.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. Rounding as required by reporting guidelines may result in apparent discrepancies between tons, grade, and contained metal content.

14.11 Discussion of Resources and Recommendations

The Strong and Harris resource estimate was done through a sectional extrusion of the mineral domain polygons discussed in this report. The resource block model consequently appears jagged when viewed in three dimensions. While this modeling strategy is unlikely to materially affect the overall estimation of project resources, three-dimensional rectification of the cross-sectional mineral domain polygons would improve spatial precision of the grade estimates and would be required for future resource estimations that included classifications above the Inferred level.

Future drilling, exploration, and resource definition at Strong and Harris should focus on increasing the understanding of geologic controls on mineralization, infill drilling in key areas to increase drill density, and drill-testing of the unconstrained limits of the deposit. Despite well-understood lithological controls on mineralization, limited data regarding the deposit structure are available. Drilling angle holes to test structures is recommended for this purpose. The authors also recommend testing the mineralization limits along strike, particularly on the northern edge of known mineralization, as well as testing the mineralization limits down-dip in favorable host-rocks.

Downhole surveys of historical drill holes, where possible, should be collected to improve spatial confidence. Mineralogical characterization of the copper, zinc, and silver, is also recommended using common analytical techniques such as x-ray diffraction, scanning electron microscopy, or otherwise, especially in the context of the deposit's oxidation zones.

As the date of this report, Mr. Bickel and Mr. Gustin are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Strong and Harris mineral resources and that are not otherwise discussed in this report. The impact of taxation was taken into consideration when establishing cut-off grade and further details are provided in Section 22: Economic Analysis.

15.0 MINERAL RESERVE ESTIMATES (ITEM 15)

There are no current mineral reserves estimated for the Strong and Harris project.

16.0 MINING METHODS (ITEM 16)

The PEA presented in this report considers open-pit mining of the Strong and Harris project. Note that a PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the economic results of the PEA will be realized.

The methodology used for mine planning to define the economics for the PEA includes:

- Define assumptions for the economic parameters;
- Define geometric parameters and constraints;
- Run pit optimizations;
- Define road and ramp parameters;
- Create pit designs;
- Create dump designs;
- Produce mine and process production schedules;
- Define personnel and equipment requirements;
- Estimate mining costs; and
- Perform an economic analysis.

Section 16 summarizes the above topics, except for the mining cost estimates which are discussed in Section 21, and the economic analysis discussed in Section 22.

16.1 Economic Parameters

The economic parameters used have been developed between Excelsior, Mr. Bowell, and MDA. Table 16.1 shows the economic parameters used for mining, leaching, flotation, and general and administrative (“G&A”) costs. Mining costs assume contract mining to reduce the capital costs for the project. Leaching and flotation costs are based on inputs from the Mr. Bowell. Leaching costs are based on a $110/ton acid price. G&A costs are based on a fixed cost of $6 million per year for site management, environmental, general site maintenance, insurance, human resources, etc. Note that the final G&A costs were reduced significantly as they consider shared personnel and resources for management of Strong and Harris mining and processing.

Royalties were applied as a Net Smelter Return (“NSR”). For most of the property the NSR royalty is 3.0% with the exception of land in Sections 23 and 24, where the rate is 17.785%.

Table 16.1 Economic Parameters

Mining	$ 2.50	$/ton Mined
Processing - Leaching	$ 5.62	$/ton Processed
Processing - Floatation	$ 11.70	$/ton Processed
Tons per Year	7,200	K TPY
G&A per Year	$ 6,000	K USD
G&A Cost per Ton	$ 0.83	$/ton Processed
Royalty	By Area	NSR

Metallurgical recoveries are shown in Table 16.2 and were provided by Mr. Bowell. These are broken out by oxidation levels. Oxide recoveries were applied to the soluble copper estimates in the model yielding a relatively high apparent recovery, though the actual total copper recovery is lower as a function of using the soluble value. All oxide material is assumed to be leached while sulfide and mixed material would be sent through a flotation plant producing a concentrate.

The assumed concentrate grades are given for reference but were not directly used in the economics.

Table 16.2 Metal Recoveries and Assumed Concentrate Grades

CuOx Recovery - Heap Leach	92.3%	% Rec CuSol in Leach
Zn Recovery - Heap Leach	82.3%	% Rec Zn in Leach
Cu Float Recovery - Sulfide	84.0%	% Recovered into Concentrate
Zn Float Recovery - Sulfide	89.0%	% Recovered into Concentrate
Cu Con Grade - Sulfide	9.4%	%
Zn Con Grade - Sulfide	8.2%	%
Cu Float Recovery - Mixed	80.1%	% Recovered into Concentrate
Zn Float Recovery - Mixed	69.7%	% Recovered into Concentrate
Cu Con Grade - Mixed	11.8%	%
Zn Con Grade - Mixed	11.2%	%

The concentrate cost assumptions are shown in Table 16.3. The mass pull represents the mass of concentrate as a function of the mass of material fed into the flotation circuit. Thus, for every 100 tons fed into the flotation circuit, 4.0 tons of concentrate would be created. The costs per ton of concentrate were multiplied by the mass pull percentage yielding a cost per ton of total material processed.

Table 16.3 Concentrate Cost Assumptions

Mass Pull - Sulfide	4%	% of Ton Feed
Silver Grade in CuCon	4.18	Oz/Ton
Transport Cost/ton	$ 140	$/ton Concentrate
Transport Cost/ton	$ 5.60	$/ton Processed
Treatment Cost/Ton	$ 76	$/ton Concentrate
Treatment Cost/Ton	$ 3.04	$/ton Processed
Zn Penalty Charge	$ 6.00	$/ton Concentrate @ 7% Zn
Zn Penalty Charge	$ 0.24	$/ton Processed

Pit slope parameters were assumed to use a 45° overall slope in all sectors and directions. The validity of this assumption will need to be determined in future studies.

16.2 Cutoff Grades

Cutoff grades were used to distinguish material that would be processed and waste material. An NSR cutoff grade was used for this selection and considered oxide and mixed resources that might be processed as leach material, and mixed and sulfide material that might be processed using flotation. The NSR was calculated into the resource model to determine the value of each block based on the grades of copper and zinc along with the economic parameters as described in Section 16.1.

The basic NSR equation is as follows:

$$NSR_{Cu} = Cu_{grd} * Cu_{rec} * Cu_{price} * (1 - roy)$$
$$NSR_{Zn} = Zn_{grd} * Zn_{rec} * Zn_{price} * (1 - roy)$$

For mixed material, the NSR was calculated for both leach and flotation and then compared to determine the most profitable method for processing the material. Note that the NSR equations do not include the processing cost. The total of the processing cost along with G&A costs were determined to be the cutoff used. Thus, for leaching the block had to have a value better than $5.62 + $0.83 or $6.45/ton. For flotation, the value had to be greater than the processing and offsite costs plus G&A. The offsite or concentrate costs were determined in terms of $/ton processed by applying the mass pull of 4%. The flotation NSR cutoff grade used is $11.70 + $0.83 + $5.60 + $3.04 + $0.24 = $21.41/ton.

16.3 Pit Optimization

Pit optimizations used the Whittle Software's (version 7.2) Lerchs-Grossman algorithm to create 3-dimensional pit shells. The base metal prices used for this analysis were $3.50/pound of copper and $1.28 per pound of zinc. Although silver is estimated in the resource block model, it was not given any value. Pit optimizations were generated using ranges of copper prices from $1.00 to $5.00 per pound in increments of $0.05 per pound. Each pit used a constant ratio between the zinc and copper price. The pit optimization results are shown in Table 16.4 from $1.5/lb to $5.0/lb copper prices in $0.25 increments.

Table 16.4 Whittle Pit Optimization Results

			Material Processed									Waste	Total	Strip
Pit	Cu Price	Zn Price	K Tons	% Cu	K lbs. Cu	% CuOx	K lbs. CuOx	% Zn	K lbs. Zn	Ag oz./T	K Ag oz.	K Tons	K Tons	Ratio
1	$ 1.50	$ 0.55	17,138	0.78	265,846	0.51	176,097	0.96	329,438	0.16	2,778	93,458	110,596	5.45
6	$ 1.75	$ 0.64	25,306	0.71	359,737	0.46	230,909	0.89	449,139	0.15	3,892	118,315	143,621	4.68
11	$ 2.00	$ 0.73	29,465	0.67	396,428	0.42	250,116	0.83	488,616	0.15	4,424	125,008	154,473	4.24
16	$ 2.25	$ 0.83	33,224	0.64	424,682	0.40	265,376	0.79	524,003	0.15	4,821	132,159	165,383	3.98
21	$ 2.50	$ 0.92	37,519	0.61	454,762	0.38	283,001	0.74	557,555	0.14	5,221	140,662	178,181	3.75
26	$ 2.75	$ 1.01	40,217	0.59	471,844	0.36	293,522	0.71	573,371	0.14	5,517	145,185	185,402	3.61
31	$ 3.00	$ 1.10	48,795	0.59	576,027	0.38	375,466	0.72	702,974	0.14	6,792	243,072	291,867	4.98
36	$ 3.25	$ 1.19	51,471	0.58	592,454	0.37	385,840	0.70	720,090	0.14	7,077	250,595	302,065	4.87
41	$ 3.50	$ 1.28	54,422	0.56	613,438	0.37	398,332	0.68	735,811	0.13	7,321	259,734	314,156	4.77
46	$ 3.75	$ 1.38	62,524	0.53	664,819	0.35	433,564	0.62	778,675	0.13	8,193	295,176	357,701	4.72
51	$ 4.00	$ 1.47	71,010	0.52	745,567	0.34	478,693	0.61	864,705	0.13	8,883	385,540	456,550	5.43
56	$ 4.25	$ 1.56	77,506	0.50	779,187	0.32	498,150	0.59	907,580	0.12	9,275	414,705	492,211	5.35
61	$ 4.50	$ 1.65	84,552	0.49	826,202	0.31	521,394	0.57	956,508	0.12	9,926	455,095	539,647	5.38
66	$ 4.75	$ 1.74	88,100	0.48	839,357	0.30	528,789	0.55	971,605	0.11	10,103	462,681	550,781	5.25
71	$ 5.00	$ 1.83	92,713	0.47	863,230	0.29	540,639	0.54	995,861	0.11	10,407	484,046	576,759	5.22

16.4 Pit by Pit Analysis

Whittle PbP analysis was completed for this study to determine simple pushbacks and the ultimate pit to be used for production scheduling.

The Pit by Pit ("PbP") analysis tool is used to generate discounted operating cash flows (note that capital is not included). The PbP node uses a constant metal price and a rough scheduling by pit phase for each pit shell to generate the discounted value for the pit. The program develops three different discounted values: best, worst, and specified. The best-case value uses each of the pit shells as pit phases or pushbacks. For example, when evaluating pit 20, there would be 19 pushbacks mined prior to pit 20, and the resulting schedule takes advantage of mining more valuable material up front to improve the discounted value. Evaluating pit 21 would have 20 pushbacks; pit 22 would have 21 pushbacks and so on. Note that this is not a realistic case as the incremental pushbacks would not have enough mining width between them to be able to mine appropriately, but this does help to define the maximum potential discounted operating cash flow.

The worst case does not use any pushbacks in determining the discounted value for each of the pit shells. Thus, each pit shell is evaluated as if mining a single pit from top to bottom. This does not provide the advantage of mining more valuable material first, so it generally provides a lower discounted value than that of the best case.

The specified case allows the user to specify pit shells to be used as pushbacks and then schedules the pushbacks and calculates the discounted cash flow. This is more realistic than the base case as it allows for more mining width, though the final pit design will have to ensure that appropriate mining width is available. The specified case has been used to determine the ultimate pit limits to be used for the economic analysis, as well as to specify guidelines for any subsequent pit designs.

PbP analysis was developed for the base case using $3.50 per pound copper and $1.10 per pound zinc prices along with the economic parameters shown in Table 16.1. The Base Case PbP results are shown in Table 16.5. The highlighted pit number 41 shows the pit with the highest specified pit value using the $3.50 per pound copper price. The results are also shown graphically in Figure 16.1.

Figure 16.1 Strong and Harris Pit by Pit Graph

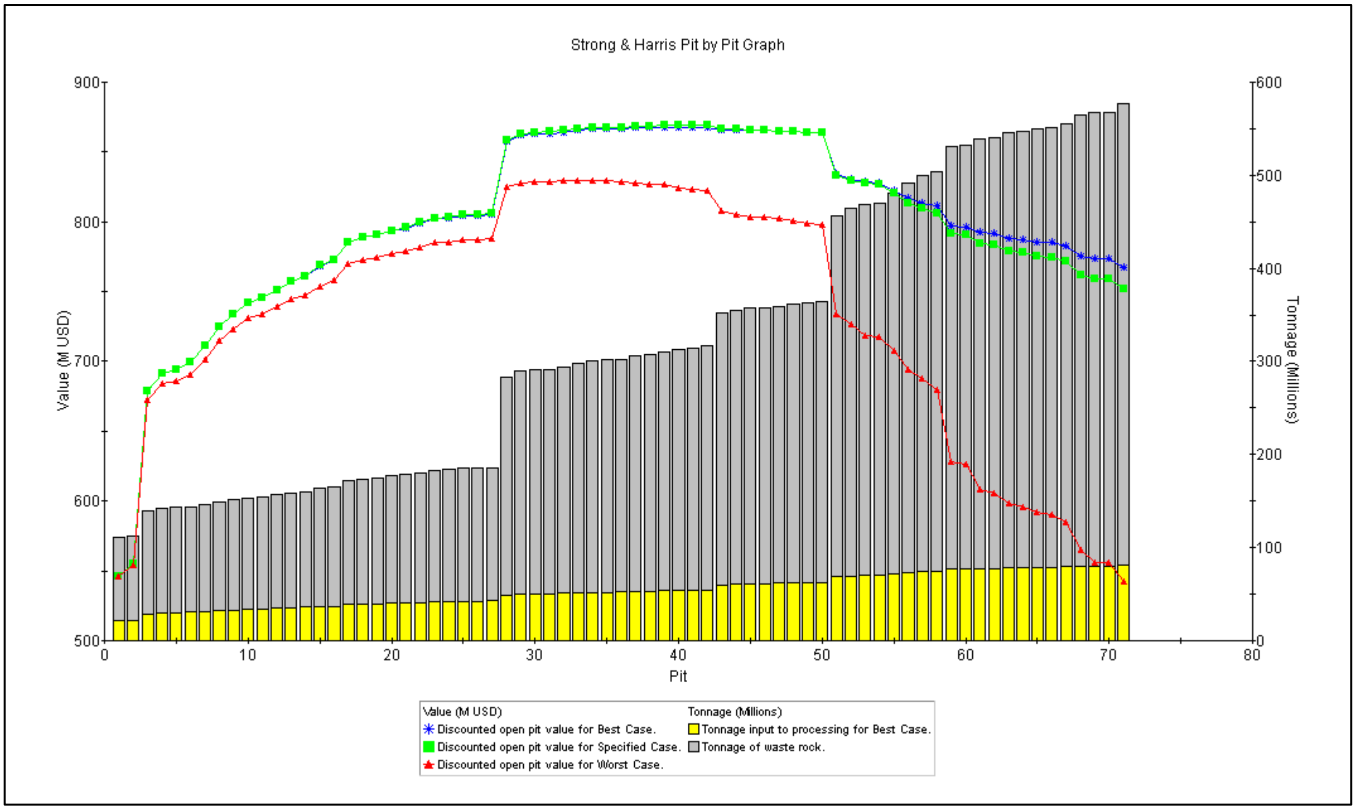


Table 16.5 Strong and Harris PbP Results

	Leach Material Processed							Floatation Material Processed							Total Material Processed							Waste	Total	Strip	Disc. Op Cash Flow (M USD)		
Pit	K Tons	% Cu	K lbs Cu	% Zn	K lbs. Zn	oz Ag/ton	K Ozs Ag	K Tons	% Cu	K lbs Cu	% Zn	K lbs. Zn	oz Ag/ton	K Ozs Ag	K Tons	% Cu	K lbs. Cu	% Zn	K lbs. Zn	oz Ag/ton	K Ozs Ag	K Tons	K Tons	Ratio	Best	Specified	Worst
1	18,314	0.54	198,083	0.68	249,251	0.131	2,408	3612.684	1.20	86,347	1.40	101,115	0.218	786	21,926	0.65	284,431	0.80	350,366	0.146	3,194	88,670	110,596	4.04	$ 546.01	$ 546.01	$ 546.01
2	18,544	0.54	200,697	0.69	255,057	0.132	2,447	3682.005	1.19	87,834	1.39	102,612	0.218	803	22,226	0.65	288,530	0.80	357,670	0.146	3,250	90,360	112,586	4.07	$ 554.71	$ 554.71	$ 554.34
3	24,025	0.52	250,966	0.68	325,774	0.131	3,146	5046.318	1.12	112,576	1.28	128,849	0.204	1,030	29,072	0.63	363,542	0.78	454,623	0.144	4,176	110,196	139,268	3.79	$ 678.87	$ 678.87	$ 672.11
4	24,589	0.52	256,948	0.68	333,045	0.130	3,207	5185.14	1.11	114,865	1.27	131,380	0.203	1,055	29,774	0.62	371,814	0.78	464,425	0.143	4,261	112,716	142,490	3.79	$ 691.79	$ 691.79	$ 684.05
5	24,725	0.52	257,916	0.68	334,135	0.130	3,225	5264.72	1.10	116,019	1.26	132,321	0.203	1,068	29,990	0.62	373,935	0.78	466,456	0.143	4,293	112,955	142,945	3.77	$ 694.48	$ 694.48	$ 685.97
6	24,950	0.52	259,347	0.67	335,703	0.131	3,256	5526.033	1.08	118,882	1.22	135,053	0.202	1,117	30,476	0.62	378,229	0.77	470,757	0.143	4,373	113,145	143,621	3.71	$ 699.72	$ 699.72	$ 690.37
7	25,505	0.52	263,553	0.67	340,473	0.130	3,318	5882.167	1.05	123,309	1.19	139,450	0.201	1,181	31,387	0.62	386,862	0.76	479,922	0.143	4,499	114,563	145,950	3.65	$ 711.57	$ 711.61	$ 701.64
8	26,186	0.51	269,335	0.66	346,658	0.129	3,387	6237.161	1.02	127,664	1.15	143,480	0.200	1,250	32,423	0.61	396,999	0.76	490,138	0.143	4,637	116,723	149,146	3.60	$ 724.89	$ 724.99	$ 714.49
9	26,521	0.51	271,661	0.66	350,124	0.129	3,433	6497.221	1.01	131,799	1.14	148,200	0.201	1,303	33,018	0.61	403,460	0.75	498,324	0.143	4,736	118,554	151,572	3.59	$ 733.75	$ 733.89	$ 722.71
10	27,306	0.51	277,116	0.65	355,194	0.129	3,515	6673.932	1.00	133,787	1.13	150,218	0.200	1,335	33,980	0.60	410,903	0.74	505,412	0.143	4,850	119,615	153,595	3.52	$ 742.15	$ 742.31	$ 730.86
11	27,490	0.51	278,509	0.65	356,853	0.128	3,532	6771.515	1.00	134,936	1.12	151,477	0.200	1,352	34,262	0.60	413,445	0.74	508,330	0.143	4,884	120,211	154,473	3.51	$ 745.27	$ 745.44	$ 733.46
12	27,917	0.50	281,725	0.65	360,431	0.128	3,572	6881.501	0.99	136,663	1.11	153,302	0.199	1,371	34,799	0.60	418,388	0.74	513,733	0.142	4,943	121,786	156,585	3.50	$ 751.10	$ 751.29	$ 738.98
13	28,442	0.50	285,455	0.64	364,970	0.127	3,619	7092.957	0.98	139,099	1.10	155,775	0.199	1,409	35,535	0.60	424,554	0.73	520,745	0.141	5,027	123,062	158,597	3.46	$ 757.56	$ 757.77	$ 744.52
14	28,767	0.50	287,653	0.64	367,363	0.127	3,648	7190.884	0.98	140,246	1.09	156,923	0.198	1,426	35,958	0.60	427,899	0.73	524,286	0.141	5,074	123,994	159,952	3.45	$ 761.09	$ 761.31	$ 747.75
15	29,248	0.50	291,198	0.64	372,507	0.126	3,690	7361.373	0.97	142,654	1.10	161,571	0.197	1,448	36,609	0.59	433,852	0.73	534,078	0.140	5,138	126,814	163,424	3.46	$ 768.41	$ 768.67	$ 754.04
16	29,732	0.50	294,549	0.63	376,841	0.125	3,729	7410.364	0.97	143,405	1.10	162,330	0.196	1,454	37,142	0.59	437,954	0.73	539,171	0.140	5,183	128,240	165,383	3.45	$ 772.45	$ 772.73	$ 757.86
17	31,294	0.49	306,159	0.62	390,485	0.123	3,857	7642.392	0.96	146,141	1.08	164,626	0.194	1,486	38,936	0.58	452,300	0.71	555,111	0.137	5,343	133,247	172,183	3.42	$ 785.29	$ 785.72	$ 769.68
18	31,666	0.49	308,762	0.62	393,605	0.123	3,890	7748.251	0.95	147,336	1.07	165,601	0.194	1,506	39,415	0.58	456,098	0.71	559,206	0.137	5,396	134,492	173,906	3.41	$ 788.62	$ 789.13	$ 772.91
19	31,863	0.49	310,200	0.62	395,150	0.122	3,903	7802.493	0.95	148,145	1.07	166,663	0.194	1,512	39,666	0.58	458,345	0.71	561,813	0.137	5,414	135,411	175,077	3.41	$ 790.51	$ 791.06	$ 774.52
20	32,199	0.49	312,421	0.62	397,295	0.122	3,931	7869.536	0.95	149,035	1.07	167,903	0.193	1,520	40,069	0.58	461,457	0.71	565,198	0.136	5,451	136,696	176,765	3.41	$ 793.14	$ 793.74	$ 776.95
21	32,629	0.48	314,986	0.61	400,062	0.122	3,967	7948.355	0.94	149,775	1.06	168,751	0.193	1,533	40,577	0.57	464,760	0.70	568,812	0.136	5,500	137,604	178,181	3.39	$ 795.64	$ 796.27	$ 779.26
22	33,270	0.48	318,707	0.61	403,281	0.122	4,061	7996.596	0.95	151,302	1.06	169,193	0.193	1,542	41,267	0.57	470,009	0.69	572,474	0.136	5,603	139,237	180,504	3.37	$ 798.79	$ 799.46	$ 782.10
23	33,813	0.48	322,464	0.60	407,236	0.122	4,115	8094.042	0.94	152,280	1.05	170,278	0.193	1,559	41,907	0.57	474,744	0.69	577,514	0.135	5,674	141,353	183,260	3.37	$ 802.19	$ 802.92	$ 785.23
24	33,908	0.48	323,041	0.60	407,977	0.122	4,120	8123.312	0.94	152,556	1.05	170,508	0.192	1,562	42,032	0.57	475,597	0.69	578,485	0.135	5,682	141,636	183,667	3.37	$ 802.72	$ 803.47	$ 785.71
25	34,248	0.47	325,289	0.60	410,003	0.121	4,157	8162.058	0.94	152,936	1.05	170,870	0.192	1,570	42,410	0.56	478,226	0.68	580,873	0.135	5,726	142,696	185,105	3.36	$ 804.27	$ 805.06	$ 787.29
26	34,299	0.47	325,581	0.60	410,574	0.121	4,160	8170.087	0.94	153,070	1.05	170,969	0.192	1,571	42,469	0.56	478,651	0.68	581,542	0.135	5,730	142,933	185,402	3.37	$ 804.54	$ 805.33	$ 787.50
27	34,523	0.47	326,855	0.60	411,768	0.121	4,179	8190.755	0.94	153,296	1.04	171,127	0.192	1,574	42,714	0.56	480,151	0.68	582,894	0.135	5,754	143,435	186,149	3.36	$ 805.27	$ 806.07	$ 788.30
28	40,485	0.50	407,734	0.63	511,018	0.127	5,138	8497.033	0.95	161,024	1.08	183,928	0.193	1,643	48,982	0.58	568,759	0.71	694,947	0.138	6,781	233,716	282,697	4.77	$ 857.88	$ 858.87	$ 825.12
29	41,487	0.50	414,096	0.63	519,385	0.126	5,241	8634.44	0.95	164,012	1.08	186,169	0.193	1,670	50,121	0.58	578,107	0.70	705,554	0.138	6,910	239,879	290,000	4.79	$ 862.18	$ 863.24	$ 827.88
30	41,837	0.50	416,030	0.62	521,890	0.126	5,271	8679.749	0.95	164,509	1.08	186,656	0.193	1,678	50,517	0.57	580,539	0.70	708,546	0.138	6,949	241,147	291,664	4.77	$ 863.17	$ 864.26	$ 828.67
31	41,917	0.50	416,425	0.62	522,278	0.126	5,279	8690.387	0.95	164,604	1.07	186,751	0.193	1,679	50,608	0.57	581,028	0.70	709,029	0.137	6,958	241,259	291,867	4.77	$ 863.31	$ 864.40	$ 828.73
32	42,339	0.49	418,877	0.62	524,946	0.126	5,317	8703.748	0.95	164,763	1.07	186,876	0.193	1,680	51,043	0.57	583,640	0.70	711,823	0.137	6,997	242,688	293,732	4.75	$ 864.21	$ 865.31	$ 829.54
33	42,814	0.49	423,638	0.62	529,523	0.125	5,370	8769.983	0.94	165,683	1.07	187,567	0.193	1,693	51,584	0.57	589,321	0.70	717,089	0.137	7,063	246,161	297,745	4.77	$ 865.59	$ 866.73	$ 829.46
34	43,088	0.49	426,138	0.62	532,604	0.126	5,411	8806.86	0.94	166,214	1.07	188,040	0.193	1,702	51,895	0.57	592,352	0.69	720,644	0.137	7,112	248,870	300,765	4.80	$ 866.44	$ 867.61	$ 829.47
35	43,250	0.49	427,108	0.62	533,375	0.125	5,422	8844.069	0.94	166,609	1.06	188,227	0.193	1,707	52,094	0.57	593,717	0.69	721,602	0.137	7,129	249,337	301,431	4.79	$ 866.61	$ 867.80	$ 829.24
36	43,376	0.49	427,798	0.62	534,208	0.125	5,433	8873.329	0.94	166,894	1.06	188,399	0.193	1,712	52,250	0.57	594,692	0.69	722,607	0.137	7,145	249,815	302,065	4.78	$ 866.72	$ 867.92	$ 829.05
37	43,830	0.49	431,648	0.61	537,807	0.125	5,479	8924.275	0.94	167,714	1.06	188,956	0.193	1,722	52,754	0.57	599,362	0.69	726,763	0.137	7,201	253,071	305,825	4.80	$ 867.29	$ 868.51	$ 828.19
38	44,132	0.49	434,384	0.61	539,374	0.125	5,495	8995.311	0.94	168,805	1.05	189,689	0.192	1,730	53,127	0.57	603,189	0.69	729,063	0.136	7,225	254,795	307,922	4.80	$ 867.40	$ 868.67	$ 827.13
39	44,604	0.49	437,017	0.61	541,498	0.124	5,529	9037.329	0.94	169,361	1.05	190,023	0.192	1,736	53,641	0.57	606,377	0.68	731,521	0.135	7,265	256,279	309,920	4.78	$ 867.64	$ 868.91	$ 826.71
40	44,985	0.49	440,209	0.60	543,318	0.123	5,548	9141.013	0.93	170,895	1.04	190,696	0.191	1,746	54,126	0.56	611,104	0.68	734,015	0.135	7,293	258,325	312,450	4.77	$ 867.86	$ 869.11	$ 824.21
41	45,236	0.49	441,885	0.60	544,716	0.123	5,571	9185.835	0.93	171,553	1.04	191,095	0.191	1,751	54,422	0.56	613,438	0.68	735,811	0.135	7,321	259,734	314,156	4.77	$ 867.89	$ 869.14	$ 823.30
42	45,568	0.49	443,776	0.60	548,274	0.123	5,594	9205.314	0.93	172,023	1.04	191,407	0.191	1,755	54,774	0.56	615,800	0.68	739,682	0.134	7,349	262,257	317,031	4.79	$ 867.85	$ 869.08	$ 822.52
43	50,263	0.47	474,539	0.57	572,961	0.122	6,147	9727.352	0.93	180,207	1.00	195,107	0.191	1,854	59,991	0.55	654,746	0.64	768,068	0.133	8,000	292,466	352,456	4.88	$ 866.12	$ 866.62	$ 807.57
44	50,710	0.47	476,449	0.57	574,813	0.122	6,185	9800.322	0.92	181,241	1.00	195,984	0.189	1,856	60,511	0.54	657,690	0.64	770,797	0.133	8,041	293,769	354,280	4.85	$ 865.86	$ 866.32	$ 804.96
45	51,314	0.47	479,020	0.56	578,063	0.122	6,236	9837.061	0.93	182,193	1.00	196,228	0.189	1,864	61,151	0.54	661,214	0.63	774,291	0.132	8,100	295,904	357,055	4.84	$ 865.41	$ 865.80	$ 803.33
46	51,493	0.47	479,649	0.56	578,851	0.121	6,248	9853.812	0.93	182,391	1.00	196,395	0.189	1,866	61,347	0.54	662,040	0.63	775,246	0.132	8,114	296,354	357,701	4.83	$ 865.26	$ 865.64	$ 802.99
47	51,858	0.46	481,240	0.56	581,276	0.121	6,279	9866.543	0.93	182,603	1.00	196,495	0.189	1,867	61,724	0.54	663,843	0.63	777,771	0.132	8,146	297,678	359,402	4.82	$ 864.87	$ 865.23	$ 802.05
48	52,081	0.46	482,736	0.56	583,046	0.121	6,293	9896.604	0.92	183,022	0.99	196,745	0.189	1,870	61,978	0.54	665,758	0.63	779,791	0.132	8,163	299,338	361,316	4.83	$ 864.42	$ 864.75	$ 800.60
49	52,330	0.46	483,888	0.56	584,092	0.121	6,308	9950.283	0.92	183,790	0.99	197,116	0.188	1,875	62,281	0.54	667,677	0.63	781,208	0.131	8,183	300,573	362,854	4.83	$ 863.91	$ 864.18	$ 798.57
50	52,586	0.46	484,852	0.56	585,368	0.120	6,330	9960.25	0.92	183,878	0.99	197,195	0.188	1,877	62,546	0.53	668,731	0.63	782,562	0.131	8,206	301,266	363,812	4.82	$ 863.54	$ 863.81	$ 798.06
51	56,989	0.46	528,038	0.55	622,043	0.116	6,624	11606.72	0.91	211,895	1.01	235,314	0.180	2,093	68,596	0.54	739,933	0.62	857,357	0.127	8,717	387,954	456,550	5.66	$ 833.74	$ 833.15	$ 733.85

16.5 Road and Ramp Design

Road designs have been completed for the PEA to allow primary access for people, equipment, and consumables to the site. This includes haul roads between the designed pits, dumps, and proposed leach facility. Within the pit designs, ramps have been established for haul truck and equipment access. The in-pit ramps will only require a single berm. Ramps outside of the pit will require two safety berms. One-lane traffic ramps are also utilized near the bottom of pits where the strip ratio is minimal, and the traffic requirements are low.

The ramps and haul roads assume the use of CAT-777 haul trucks with an operating width of 20 feet. For two-way access the goal of the road design is to allow a running width of near 3.5 times the width of the trucks. MSHA regulations specify that safety berms be maintained at least ½ of the diameter of the tires of the haul trucks that will travel on roads. The ½ height of the CAT-777 haul trucks tires is 4.5 feet. An extra 10% was added to berm height design to ensure that all berms are sufficient in height.

Safety berms assume a slope of 1.5 horizontal to 1.0 vertical. Considering that ramps in the pit only need one berm, the road width of 85 feet was determined for two-lane traffic, which allows for 3.5 times the operating width of the haul trucks. Single-lane traffic roads are estimated to require 55 feet which allows 2.0 times the operating width of the CAT-777 haul trucks.

Roads outside of the pit will require two berms and widths are estimated to be 100 feet, allowing 3.5 times the width of the CAT-777 haul trucks.

Road designs are intended to have a maximum of 10% gradient, though some may exceed this for short distances around inside turns. Where switchbacks are utilized, the centerline gradient is reduced to about 8%. This keeps the inside gradient approximately 12%. Switchback designs have not added the detail for super elevation through the curves, but is it assumed that this will be done when they are constructed.

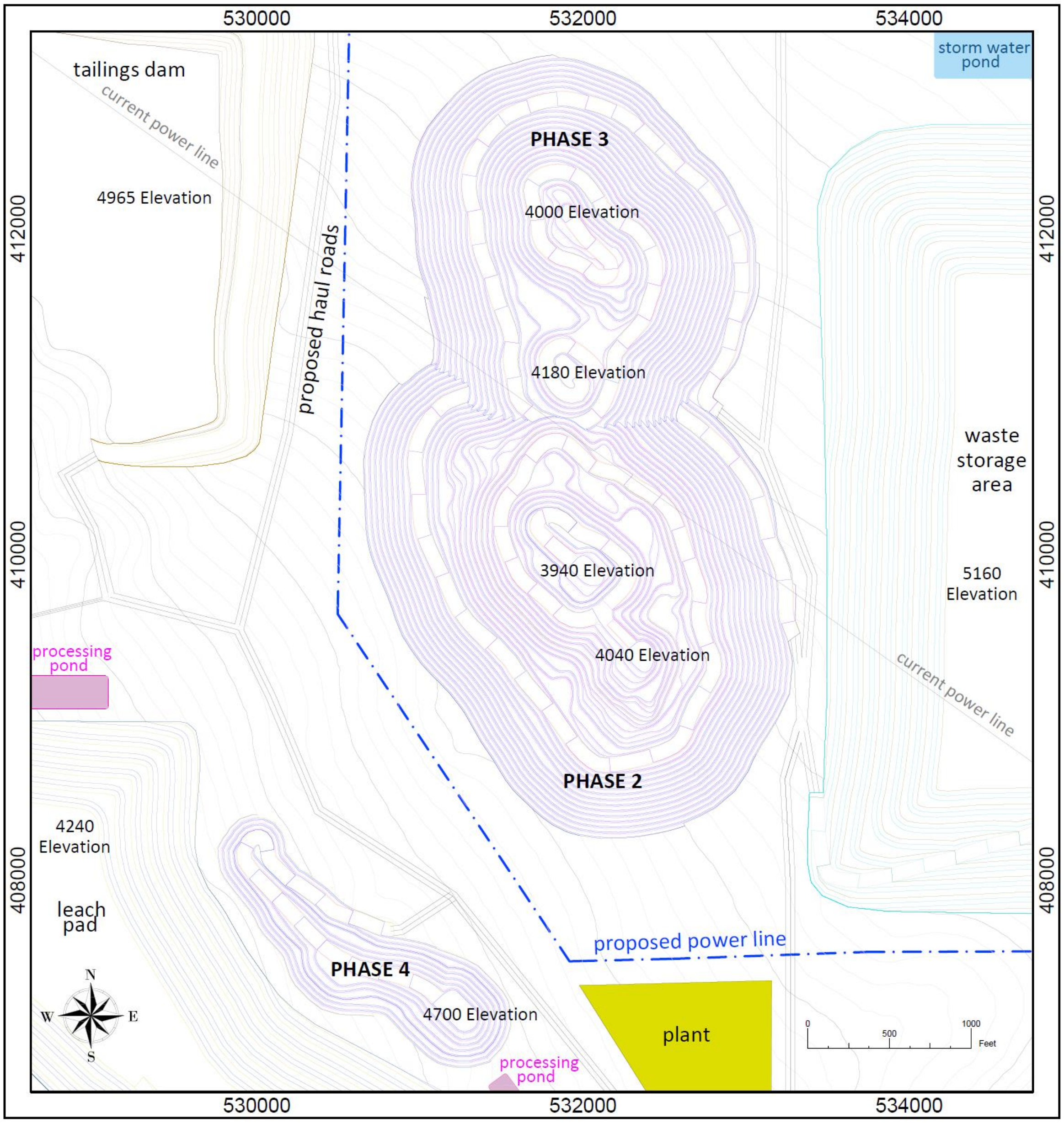
16.6 Pit Design

Pit designs were completed for Strong and Harris using Surpac™ software (version 2020.1). Each of the designs utilize 20-foot benches with a catch bench of 19 feet wide installed every other bench, or 40 feet apart. The bench face angle used is 70°. The resulting inner-ramp slope is 50°.

Strong and Harris pit designs were completed using four pit phases. Phase one is mined as a starter pit in the southern portion of the main pit. Phase two is mined around phase one to complete the southern portion of the main pit, followed by phase three which is designed as the northern portion of the main pit. Phase four is the small pit located to the south of the main pit designs.

The ultimate pit design is shown in Figure 16.2 while Figure 16.3 and Figure 16.4 show the phase one and phase two pit designs respectively.

Figure 16.2 Strong and Harris Ultimate Pit Design



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PROJECT
STRONG & HARRIS PROJECT
Copper-Zinc-Silver Deposit
Cochise County, Arizona

TITLE
ULTIMATE PIT DESIGN

COORDINATE SYSTEM:	Arizona State Plane East NAD 83 USFT
DDD-MM-YYYY:	13-Oct-2021
FILE NAME:	Ultimate_Pit.dwg

Figure 16.3 Strong and Harris Phase One Pit Design

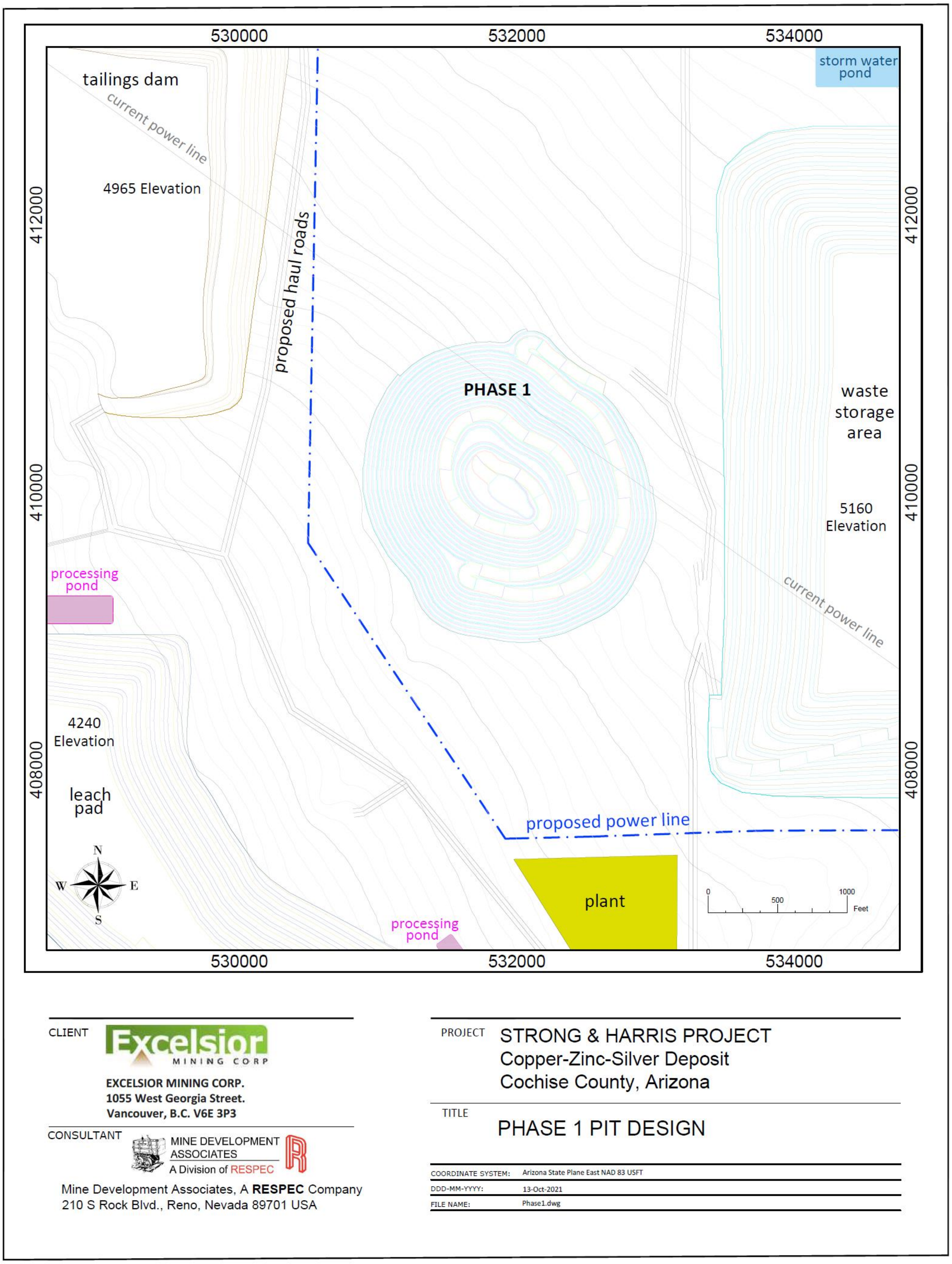
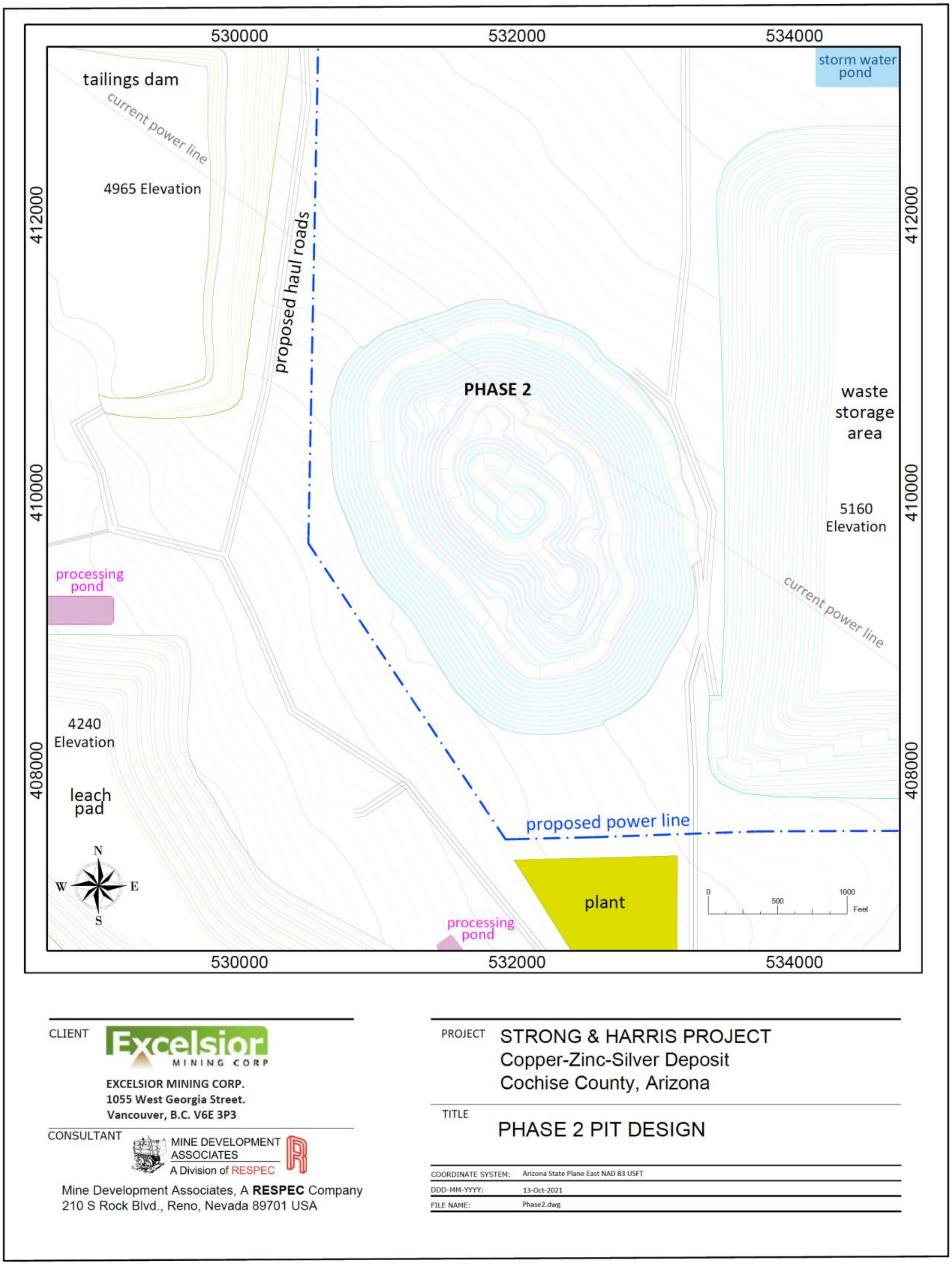


Figure 16.4 Strong and Harris Phase Two Pit Design



Note that the ultimate pit design shows a ramp that is "cutoff" by the mining of phase three at about the 4332 elevation. This is intended to be completed using backfill from phase three mining into phase two to complete the ramp. The backfill is shown in the overall site map in Figure 18.1.

16.7 In-Pit Resources

In-pit resources (Table 16.6) were estimated based on the resource block model, NSR cutoff grades, and the pit designs discussed above.

Table 16.6 Total In-Pit Resource Resources by Material Type

	Material Processed							
	Leach Processed		Flotation Processed		Total	Waste	Total	Strip
Units	Oxide	Mixed	Mixed	Sulfide	Processed	Mined	Mined	Ratio
K Tons	24,419	20,676	4,225	4,310	53,629	275,240	328,869	5.13
CuOx %	0.45	0.28						
K Lbs CuOx	218,359	115,340	N/A	N/A	N/A			
Total Cu %	0.53	0.44	1.23	0.66	0.56			
K Lbs Total Cu	256,542	180,154	104,218	56,886	597,800			
Zn %	0.65	0.54	1.44	0.72	0.68			
K Lbs Zn	318,864	224,697	121,935	61,699	727,195			
oz Ag/ton	0.114	0.132	0.189	0.199	0.134			
K Ozs Ag *	2,780	2,729	801	859	7,168			

**** Due to uncertainties in Ag refining and payments no value is given to Ag***
Note that all material processed is from Inferred Mineral Resources

	Total Leach Material Above Cutoff						
Phase	K Tons	Cu %	K Cu Lbs	Zn %	K Zn Lbs	z Ag/to	K Ozs Ag
Phase 1	16,197	0.52	169,550	0.65	209,944	0.126	2,044
Phase 2	20,011	0.42	166,512	0.54	216,178	0.110	2,207
Phase 3	6,051	0.68	82,523	0.84	101,610	0.168	1,017
Phase 4	2,835	0.32	18,111	0.28	15,828	0.085	241
Total	45,095	0.48	436,696	0.60	543,561	0.122	5,509

	Total Flotation Material Above Cutoff						
Phase	K Tons	Cu %	K Cu Lbs	Zn %	K Zn Lbs	z Ag/to	K Ozs Ag
Phase 1	3,072	1.12	68,921	1.31	80,561	0.221	680
Phase 2	5,160	0.81	84,112	0.90	93,304	0.175	902
Phase 3	277	1.37	7,596	1.73	9,590	0.272	75
Phase 4	25	0.94	476	0.35	179	0.093	2
Total	8,535	0.94	161,104	1.08	183,634	0.194	1,660

	Total Material Above Cutoff						
Phase	K Tons	Cu %	K Cu Lbs	Zn %	K Zn Lbs	oz Ag/to	K Ozs Ag
Phase 1	19,269	0.62	238,471	0.75	290,505	0.141	2,724
Phase 2	25,172	0.50	250,623	0.61	309,482	0.123	3,109
Phase 3	6,328	0.71	90,119	0.88	111,200	0.173	1,093
Phase 4	2,860	0.32	18,587	0.28	16,007	0.085	243
Total	53,629	0.56	597,800	0.68	727,195	0.134	7,168

	Waste Material (K Tons)				Total	Strip
Phase	Oxide	Mixed	Sulfide	Total	K Tons	Ratio
Phase 1	83,899	1,012	256	85,167	104,435	4.42
Phase 2	71,401	4,255	2,529	78,185	103,357	3.11
Phase 3	102,392	1,324	39	103,755	110,083	16.40
Phase 4	7,598	534	-	8,133	10,993	2.84
Total	265,290	7,126	2,824	275,240	328,869	5.13

16.8 Dump Design

A single waste dump is designed to the east of the ultimate pit along with in-pit back fill located in the phase two pit design. These designs are shown in the site layout in Figure 18.1. The in-pit backfill is minimized and is used to maintain the ramp for the northern phase 3 design.

The east dump was designed to contain up to 163,212,000 cubic yards of material, which is within 1% or 2,000 cubic yards less than the total capacity needed considering a 1.4 swell factor. The remainder of the waste material can be placed into the phase 1 and 2 pit designs. Depending on efficiencies, it may be beneficial to costs to add more material to the in-pit backfill. This will be confirmed in additional studies.

16.9 Production Schedule

Production scheduling was completed using MineSched software (version 2020.1). The production was primarily driven by constraining the throughput of oxide and mixed leach material to 7.2 million tons per year, and mixed and sulfide material to the flotation plant to 1.8 million tons per year. Additional constraints to production limits included up to 2 benches per month could be mined in any given pit phase to manage the monthly sink rate. Mining assumes the use of contractors and their equipment to sustain the productivity required to feed the leach pad and flotation plant.

The resulting production schedule requires approximately 12 months of preproduction to strip waste above the deposit with some leach material being mined in month minus one. Leach production is ramped up through the first four months of production to full capacity in month five.

Flotation production is assumed to start at the beginning of production year two. Some flotation material will be stockpiled from the lower portions of phase 1 mining. It is assumed that some ramp up and commissioning of the plant will happen in year one, but no production is attributed to year one.

Table 16.7 shows the mine production schedule, while Table 16.8 and Table 16.9 show the leach and flotation production respectively.

Table 16.7 Mine Production Schedule

		Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Total
Total	Total Leach	K Tons	7	6,757	10,437	7,794	7,192	5,152	7,756	-	45,095
		Cu %	0.214	0.422	0.530	0.410	0.477	0.411	0.607	-	0.484
		K Cu Lbs	28	57,039	110,558	63,932	68,658	42,306	94,176	-	436,696
		Zn %	0.04	0.50	0.65	0.52	0.66	0.55	0.68	-	0.60
		K Zn Lbs	5	68,200	136,534	81,211	94,991	56,880	105,739	-	543,561
		oz Ag/ton	0.065	0.090	0.129	0.117	0.122	0.130	0.143	-	0.122
		K Ozs Ag	0	606	1,342	909	875	669	1,106	-	5,509
	Total Flotation	K Tons	-	391	2,419	1,133	1,595	2,694	302	-	8,535
		Cu %	-	1.162	1.067	0.993	0.936	0.742	1.335	-	0.944
		K Cu Lbs	-	9,081	51,611	22,497	29,852	39,990	8,072	-	161,104
		Zn %	-	1.33	1.25	0.94	1.11	0.85	1.62	-	1.08
		K Zn Lbs	-	10,374	60,686	21,362	35,497	45,946	9,769	-	183,634
		oz Ag/ton	-	0.148	0.217	0.183	0.176	0.190	0.257	-	0.194
		K Ozs Ag	-	58	526	207	280	511	78	-	1,660
	Total Processed	K Tons	7	7,148	12,856	8,927	8,787	7,846	8,058	-	53,629
		Cu %	0.214	0.463	0.631	0.484	0.561	0.524	0.634	-	0.557
		K Cu Lbs	28	66,120	162,169	86,429	98,510	82,296	102,248	-	597,800
		Zn %	0.04	0.55	0.77	0.57	0.74	0.66	0.72	-	0.68
		K Zn Lbs	5	78,574	197,221	102,573	130,488	102,826	115,508	-	727,195
		oz Ag/ton	0.065	0.093	0.145	0.125	0.131	0.150	0.147	-	0.134
		K Ozs Ag	0	664	1,868	1,116	1,155	1,181	1,184	-	7,168
	Ox_Wst	K Tons	53,800	47,525	40,999	44,674	36,430	24,755	17,107	-	265,290
	Mx_Wst	K Tons	-	46	820	1,111	978	2,312	1,859	-	7,126
	Su_Wst	K Tons	-	31	224	38	905	1,588	39	-	2,824
	Total Waste	K Tons	53,800	47,602	42,044	45,823	38,313	28,654	19,004	-	275,240
	Total Mined	K Tons	53,807	54,750	54,900	54,750	47,100	36,500	27,063	-	328,869
	Strip Ratio	W:O	8,273.24	6.66	3.27	5.13	4.36	3.65	2.36		5.13

Table 16.8 Leach Process Production Schedule

Leach Material Processed	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Oxide	K Tons	-	4,196	4,788	2,986	2,988	2,197	5,305	1,959	-	24,419
	Cu %	-	0.403	0.524	0.386	0.605	0.435	0.601	0.778	-	0.525
	K Lbs Cu	-	33,789	50,199	23,054	36,138	19,093	63,785	30,485	-	256,542
	K Lbs Cu Prod	-	20,516	38,925	19,769	30,376	11,582	50,648	29,729	-	201,545
	Zn %	-	0.472	0.676	0.527	0.884	0.617	0.641	0.895	-	0.653
	K Lbs Zn	-	39,594	64,754	31,480	52,818	27,103	68,046	35,069	-	318,864
	K Lbs Zn Prod	-	25,855	51,359	29,572	45,758	18,062	56,455	35,365	-	262,425
	oz Ag/ton	-	0.08	0.11	0.09	0.12	0.13	0.14	0.13	-	0.11
	K Ozs Ag	-	325	549	281	352	279	735	258	-	2,780
Mixed	K Tons	-	1,712	2,432	4,214	4,212	5,003	1,915	1,188	-	20,676
	Cu %	-	0.473	0.518	0.402	0.387	0.449	0.437	0.448	-	0.436
	K Lbs Cu	-	16,183	25,188	33,851	32,605	44,941	16,727	10,658	-	180,154
	K Lbs Cu Prod	-	6,580	13,595	22,385	15,856	30,737	7,509	9,797	-	106,459
	Zn %	-	0.630	0.584	0.482	0.502	0.560	0.569	0.591	-	0.543
	K Lbs Zn	-	21,554	28,408	40,635	42,275	55,990	21,791	14,044	-	224,697
	K Lbs Zn Prod	-	12,540	21,836	35,981	30,414	52,268	15,208	16,679	-	184,926
	oz Ag/ton	-	0.12	0.14	0.13	0.12	0.13	0.14	0.16	-	0.13
	K Ozs Ag	-	208	345	529	524	661	271	191	-	2,729
Total Leach Processed	K Tons	-	5,908	7,220	7,200	7,200	7,200	7,220	3,147	-	45,095
	Cu %	-	0.423	0.522	0.395	0.477	0.445	0.558	0.654	-	0.484
	K Lbs Cu	-	49,972	75,387	56,905	68,743	64,034	80,512	41,143	-	436,696
	K Lbs Cu Prod	-	27,097	52,520	42,154	46,232	42,319	58,157	39,526	-	308,004
	Zn %	-	0.518	0.645	0.501	0.660	0.577	0.622	0.780	-	0.603
	K Lbs Zn	-	61,147	93,162	72,115	95,094	83,093	89,837	49,113	-	543,561
	K Lbs Zn Prod	-	38,396	73,194	65,552	76,172	70,330	71,662	52,044	-	447,351
	oz Ag/ton	-	0.09	0.12	0.11	0.12	0.13	0.14	0.14	-	0.12
	K Ozs Ag	-	532	894	810	876	941	1,006	449	-	5,509

Table 16.9 Flotation Process Production Schedule

Flotation Material Process	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Mixed	K Tons	-	-	951	1,291	1,031	561	391	-	-	4,225
	Cu %	-	-	1.486	1.032	1.213	1.200	1.384	-	-	1.233
	K Lbs Cu	-	-	28,272	26,648	25,025	13,465	10,807	-	-	104,218
	K Lbs Cu Prod	-	-	17,284	24,201	19,940	11,356	10,492	206	-	83,479
	Zn %	-	-	1.833	1.048	1.467	1.360	1.856	-	-	1.443
	K Lbs Zn	-	-	34,883	27,039	30,262	15,259	14,492	-	-	121,935
	K Lbs Zn Prod	-	-	18,489	22,530	20,391	11,229	12,276	73	-	84,989
	oz Ag/ton	-	-	0.19	0.18	0.18	0.19	0.25	-	-	0.19
	K Ozs Ag	-	-	185	227	183	107	98	-	-	801
Sulfide	K Tons	-	-	853	509	737	1,239	971	-	-	4,310
	Cu %	-	-	0.751	0.819	0.565	0.599	0.646	-	-	0.660
	K Lbs Cu	-	-	12,826	8,341	8,327	14,855	12,536	-	-	56,886
	K Lbs Cu Prod	-	-	9,494	6,698	6,651	11,640	10,931	153	-	45,566
	Zn %	-	-	0.745	0.909	0.600	0.658	0.750	-	-	0.716
	K Lbs Zn	-	-	12,723	9,256	8,845	16,309	14,566	-	-	61,699
	K Lbs Zn Prod	-	-	8,007	6,577	5,830	11,294	11,177	120	-	43,004
	oz Ag/ton	-	-	0.21	0.25	0.20	0.19	0.18	-	-	0.20
	K Ozs Ag	-	-	177	125	146	238	173	-	-	859
Total Floatation Processed	K Tons	-	-	1,805	1,800	1,768	1,800	1,361	-	-	8,535
	Cu %	-	-	1.139	0.972	0.943	0.787	0.857	-	-	0.944
	K Lbs Cu	-	-	41,099	34,989	33,353	28,320	23,343	-	-	161,104
	K Lbs Cu Prod	-	-	26,778	30,898	26,591	22,996	21,423	358	-	129,044
	Zn %	-	-	1.319	1.008	1.106	0.877	1.067	-	-	1.076
	K Lbs Zn	-	-	47,606	36,295	39,107	31,568	29,058	-	-	183,634
	K Lbs Zn Prod	-	-	26,496	29,107	26,221	22,523	23,453	193	-	127,993
	oz Ag/ton	-	-	0.20	0.20	0.19	0.19	0.20	-	-	0.19
	K Ozs Ag	-	-	361	353	329	345	271	-	-	1,660

16.10 Equipment Requirements

The Strong and Harris project assumes mining will be done using 100-ton capacity haul trucks and wheeled loaders capable of loading haul trucks in four passes. Additional primary mining equipment will be required for drilling and blasting, and additional equipment will be needed to support the primary mining equipment. This equipment will be provided by the mining contractor to provide the productivity needed to meet the production requirements. It is anticipated that the contractor will need the following equipment during peak operations:

- A peak of four 15-yard capacity wheel loaders to load haul trucks and help maintain feed from stockpiles as needed;
- Up to 18 100-ton capacity haul trucks to haul material from the pits to the dumps, pads, and stockpiles;
- Three blast hole drills capable of drilling single passes of up to 30-foot deep blast holes;
- Two powder trucks and one skid loader to load and stem shot patterns;
- Three 600-HP sized dozers with rippers and U-Blades for maintaining waste dump and leach pad locations;

- One 440-HP dozer for general maintenance of roads, pit floors, and stockpiles;
- Two 18-ft moboard graders for road maintenance;
- Two 20,000 gallon water trucks required for dust suppression on haul roads;
- One large lubricant and fuel truck for fueling of equipment that stays in the pit or on dumps (dozers, drills, loaders, etc.);
- A fueling distribution system capable of storing and delivering fuel to mobile equipment with the capacity to hold up to 10 days of fuel to sustain production as required;
- A 1.5-cubic foot wheeled backhoe for general construction and maintenance around the site; and
- An 8-cubic foot track excavator with a rock breaker attachment for utility work and breakage of oversized rock as needed.

16.11 Mine Personnel

Mining personnel will be minimized through the use of contract mining. The PEA assumes the contractor will provide the equipment and personnel required to meet the production schedule. Depending on the type of equipment used, the contractor will likely have a maximum of 120 personnel during the first five years.

In addition to the contractor crews, the owner will require additional personnel to manage the mining operations. The PEA owner's mining cost assumes a total of seven salary and hourly personnel including:

- One Chief Mine Engineer to manage mine planning, surveying and mine capital projects;
- One Staff Mine Engineer to assist the Chief Engineer and surveyor;
- One surveyor for general projects around the mine and to confirm contractor mined volumes;
- One Chief Geologist to supervise and maintain geological mapping and grade control;
- One Staff Geologist to assist the Chief Geologist and samplers; and
- One Sampler to monitor and collect grade-control samples from the contractor drillers.

The contractor will provide personnel as required to operate and manage the contractor mining fleet and meet production requirements. The contractor will be required to operate 24 hours per day and seven days per week with the exception of holidays and weather days. The contractor will likely have a general supervisor overseeing the operation with their own surveying crew for mining controls. The contractor will operate either two or three crews per day, depending on the contractor mining strategy. To match the equipment requirements, each crew will require approximately twenty load and haul operators, seven drill and blasting personnel, seven mine support operators, and six mechanics and service persons for a total of about 41 people for mine operations.

17.0 RECOVERY METHODS (ITEM 17)

17.1 Introduction

At this point in the development of the Strong and Harris project, no recent testing has been done on the processing of material from the three mineralogical zones. For purposes of this assessment, two conceptual processing routes have been developed using information on successful processing of material that is chemically and geologically similar to that found at Strong and Harris, as well as the historical testwork. For the sulfide material, flotation will be used to recover and then separate the copper and zinc. Acid heap leaching, followed by SX-EW will be used to treat the oxide material. The transition material may be treated by either flotation or heap leach. The choice will depend on the mineralized grade and metal prices in effect when the material is scheduled for mining. Note that the flotation route has the advantage of recovering some portion of any silver that may be present. The acid leach will not solubilize the silver.

There are numerous examples of polymetallic operations that treat sulfide ores and produce separate copper and zinc concentrate by flotation. This is well established technology and the common elements of these operations were used to develop the Strong and Harris process flow diagram ("PFD") for treating the sulfide and selected mineralized transition material.

There are also numerous operations extracting copper by acid leaching, followed by recovery of the copper as cathodes using SX-EW technology. Many of these operations treat ores with high acid-soluble copper contents. However, the chemistry of the leach solution can be controlled so that various copper sulfide minerals can also be oxidized and solubilized. Thus, there are also a number of successful heap leach operations treating sulfide ores where the copper is recovered using SX-EW technology.

The situation is not the same for zinc, where there is only a single operation commercially recovering zinc using a leach step, followed by a SX-EW operation. This is the Skorpion zinc mine in Namibia. However, here the leaching is done in agitated tanks treating ground material. Thus, Skorpion does not utilize the heap leaching of crushed or run-of-mine ("ROM") material. Also, it should be noted that the Skorpion operation is not a binary metal leach like that proposed for Strong and Harris. At Skorpion the copper impurity in the leach liquor is scrubbed from the solution prior to zinc solvent extraction. However, the copper is currently being recovered from the wash solution as a by-product.

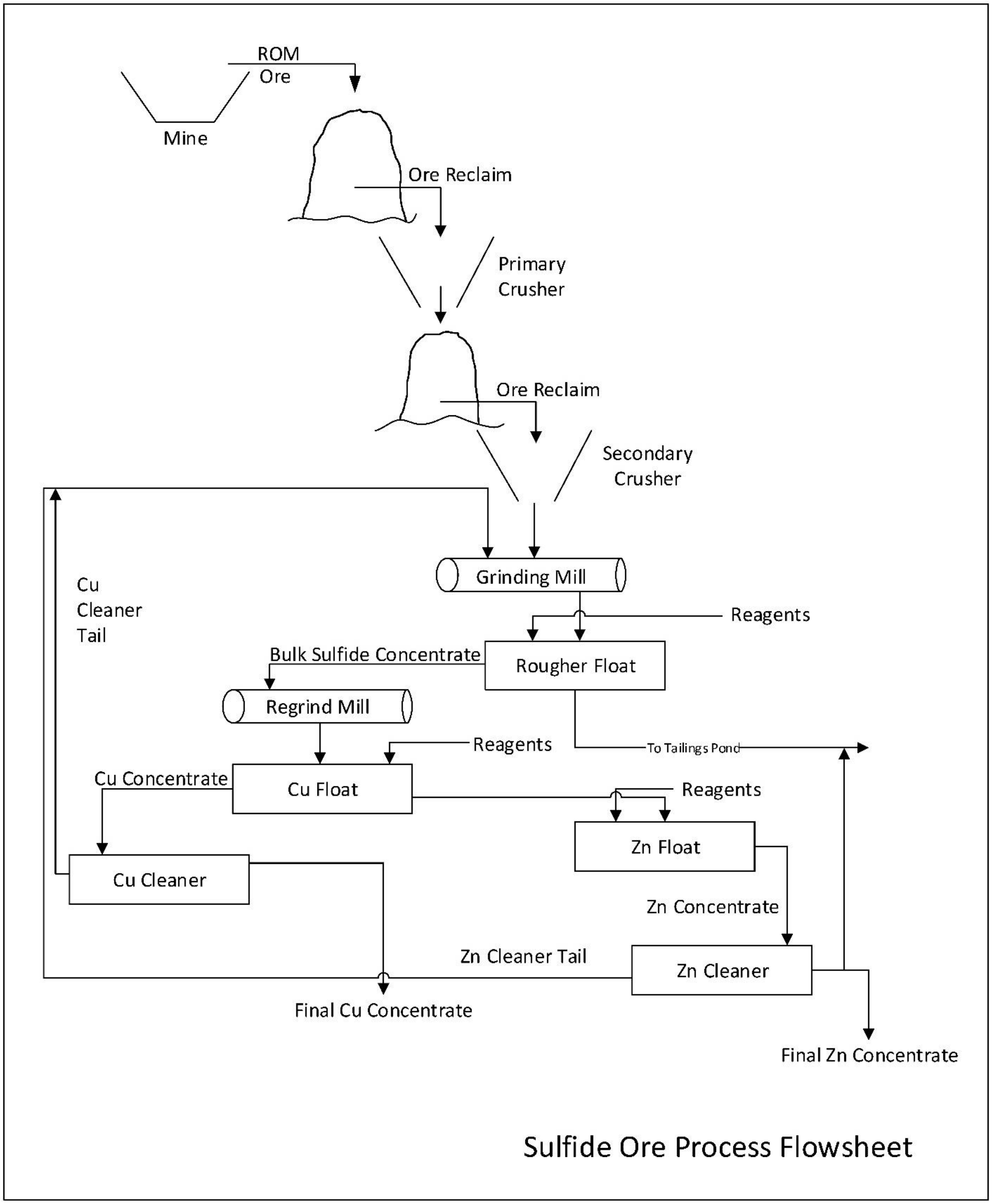
Further evidence supporting a zinc heap leach-solvent extraction process is provided by Qin et al. (2007). This work describes small-scale column-leach tests on oxide zinc material integrated with a laboratory scale solvent extraction mixer-settler. The pregnant leach solution ("PLS") from the column was subjected to solvent extraction, scrubbing and selective stripping for the enrichment of the zinc and the removal of impurities. The resulting zinc sulfate solution was suitable for zinc electrowinning. While copper was leached along with the zinc, the copper was much lower grade and was not recovered. Additional information on zinc solvent extraction and electrowinning is provided in Soles et al. (2020).

In view of the foregoing, the PFD developed for the Strong and Harris copper-zinc heap leach is conceptual in nature. The portions on copper leaching and recovery using SX-EW are based on generally accepted practices within the industry. The portions relating to zinc leaching and recovery are more problematic. However, as reported in Section 13, high zinc extractions can be achieved when acid leaching of the oxide mineralization for zinc. Although zinc can be recovered from PLS using SX-EW, there is uncertainty in how effectively zinc can be recovered sequentially when treating the raffinate from the copper SX circuit. The two PFDs are presented in the following sections, along with some additional comments on each.

17.2 Recovery of Copper and Zinc by Flotation.

The generic PFD developed for the Strong and Harris sulfide and selected transition material is shown in Figure 17.1. Generally speaking, the flow of material is from top to bottom. Both ROM and crushed rock stockpiles are included to smooth the flow of material to grinding. A grizzly may be used with the primary crusher to remove oversized material for further breakage. The secondary crusher discharge will be fed to the grinding mill, along with sufficient water to produce the slurry density wanted for flotation.

Figure 17.1 Flotation Process Flow Diagram: Separate Copper and Zinc Sulfide Concentrates



In some cases the mill discharge may be screened with the oversized fraction returned to the mill feed. The screen undersized then goes to flotation.

The flotation circuit starts with the rougher flotation section. Here reagents are used to collect the sulfides into a bulk sulfide concentrate that contains both the copper and the zinc, plus any silver. The non-sulfidic gangue does not float and is discharged to tailings dam. The solids in the tailings will settle out and sink leaving a water-filled pond. Water from the pond is typically returned to the grinding circuit in order to conserve both water and the contained flotation reagents.

The bulk concentrate is reground to improve liberation of the individual copper and zinc sulfide particles from each other or from gangue particles. The reground material advances to the copper float cells where reagents are added to float the copper and depress the zinc minerals. The copper flotation concentrate will advance to one or more cleaning stages intended to increase the copper grade to a level acceptable to copper smelters. There will be some reject material produced during cleaning. Since this cleaner tail is likely high in copper, it will be recycled back to the grinding mill for another pass through the flotation process.

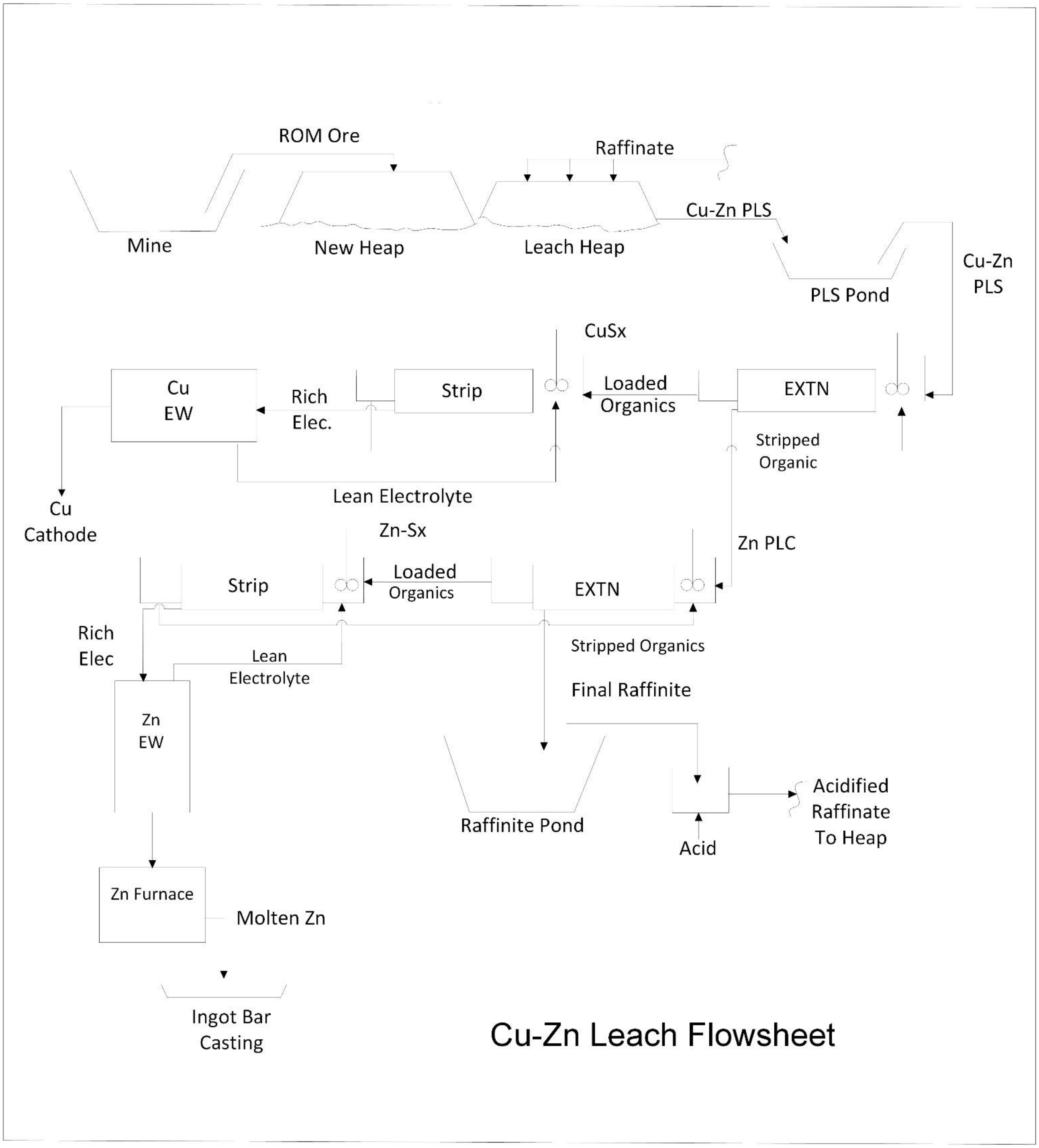
The zinc bearing tail from the copper flotation stage will be sent to the zinc flotation circuit. Here reagents will be added to overcome the zinc depression and float the zinc sulfides. As with the copper concentrate, the zinc concentrate will be sent to one or more cleaner steps in order to meet the required zinc content. Any zinc cleaner tails will also be returned to the grinding circuit for another pass through the flotation plant.

17.3 Recovery of Copper and Zinc by Leaching and Solvent Extraction-Electrowinning

The conceptual process flow diagram for the Strong and Harris leach option is shown in Figure 17.2. As shown, the leaching will be done on ROM material stacked by trucks, then leveled and ripped with a dozer. Drilling and blasting parameters will be adjusted as necessary to minimize the production of excessive amounts of very coarse material (> 6 inches in diameter) or fines that could cause heap permeability problems.

Based on historical reports, the material to be leached is a high acid consumer. In order to avoid excessive acid consumption and even possible precipitation of the metal values, relatively shallow lifts will be needed, on the order of 8.0 to 12 feet. Each lift will be leached to exhaustion. Then the heap surface will be covered with an impervious liner, with drains installed before the next lift of material is emplaced. This so-called interlift liner will prevent high-grade PLS from percolating from the fresh lift down into leached-out material. If this were to happen, the solution grade would be diluted. There could also be possible re-deposition of copper if acid consumption was to continue, giving rise to an increase in the solution pH and a decrease in copper extraction.

Figure 17.2 Leaching Process Flow Diagram: Copper and Zinc, Oxide and Transition Material



To achieve maximum metal extraction, several leach parameters must be optimized in concert based on an appropriate testing program. These include the irrigation rate, acid concentration, and leach cycle time. The irrigation system will be laid out on the heap surface and the drip lines should be fairly closely spaced in order to wet the entire lift of material. Note that with interlift liners, there is no residual leaching of underlying material.

The PLS coming from various leach areas will comingle in the PLS pond. This mixing will help maintain a uniform PLS grade going to the solvent extraction plant. The pond will also allow for the settlement of any solids that wash in with the PLS due to a precipitation event or a broken leach line. From the pond the PLS will advance to the extraction mix box in the copper SX circuit. Here the PLS will be mixed with the stripped organic returned from the strip circuit. During mixing the copper will transfer from the PLS to the organic. The extractant will be one that is specific for copper and does not load the zinc. The aqueous-organic emulsion discharging from the mix box into the settler will separate into a loaded organic layer on the top and an aqueous layer on the bottom.

The loaded organic will advance to the copper strip section where the organic is mixed with lean electrolyte being returned from the copper tankhouse. Due to the high acid level in the electrolyte the copper transfer reaction is reversed with copper pulled from the organic into the electrolyte. When the organic-electrolyte emulsion discharges into the strip settler, there will again be a floating organic layer above the denser aqueous layer. The resulting stripped organic will advance back to the extraction mixer box. The aqueous is the rich electrolyte and will advance to the copper tankhouse where the copper is deposited on stainless steel blanks. Once these copper deposits reach the desired weight the resulting copper cathodes will be stripped as the final product.

The de-copperized aqueous phase produced in the copper extraction circuit will contain the zinc and becomes the feed to the zinc SX circuit. It is critical that this stream does not carry over any entrained copper organic into the zinc circuit. Therefore the zinc feed stream may require an additional organic coalescer stage to maximize organic removal. Any organic recovered here can be returned to the copper strip circuit.

The zinc SX circuit will operate in the same manner as the copper SX circuit described above. The incoming zinc-bearing leach solution will be mixed with stripped organic and the emulsion will separate into the loaded organic and raffinate in the extraction settler. The organic will advance to the zinc strip circuit where it will be mixed with lean zinc electrolyte. This emulsion will settle, producing a lean organic phase to return to the zinc extraction circuit and rich electrolyte for the zinc tankhouse. The zinc will then be plated out as cathodes. These cathodes will be stripped, then melted and cast into zinc ingot bars as the final product.

The aqueous phase produced in the zinc extraction circuit contains minimal amounts of both copper and zinc and is discharged into the raffinate pond. Here any entrained zinc organic can be recovered and returned to the zinc circuit. The raffinate will be acidified as needed and returned to the leach heap.

It should be understood that Figure 17.2 is generic and just shows a single extraction and strip stage for each circuit. In reality, optimum performance may involve multiple extraction or stripping stages. To optimize either tankhouse operation it is desirable to minimize the carryover of organic into the tankhouse. Thus, either operation or both may include some sort of organic coalescer on the rich electrolyte stream. Both electrowinning operations will also require small electrolyte bleeds to control the buildup of impurities. These bleeds can either be returned to their respective extraction stages or the final raffinate pond.

17.4 Conclusions

The proposed PFD for sulfide flotation has been developed from a combination of limited historical testwork done on district material, and analogous successful flotation operations on copper-zinc sulfide ores undertaken elsewhere. Once proper parametric studies have been completed, the proposed approach is expected to be appropriate for Strong and Harris.

The PFD proposed for leaching and recovery of zinc is more conceptual than that for flotation. Both copper heap leaching and copper recovery using SX-EW are widely practiced and provide the basis for the processing route shown in Figure 17.2. Zinc recovery via SX-EW is also established but is much less common. The conceptual part of the leach-SX-EW process flow diagram is the binary heap leaching of both copper and zinc with the sequential solvent extraction of copper, followed by the recovery of zinc in a parallel zinc SX circuit.

17.5 Recommendations

Leach processing of the oxide and transition mineralization will be impacted by the reportedly high acid consumption and the problems associated with sequential solvent extraction of copper, then zinc. Recommended activities needed to support leach optimization include the following:

- Prepare composites for each type of mineralized material that is potentially planned for leaching. If more than one mineralized rock type will be leached, then each type should be composited separately, as acid consumption may be controlled more by rock type than the copper/zinc mineralization. The composites will be used study the effect of such parameters as leach solution pH and irrigation rate on metal recovery and acid consumption. The effect of crush size on these parameters should also be investigated;
- Sighter tests should be performed to determine if leaching is enhanced by the presence of microbial species such as Thiobacillus ferrooxidans and/or Thiobacillus thiooxidans. Studies by Harahuc, et al. (2000) suggest that bioleaching would be beneficial. Microbial activity in copper leach operations has been found to increase the oxidation-reduction (redox) potential in the system, promoting the leaching of sulfide mineralization;

- Mineralized samples covering the expected grade range for each type of mineralized material should be prepared. These samples should then be used to determine grade-recovery curves for both copper and zinc; and
- Any benefit of having a rest period during the leach cycle should be investigated. In some cases the rest cycle has been found to boost recovery.

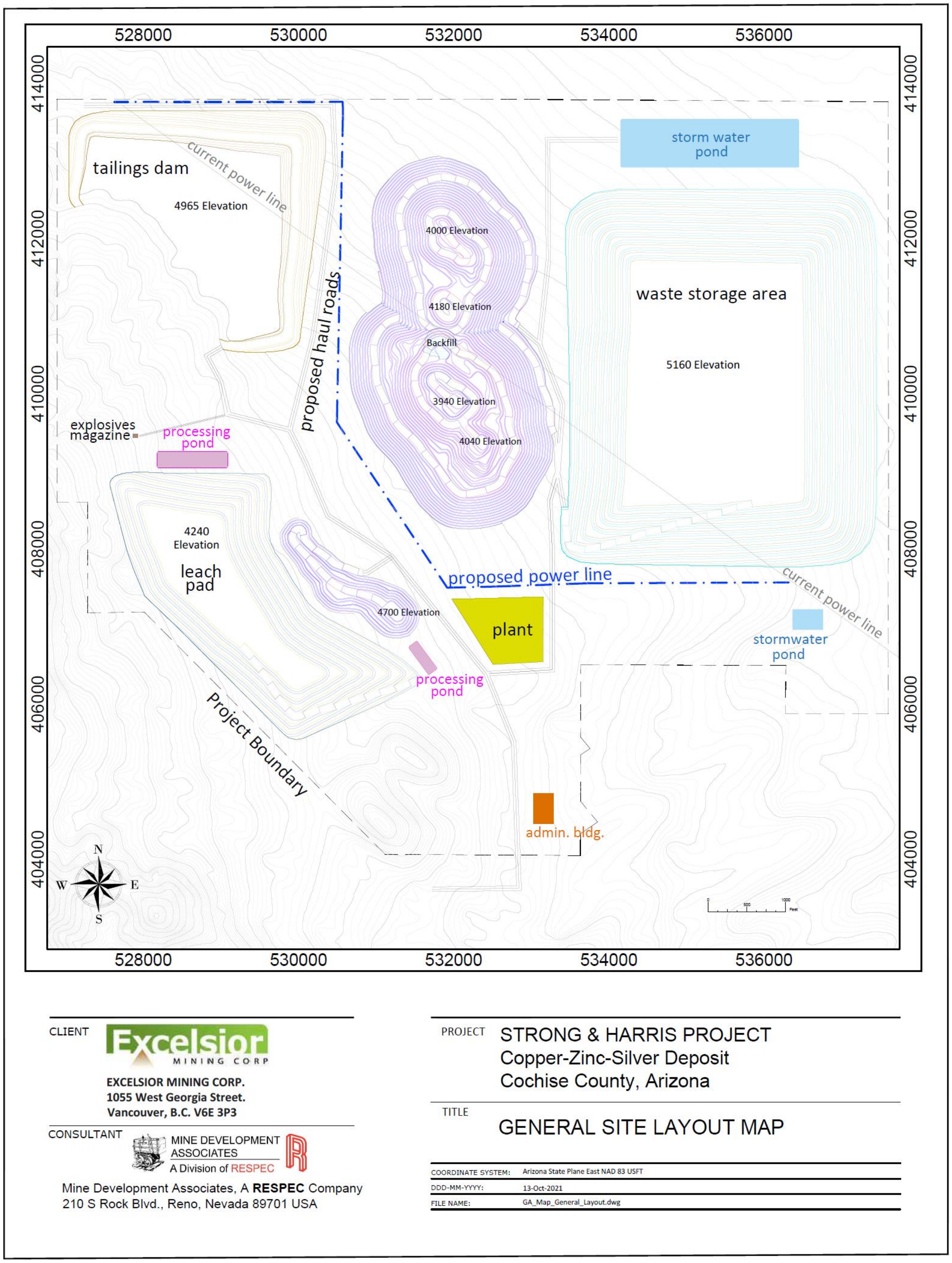
When the mixed or transition material is treated by flotation, the mineralization in the flotation tails may be similar to that found in the oxide material. Agitation leaching of the tailings would be similar to the Skorpion Zinc circuit. If successful, the PLS from a tailings leach operation could be combined with the feed to the existing SX circuit.

The proposed sequential recovery of copper, then zinc is not well established. If testing shows that zinc recovery is difficult, other zinc recovery options should be considered. One possibility would be to use ion exchange. This was done at Kennecott’s Bingham Canyon dump leach operation. Here uranium was recovered from the copper tail water using an ion exchange system. Another possibility might involve precipitation as zinc sulfide. If successful this product could be combined with the zinc flotation concentrate.

18.0 PROPERTY INFRASTRUCTURE (ITEM 18)

The general arrangement drawing is shown in Figure 18.1.

Figure 18.1 Strong and Harris General Arrangement Drawing



18.1 Power

Currently there is an existing 115 kV powerline that crosses the property from southeast to the northwest. This powerline will be relocated by constructing a new powerline from a location to the south of the waste storage area to the plant, and then north to the property boundary, and then running west and connecting to the existing powerline in the northwest.

A substation would be located in the plant area to provide needed power for the site. The bulk of the site power requirements will be for the plant and leaching facilities. A small amount of power will be used for administrative and mine contractor facilities.

18.2 Access Road and Administration Building

The current access road will be utilized to enter the site from the south side of the project. A double wide trailer is assumed to be used as an administration building near the entrance. This will house the offices for mine management and personnel with approximately eight offices and a general reception area.

18.3 Tailings Dam

The proposed tailings facility will be located in the northwest corner of the project area. The design shown in Figure 18.1 is conceptional and only assumes the storage of waste rock material in the embankment along with the volume requirements to store tailings. The tailings requirement is estimated to be 5.3 million cubic yards based on a 1.4 swell factor.

18.4 Explosives Silos and Magazines

The explosive silos and magazines are proposed to be to the south of the tailings facility and to the north of the leach pad. This will provide a suitable location to maintain security of explosives interior to the project, while being located at a distance from where personnel gather. It is assumed that two silos will be erected by the mining contractor for use in the storage of ammonia nitrate. The explosive magazines will be stored at a distance from the silos.

18.5 Growth Media Stockpiles

Growth media stockpiles will be located around the site to allow for the storage of topsoil removed from facility areas. The material will be used during reclamation at the end of the mine life.

19.0 MARKET STUDIES AND CONTRACTS (ITEM 19)

Other than royalties that are discussed in Section 4.5, no other contractual obligations have been entered into. The PEA assumes copper from heap leaching will be processed on site and sold on the copper market.

Flotation will produce a concentrate to be processed by a third-party refinery. No current contracts exist, but there are a large number of refineries throughout the world that provide refining services. The copper treatment, transportation, and penalty costs assumed are shown in Table 16.3.

The anticipated long-term demand for copper and zinc is not easily determined but for the purpose of this report, it has been assumed that markets for this product will remain steady. To date, no market study has been conducted for this project and there are no contracts in place related to mineral sales at the time of this report. No direct marketing has been done for the copper or zinc that would potentially be produced at the project and therefore no off-take agreements exist. These options will be reviewed in detail when the project proceeds to the feasibility stage. With all that being said, the copper market historically has been robust as to consumption requirements.

While initial considerations for the copper price for the PEA used \$3.00 per pound, the final optimizations, designs, and economic evaluations for the PEA use a \$3.50 per pound copper price. This is between the one-year average of \$3.85 per pound and the two-year average of \$3.41 per pound based on data from NASDAQ historical copper prices. Of note, during the last six months prior to the Effective Date, the copper price has been over \$4.00 per pound, with a high of \$4.61 per pound in the month of May 2021.

The average spot price for zinc in the month of August 2021 was \$1.355 per pound according to zinc monthly commodity prices published by IndexMundi, and with a one-year rolling average of \$1.261 per pound. Initial considerations for zinc were \$1.10 per pound and the ratio to copper was \$2.73 copper to \$1.00 zinc. For the PEA this ratio was maintained. Thus, the price of \$1.28 per pound of zinc was used.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT (ITEM 20)

This section has been prepared based on information provided by Mrs. Cindi Byrns, Excelsior's environmental expert.

20.1 Introduction

The Strong and Harris project is located within a closed hydrographic basin along the eastern side of the Little Dragoon Mountains. There are several full-time residences approximately three miles south of the Strong and Harris project. The area is used for livestock grazing. Although there are currently no known environmental studies, based upon the location and existing mining activities in the near vicinity, significant environmental impacts are not anticipated. The anticipated environmental liabilities will be addressed by the various permits prior to construction.

Excelsior is in the initial stages of the permitting process. Studies will be conducted to: characterize the potential mineralized material and waste; develop models to demonstrate non-degradation of groundwater; design waste rock stock piles and tailings impoundments; establish right-of-ways on county roads; and prepare the Plan of Development for the power line and road for submittal to the BLM.

As Strong and Harris is within an existing mining district, it is anticipated that social and/or community impact will be minimal. The project is accessed through the existing Johnson Camp facility, precluding public access. There are no social requirements or plans with the local communities.

20.2 Required Permits and Status

To date, no environmental studies have been initiated and no permits have been applied for. Studies and permits will be required prior to the project moving forward. The permits required for the Strong and Harris project are listed in Table 20.1. There are several layers of permitting with Federal, State and County regulatory agencies. In general, the State of Arizona has primacy of all environmental permitting obligations; therefore, the US Environmental Protection Agency ("EPA") has a limited role relegated to waste management.

Table 20.1 Required Environmental Permits and Plans

Agency	Permit	Description	Citation	When Required
Federal				
Bureau of Alcohol, Tobacco & Firearms	Federal Explosives License	Permit required before purchasing explosives from in state or out of state source. License required before manufacturing, selling, or importing explosives.	27 CFR 555	Required prior to blasting

Agency	Permit	Description	Citation	When Required
		• License is required for persons mixing two component explosives; • Storage requirements apply to all persons storing explosives even if no license or permit is held.		
	Mining	1. Notice Level Operations may not exceed 5 acres. 2. All operations on public lands that disturb the surface require a Plan of Operations will require an environmental assessment and posting of a reclamation bond.	43 CFR 3809	Applicable on small portion of deposit at Strong & Harris
US Fish & Wildlife Service	Incidental Take Permit	Mining activities that may affect species listed as endangered or threatened need to conduct studies to identify any targeted species and to apply for a permit to conduct their activities. Any identified threatened or endangered species identified in pre-mining surveys would need to be mitigated before mining could proceed.	Endangered Species Act & Amendments Section 10	As needed
State of Arizona				
Arizona Department of Environmental Quality				
Air Quality Division	Air Quality Control Permit	Ensures air pollutants from any source does not exceed the National Ambient Air Quality Standards	ARS §49-402	Prior to disturbance
Groundwater Section	Aquifer Protection Permit	Covers surface impoundments, solid waste disposal facilities, mine tailings piles and ponds, mine leaching operations. This permit requires designs for the proper management of process facilities, ponds, tailings impoundments, and includes monitoring requirements to ensure compliance with the permit.	AAC R18-9 Articles 1 - 4	Prior to disturbance
	Reclamation & Closure Plan for Facilities covered by APP	Reclamation plan; estimated cost of executing reclamation plan and surety bond. The reclamation plan includes reclamation activities and post-closure monitoring and bonding estimate must be approved by the agencies and the bond must be posted prior to commencement of construction.	AAC R18-9 Articles 1 - 4	Prior to disturbance

Agency	Permit	Description	Citation	When Required
Waste Management Division				
	EPA ID Number	Generators of hazardous waste must have an EPA ID prior to offering the waste for shipment.	ARS §49-922	Prior to construction
	Pollution Prevention Plan	Plan identifying opportunities to reduce waste.	ARS §49-961 thru 973	Annually
	Toxic Release Inventory	Submit Form R for quantity of copper in waste rock.	40 CFR 372	Annually
Arizona Dept of Water Resources				
Arizona State Mine Inspector	Mined Land Reclamation Plan and Bond	Exploration and mining activities on private land with greater than 5 acres disturbance. Does not include facilities covered in Aquifer Protection Permit..	AAC R11-2-101 thru 822	Required prior to start of operations
Arizona Department of Agriculture	Notice of Intent to Clear Land	Ensures enforcement of Arizona Native Plant Law's	ARS §3-904	60 days prior to disturbance
Arizona Game and Fish Department		Ascertain whether or not the mining operation would endanger fish and game habitat, etc	AAC Title 12	
State Historic Preservation Office		Submit a legal description with map of the area to be disturbed SHPO can inform applicants whether work will occur in a state designated historic district	ARS §43-861	Only applies to thin strip of BLM land

21.0 CAPITAL AND OPERATING COSTS (ITEM 21)

The author has compiled the capital and operating costs based on recent similar projects, as well as inputs from Excelsior and their metallurgical consultants.

21.1 Capital Cost Estimate

Table 21.1 shows the project capital cost estimate which includes a total of $326.5 million in initial capital and $37.4 million in sustaining capital. This totals $363.8 million through the life of mine. The majority of the sustaining capital is for the flotation plant construction which occurs in year two.

Process capital is considered conceptual in nature and has been scaled from other analogue projects such as Tres Mares in Mexico, Skorpion in Namibia and San Manuel in Arizona. None are identical, but for the purpose of a PEA are considered suitable to provide an initial cost estimate. Given the limited testwork supporting this to date, this approach is considered in line with other estimates for the metallurgical work.

Table 21.1 PEA Capital Cost Summary

		Stong & Harris Capital Summary		
	Units	Initial	Sustaining	Total
Preproduction	K USD	$ 136,104	$ -	$ 136,104
Mining Capital	K USD	$ 3,194	$ 200	$ 3,394
Process Capital	K USD	$ 118,500	$ 83,000	$ 201,500
Buildings and Infrastructure	K USD	$ 3,000	$ -	$ 3,000
Working Capital	K USD	$ 54,149	$ (54,149)	$ -
Contingency & EPCM	K USD	$ 11,500	$ 8,300	$ 19,800
Total Capital	K USD	$ 326,447	$ 37,351	$ 363,798

In addition, a bonding cost of $1.1 million is added to the initial capital for the cash-flow evaluation.

The following subsections summarize the capital costs by category.

21.1.1 Preproduction Capital

Preproduction capital was estimated based on the general and administrative and mining operating costs for the 12-month preproduction period. The general and administrative ("G&A") cost portion is estimated at a full year at a cost of $2.4 million, while the mining portion is estimated to be $133.7 million. For details see the mine operating costs in Section 21.2.1 and G&A costs in Section 21.2.3.

21.1.2 Mining Capital Costs

Mining capital cost is reduced based on the use of a mining contract. Under the contract, the contractor would provide all equipment and personnel to achieve the proposed production rates. Thus, only a minimum of surveying, office equipment, and light vehicles would be required. The total owner mining capital was estimated to be $394,000.

In addition to the owner mining capital, the contractor will require mobilization capital along with a laydown area, power and maintenance pad. This was assumed to be $2,800,000 in year one, followed by $200,000 more in year two. The contractor portion of capital is based on previous similar studies that MDA was involved in.

21.1.3 Process Capital

The process capital estimate was provided by Mr. Robert Bowell. This is shown in Table 21.2. The heap leach costs are consistent with those that are reported from the nearby Johnson Camp Mine.

Table 21.2 Process Capital Costs

Facility	$ Unit cost in millions	Description
	Heap Leach	
SX-EW heap	$75	Western Mining Engineering Cost Service Guide
Zn circuit	$35	Assumed based on Western Mining Engineering Cost Service Guide
	Sulfide mill	
Concentrator	$83	Western Mining Engineering Cost Service Guide
Power capex	$5	Move power line electrical capital
Other	$3.5	Stockpiles, ore storage, civil infrastructure
Total	**$201.5**	

21.1.4 Other Capital Costs

Other capital costs assume $3.0 million for buildings and other infrastructure. Additional details will require additional studies.

In addition, working capital was included based on one quarter of a year of operating costs, which is returned to the cash flow at the end of the mine life.

21.2 Operating Cost Estimate

Table 21.3 shows the PEA operating cost summary and is followed by subsections discussing the costs.

Table 21.3 PEA Operating Cost Summary

	K USD LOM Cost	$/ton Processed	$/oz CuEq
Mining	$ 687,198	$ 15.24	$ 1.06
Process	$ 353,286	$ 7.83	$ 0.55
Treatment, Transport, and Penalties	$ 75,787	$ 1.68	$ 0.12
G&A	$ 14,358	$ 0.32	$ 0.02
Site-Reclamation	$ 11,000	$ 0.24	$ 0.02
Total Operating Costs	$ 1,141,629	$ 25.32	$ 1.76

21.2.1 Mine Operating Cost

Mine operating costs have been estimated based on owner personnel and supplies, along with contract mining costs based on previous similar studies. The personnel costs are based on salary and hourly personnel to manage the contractor, provide surveying support for the mine, and provide mine planning and grade control services for the operations. This included $839,000 per year in salaries and hourly wages, and includes 38% burdens for benefits and bonuses.

A total of $388,000 per year was added to the owner's mining costs for general supplies, maintenance, light vehicles, software maintenance and support, and outside services.

Contract mining costs assume $2.46/ton costs for both waste and material processed, and is based on actual contractor quotations received for similar projects. In addition, a rehandle cost of $0.25/ton was added and applied to all tonnages of flotation rehandle from the stockpile schedule.

The mine operating costs are shown in Table 21.4 and total $687.2 million for the life of mine.

Table 21.4 Mine Operating Cost Estimates

Mining Cost After Preproduction	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Total
Total Tonnage Mined (Ex Pre-Prod)	K USD	-	54,750	54,900	54,750	47,100	36,500	27,063	-	275,062
Owner Mining Personnel	K USD	$ -	$ 839	$ 839	$ 839	$ 839	$ 839	$ 839	$ -	$ 5,031
Owner Supplies and Misc.	K USD	$ -	$ 388	$ 388	$ 388	$ 388	$ 388	$ 388	$ -	$ 2,331
Total Owners Mining Costs	K USD	$ -	$ 1,227	$ 1,227	$ 1,227	$ 1,227	$ 1,227	$ 1,227	$ -	$ 7,362
Contractor Mining Cost	K USD	$ -	$ 134,807	$ 135,255	$ 135,079	$ 116,089	$ 90,521	$ 67,238	$ 847	$ 679,836
Total Mine Operating Cost	K USD	$ -	$ 136,034	$ 136,482	$ 136,306	$ 117,316	$ 91,748	$ 68,465	$ 847	$ 687,198
Owner Mining Personnel	$/t	$ -	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ -	$ 0.02
Owner Supplies and Misc.	$/t	$ -	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ -	$ 0.01
Total Owners Mining Costs	$/t	$ -	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.05	$ -	$ 0.03
Contractor Mining Cost	$/t	$ -	$ 2.46	$ 2.46	$ 2.47	$ 2.46	$ 2.48	$ 2.48	$ -	$ 2.47
Total Mine Operating Cost	$/t	$ -	$ 2.48	$ 2.49	$ 2.49	$ 2.49	$ 2.51	$ 2.53	$ -	$ 2.50

21.2.2 Process Operating Costs

Process operating costs were taken from analogue studies and supplemented by onsite costs from Excelsior's Gunnison SX-EW and copper cathode production.

21.2.3 General and Administrative Costs

G&A costs have been estimated based on personnel, supplies, and expenses. G&A costs are minimized by using personnel from other on-going Excelsior operations in the vicinity for site management, environmental, safety, accounting and other functions at Strong and Harris. Thus, only personnel for janitorial services at $62,000 per year are considered under G&A for the PEA. This includes a 28% burden.

Other G&A costs are based on supplies and expenses as shown in Table 21.5. A total life of mine cost of $16.8 million was estimated for G&A, or about $2.9 million per year. Note that year -1 G&A shown in Table 21.5 is capitalized and not included in the operating costs for the cash-flow analysis.

Table 21.5 General & Administrative Costs

Personnel Costs	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Total
Janitor	K USD	62	62	62	62	62	62	62	-	433
G&A Expenses										
Supplies & General Maintenance	K USD	120	120	120	120	120	120	120	-	840
Utilities (Power , water, and heating)	K USD	120	120	120	120	120	120	120	-	840
Land Holdings	K USD	240	240	240	240	240	240	240	-	1,680
Off Site Overhead	K USD	-	-	-	-	-	-	-	-	-
Legal, Audits, Consulting, MSHA	K USD	120	120	120	120	120	120	120	-	840
Computers, IT, Internet, Software, Hardware	K USD	136	136	136	136	136	136	136	-	953
Environmental, Montoring Wells, Reporting	K USD	120	120	120	120	120	120	120	-	840
Bonding Interest Carry	K USD	495	495	495	495	495	495	495	-	3,465
Donations, Dues, Public Relations	K USD	30	30	30	30	30	30	30	-	210
Insurance (Excluding Workmans Comp)	K USD	739	739	739	739	739	739	739	-	5,173
Travel, Lodging, Meals, Entertainment	K USD	60	60	60	60	60	60	60	-	420
Telephones, Computers, Cell Phones	K USD	24	24	24	24	24	24	24	-	168
Small Tools, Janitorial, Safety Supplies	K USD	85	85	85	85	85	85	85	-	595
Equipment Rentals	K USD	24	24	24	24	24	24	24	-	168
Access Road Maintenance	K USD	18	18	18	18	18	18	18	-	126
Total General G&A Costs	K USD	2,331	2,331	2,331	2,331	2,331	2,331	2,331	-	16,318
Total G&A	K USD	2,393	2,393	2,393	2,393	2,393	2,393	2,393	-	16,752

22.0 ECONOMIC ANALYSIS (ITEM 22)

MDA created the cash-flow model used for the economic analysis based on the production schedule and resulting revenue stream along with the costs presented. MDA applied tax considerations based on inputs provided by Excelsior.

The PEA is preliminary in nature and 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 resources. There is no certainty that the conclusions reached in the PEA will be realized. Mineral resources that are not mineral reserves do not have the demonstrated economic viability.

The PEA economic evaluation results in:

- Mine life of about seven years;
- Approximately 54 million tons processed;
- Average of 0.56% copper and 0.68% zinc grades processed;
- $328 million in initial capital costs;
- Operating costs of $1.76 per pound of equivalent copper;
- Average annual production of 62 million pounds of copper and 82 million pounds of zinc;
- $325,466,000 pre-tax NPV (8%);
- $186,958,000 after-tax NPV (8%);
- 19% IRR; and
- 3.1 year payback on initial investment.

22.1 Economic Assumptions

The economic assumptions include the costs as shown in Section 21.0, recoveries, and metal prices. Basic assumptions include:

- Copper heap leach recovery of 92.3% and flotation recovery of 80.1%;
- Zinc heap leach recovery of 83.3% and flotation recovery of 69.7%;
- Silver recovery of 0% for both leach and flotation processes;
- Metal prices of $3.50/pound copper and $1.28/pound zinc;
- Depreciation-based depletion of equivalent copper produced by year;
- Federal tax rate increase by the current administration from 21% to 28%;
- Arizona income tax rate of 4.9%; and

- 1.49% Arizona property tax applied on depreciated asset value.

22.2 Cash-Flow Model Physicals

The mining and process physicals used for the economic analysis were summarized based on the mine and process production schedules and are shown in Table 22.1. The resulting total copper equivalent metal produced is 648,007,000 pounds.

Table 22.1 PEA Cash-Flow Model Physicals

Mine Production	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Mined Above Cog	K Tons	7	7,148	12,856	8,927	8,787	7,846	8,058	-	-	53,629
	Cu %	0.214	0.463	0.631	0.484	0.561	0.524	0.634	-	-	0.557
	K Cu Lbs	28	66,120	162,169	86,429	98,510	82,296	102,248	-	-	597,800
	Zn %	0.04	0.55	0.77	0.57	0.74	0.66	0.72	-	-	0.68
	K Zn Lbs	5	78,574	197,221	102,573	130,488	102,826	115,508	-	-	727,195
	oz Ag/ton	0.065	0.093	0.145	0.125	0.131	0.150	0.147	-	-	0.134
	K Ozs Ag	0	664	1,868	1,116	1,155	1,181	1,184	-	-	7,168
Total Waste Mined	K Tons	53,800	47,602	42,044	45,823	38,313	28,654	19,004	-	-	275,240
Total Mined	K Tons	53,807	54,750	54,900	54,750	47,100	36,500	27,063	-	-	328,869
Strip Ratio	W:O	8,273.24	6.66	3.27	5.13	4.36	3.65	2.36			5.13
Rehandle	K Tons	-	7	323	1,098	413	2,446	2,176	3,147	-	9,609
	Cu %	-	0.214	1.080	0.927	1.014	0.591	0.665	0.654	-	0.701
	K Cu Lbs	-	28	6,972	20,359	8,376	28,914	28,925	41,143	-	134,717
	Zn %	-	0.04	1.24	1.11	1.21	0.72	0.82	0.78	-	0.85
	K Zn Lbs	-	5	7,984	24,332	10,011	35,305	35,830	49,113	-	162,580
	oz Ag/ton	-	0.065	0.147	0.208	0.213	0.140	0.164	0.143	-	0.157
	K Ozs Ag	-	0	47	229	88	343	356	449	-	1,513
Process Production											
Processed as Leach	K Tons	-	5,908	7,220	7,200	7,200	7,200	7,220	3,147	-	45,095
	Cu %	-	0.423	0.522	0.395	0.477	0.445	0.558	0.654	-	0.484
	K Lbs Cu	-	49,972	75,387	56,905	68,743	64,034	80,512	41,143	-	436,696
	K Lbs Cu Prod	-	27,097	52,520	42,154	46,232	42,319	58,157	39,526	-	308,004
	Zn %	-	0.518	0.645	0.501	0.660	0.577	0.622	0.780	-	0.603
	K Lbs Zn	-	61,147	93,162	72,115	95,094	83,093	89,837	49,113	-	543,561
	K Lbs Zn Prod	-	38,396	73,194	65,552	76,172	70,330	71,662	52,044	-	447,351
	oz Ag/ton	-	0.09	0.12	0.11	0.12	0.13	0.14	0.14	-	0.12
	K Ozs Ag	-	532	894	810	876	941	1,006	449	-	5,509
	K Ozs Ag Prod	-	-	-	-	-	-	-	-	-	-
Processed for Flotation	K Tons	-	-	1,805	1,800	1,768	1,800	1,361	-	-	8,535
	Cu %	-	-	1.139	0.972	0.943	0.787	0.857	-	-	0.944
	K Lbs Cu	-	-	41,099	34,989	33,353	28,320	23,343	-	-	161,104
	K Lbs Cu Prod	-	-	26,778	30,898	26,591	22,996	21,423	358	-	129,044
	Zn %	-	-	1.319	1.008	1.106	0.877	1.067	-	-	1.076
	K Lbs Zn	-	-	47,606	36,295	39,107	31,568	29,058	-	-	183,634
	K Lbs Zn Prod	-	-	26,496	29,107	26,221	22,523	23,453	193	-	127,993
	oz Ag/ton	-	-	0.20	0.20	0.19	0.19	0.20	-	-	0.19
	K Ozs Ag	-	-	361	353	329	345	271	-	-	1,660
	K Ozs Ag Prod	-	-	-	-	-	-	-	-	-	-
Cu Equivelant	K Lbs CuEq Prod	-	41,175	115,851	107,760	110,368	99,360	114,455	59,038	-	648,007

22.3 Tax Considerations

Tax considerations include depreciation of initial assets of $227,694,000 in capital (including mining, processing, buildings and infrastructure, and contingency and construction initial capital). A 1.49%

property tax is applied to the depreciated value of the assets. The 4.9% Arizona state income tax is applied to the taxable profit and a 28% federal tax rate was applied to taxable income after the Arizona taxes were applied.

The resulting taxes are shown in Table 22.2. When adding property, severance, and income taxes, a total Arizona taxes cost of $56.8 million is estimated through the life of the mine. A total of $178.2 million is estimated for federal taxes.

22.4 Cash-Flow Model

The PEA cash-flow model is shown in Table 22.3. This results in a net, after-tax, cash-flow of $461.0 million, or $187.0 million NPV at 8%.

Table 22.2 PEA Tax Considerations

	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Pre-Tax Cash Flow	K USD	$ (327,547)	$ (123,340)	$ 176,719	$ 149,639	$ 178,134	$ 165,679	$ 242,262	$ 176,327	$ (3,667)	$ 51,362	$ -	$ 685,570
Depreciation and Depletion													
Recoverable CuEq (Start of Year)	K Ozs AuEq			648,007	532,157	424,396	314,028	214,668	100,213	41,175	41,175	41,175	
Produced CuEq	K Ozs AuEq	-	41,175	115,851	107,760	110,368	99,360	114,455	59,038	-	-	-	
Depreciation Factor (LOM)	%	0%	6%	18%	17%	17%	15%	18%	9%	0%	0%	0%	
Capital for Depreciation	K USD	$ 227,694											$ 227,694
Depreciation of Orig Capex	K USD	$ -	$ 14,468	$ 40,707	$ 37,864	$ 38,780	$ 34,913	$ 40,217	$ 20,744	$ -	$ -	$ -	$ 227,694
Taxable Income After Depreciation	K USD	$ -	$ 157,161	$ 175,888	$ 178,343	$ 157,784	$ 136,737	$ 99,232	$ 1,457	$ 3,667	$ 3,667	$ -	$ 913,935
Assesed Value (Capital less Depreciation)	K USD	$ 227,694	$ 213,226	$ 172,519	$ 134,654	$ 95,874	$ 60,961	$ 20,744	$ -	$ -	$ -	$ -	
Property Tax	K USD	$ -	$ 3,169	$ 2,564	$ 2,001	$ 1,425	$ 906	$ 308	$ -	$ -	$ -	$ -	$ 10,372
Severence Tax	K USD	$ -	$ -	$ 2,209	$ 1,870	$ 2,227	$ 2,071	$ 3,028	$ 2,204	$ -	$ -	$ -	$ 13,610
Operating Income + Preprod Opex - Prop&Sev Tax	K USD	$ (136,104)	$ (35,008)	$ 171,946	$ 145,768	$ 174,483	$ 162,702	$ 238,926	$ 174,123	$ (3,667)	$ (3,667)	$ -	$ 889,502
Depreciation	K USD	$ -	$ (14,468)	$ (40,707)	$ (37,864)	$ (38,780)	$ (34,913)	$ (40,217)	$ (20,744)	$ -	$ -	$ -	$ (227,694)
Taxable Losses b/f	K USD	$ -	$ (136,104)	$ (185,580)	$ (54,341)	$ -	$ -	$ -	$ -	$ -	$ (3,667)	$ (7,333)	
Taxable Profit	K USD	$ -	$ -	$ -	$ 53,562	$ 135,703	$ 127,790	$ 198,709	$ 153,379	$ -	$ -	$ -	$ 669,142
Arizona State Income	K USD	$ -	$ -	$ -	$ 2,625	$ 6,649	$ 6,262	$ 9,737	$ 7,516	$ -	$ -	$ -	$ 32,788
US Federal Corporate Tax	K USD	$ -	$ -	$ -	$ 14,262	$ 36,135	$ 34,028	$ 52,912	$ 40,842	$ -	$ -	$ -	$ 178,179
Taxable Losses Carried Forward	K USD	$ (136,104)	$ (185,580)	$ (54,341)	$ -	$ -	$ -	$ -	$ -	$ (3,667)	$ (7,333)	$ (7,333)	

Table 22.3 PEA Cash Flow Model

Revenues	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Gross Revenue - Cu Leach	K USD	$ -	$ 94,838	$ 183,819	$ 147,538	$ 161,812	$ 148,116	$ 203,550	$ 138,342	$ -	$ -	$ -	$ 1,078,014
Cu Refining - Leach	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Gross Revenue - Zn Leach	K USD	$ -	$ 49,274	$ 93,932	$ 84,126	$ 97,755	$ 90,257	$ 91,967	$ 66,790	$ -	$ -	$ -	$ 574,100
Zn Refining - Leach	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Gross Revenue - Ag Leach	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Ag Refining - Leach	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Gross Revenue - Cu Flot	K USD	$ -	$ -	$ 93,723	$ 108,144	$ 93,070	$ 80,485	$ 74,979	$ 1,254	$ -	$ -	$ -	$ 451,655
Cu Refining - Flot	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Gross Revenue - Zn Flot	K USD	$ -	$ -	$ 34,003	$ 37,355	$ 33,650	$ 28,904	$ 30,098	$ 247	$ -	$ -	$ -	$ 164,258
Zn Refining - Flot	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Gross Revenue - Ag Flot	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Ag Refining - Flot	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Royalty	K USD	$ -	$ 4,323	$ 12,164	$ 11,315	$ 11,589	$ 10,433	$ 18,882	$ 8,104	$ -	$ -	$ -	$ 76,810
Total Revenues	K USD	$ -	$ 139,789	$ 393,314	$ 365,846	$ 374,699	$ 337,329	$ 381,711	$ 198,528	$ -	$ -	$ -	$ 2,191,216
Operating Costs													
Mining	K USD	$ -	$ 136,034	$ 136,482	$ 136,306	$ 117,316	$ 91,748	$ 68,465	$ 847	$ -	$ -	$ -	$ 687,198
Process	K USD	$ -	$ 33,203	$ 61,693	$ 61,524	$ 61,153	$ 61,524	$ 56,502	$ 17,688	$ -	$ -	$ -	$ 353,286
Treatment, Transport, and Penalties	K USD	$ -	$ -	$ 16,028	$ 15,984	$ 15,702	$ 15,984	$ 12,089	$ -	$ -	$ -	$ -	$ 75,787
G&A	K USD	$ -	$ 2,393	$ 2,393	$ 2,393	$ 2,393	$ 2,393	$ 2,393	$ -	$ -	$ -	$ -	$ 14,358
Site-Reclamation	K USD								$ 3,667	$ 3,667	$ 3,667		$ 11,000
Total Operating Costs	K USD	$ -	$ 171,629	$ 216,595	$ 216,207	$ 196,564	$ 171,649	$ 139,449	$ 22,201	$ 3,667	$ 3,667	$ -	$ 1,141,629
Net Operating Cash-flow	***K USD***	$ -	$ (31,840)	$ 176,719	$ 149,639	$ 178,134	$ 165,679	$ 242,262	$ 176,327	$ (3,667)	$ (3,667)	$ -	$ 1,049,587
Capital Cost													
Pre-Production	K USD	$ 136,104	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 136,104
Mining Capital	K USD	$ 3,194	$ 200	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,394
Process Capital	K USD	$ 118,500	$ 83,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 201,500
Buildings and Infrastructure	K USD	$ 3,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,000
Bonding	K USD	$ 1,100									$ (880)		$ 220
Working Capital	K USD	$ 54,149									$ (54,149)		$ -
Contingency & EPCM	K USD	$ 11,500	$ 8,300	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 19,800
Total Capital	K USD	$ 327,547	$ 91,500	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (55,029)	$ -	$ 364,018
Net Pre-Tax Cash Flow	***K USD***	$ (327,547)	$(123,340)	$ 176,719	$ 149,639	$ 178,134	$ 165,679	$ 242,262	$ 176,327	$ (3,667)	$ 51,362	$ -	$ 685,570
Cumulative Pre-Tax Cash-Flow	***K USD***	$ (327,547)	$(450,886)	$ (274,168)	$ (124,529)	$ 53,606	$ 219,285	$ 461,547	$ 637,874	$ 634,208	$ 685,570	$ -	
Severence Tax	K USD	$ -	$ -	$ 2,209	$ 1,870	$ 2,227	$ 2,071	$ 3,028	$ 2,204	$ -	$ -	$ -	$ 13,610
Depreciation	K USD	$ -	$ 14,468	$ 40,707	$ 37,864	$ 38,780	$ 34,913	$ 40,217	$ 20,744	$ -	$ -	$ -	$ 227,694
Net Taxable After Depreciation	K USD	$ -	$ -	$ 136,012	$ 111,775	$ 139,354	$ 130,766	$ 202,045	$ 155,583	$ -	$ -	$ -	$ 875,535
Taxable Losses b/f	K USD	$ -	$(136,104)	$ (185,580)	$ (54,341)	$ -	$ -	$ -	$ -	$ -	$ (3,667)	$ (7,333)	
Az State Income Tax	K USD	$ -	$ -	$ -	$ 2,625	$ 6,649	$ 6,262	$ 9,737	$ 7,516	$ -	$ -	$ -	$ 32,788
US Federal Corporate Tax	K USD	$ -	$ -	$ -	$ 14,262	$ 36,135	$ 34,028	$ 52,912	$ 40,842	$ -	$ -	$ -	$ 178,179
Net Taxes Paid	K USD	$ -	$ -	$ 2,209	$ 18,757	$ 45,011	$ 42,360	$ 65,677	$ 50,561	$ -	$ -	$ -	$ 224,576
Net After-Tax Cash Flow	***K USD***	$ (327,547)	$(123,340)	$ 174,510	$ 130,882	$ 133,123	$ 123,319	$ 176,585	$ 125,766	$ (3,667)	$ 51,362	$ -	$ 460,993
Cumulative After-Tax Cash-Flow	***K USD***	$ (327,547)	$(450,886)	$ (276,377)	$ (145,495)	$ (12,372)	$ 110,947	$ 287,532	$ 413,298	$ 409,631	$ 460,993	$ 460,993	

	%	19%
LOM After-Tax Cash Flow	K USD	$ 460,993
After-Tax NPV(5%)	K USD	$270,718
After-Tax NPV(8%)	K USD	$186,958
After-Tax NPV(10%)	K USD	$140,606
After-Tax Payback (From start of Prod)	Yrs	3.10

22.5 Sensitivities

Sensitivity tables were created based on metal prices, operating costs, and capital costs. These are presented in Table 22.4, Table 22.5, and Table 22.6, respectively, with PEA economic model values highlighted in the tables.
Figure 22.1 shows the sensitivities graphically. PEA economic model values are highlighted in the tables.

As shown in the graph and tables, the project is most sensitive to changes in revenues, which is similar to other metal projects. It is least sensitive to changes in capital costs as shown by the flatter slope in the graph.

Table 22.4 Metal Price Sensitivity

		After Tax Sensitivity - Metal Price (K USD)				
Cu Price	Zn Price	Undisc. CF	NPV 5%	NPV 8%	NPV 10%	IRR
$ 2.50	$ 0.92	$ 37,333	$ (70,617)	$ (115,261)	$ (138,904)	1%
$ 2.75	$ 1.01	$ 143,248	$ 16,703	$ (37,078)	$ (66,122)	6%
$ 3.00	$ 1.10	$ 249,163	$ 102,408	$ 38,999	$ 4,354	10%
$ 3.25	$ 1.19	$ 355,078	$ 186,894	$ 113,438	$ 73,004	15%
$ 3.50	$ 1.28	$ 460,993	$ 270,718	$ 186,958	$ 140,606	19%
$ 3.75	$ 1.38	$ 566,908	$ 354,419	$ 260,306	$ 208,012	23%
$ 4.00	$ 1.47	$ 672,823	$ 437,846	$ 333,264	$ 274,963	27%
$ 4.25	$ 1.56	$ 778,738	$ 521,058	$ 405,913	$ 341,557	31%
$ 4.50	$ 1.65	$ 884,584	$ 604,205	$ 478,500	$ 408,091	34%

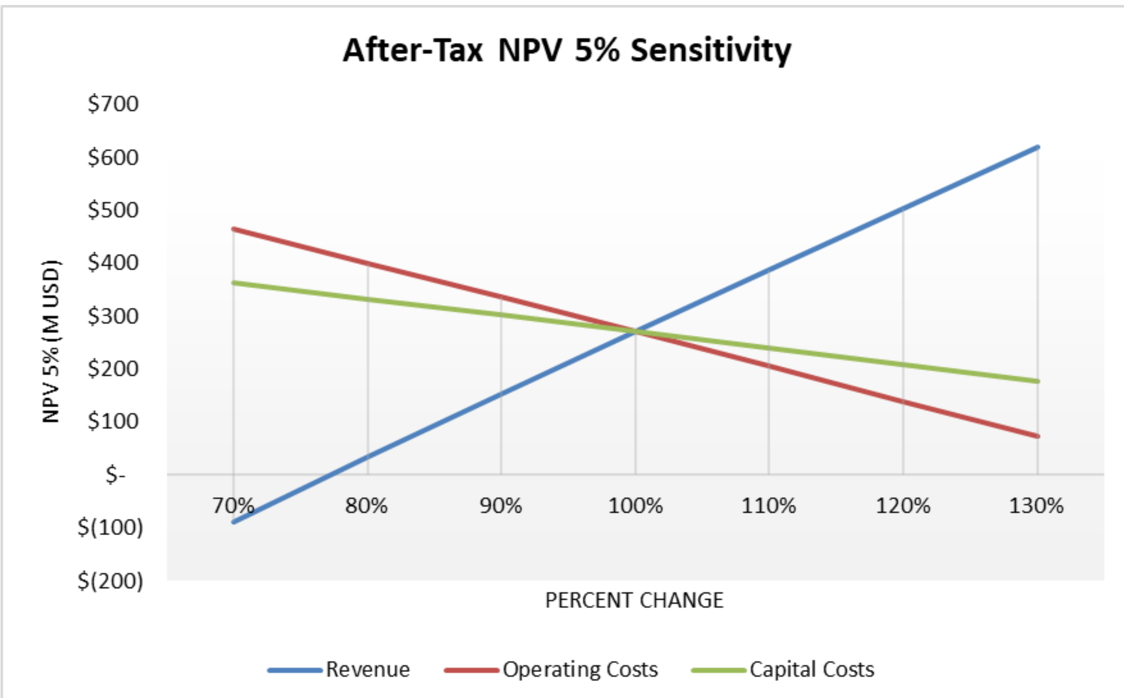
Table 22.5 Operating Cost Sensitivity

	After Tax Sensitivity - Operating Costs (K USD)				
% of Base	Undisc. CF	NPV 5%	NPV 8%	NPV 10%	IRR
70%	$ 693,556	$ 465,238	$ 362,781	$ 305,376	30%
80%	$ 616,126	$ 400,717	$ 304,589	$ 250,921	26%
90%	$ 538,570	$ 335,875	$ 245,994	$ 196,018	23%
100%	$ 460,993	$ 270,718	$ 186,958	$ 140,606	19%
110%	$ 383,416	$ 205,482	$ 127,813	$ 85,070	15%
120%	$ 305,839	$ 139,586	$ 67,749	$ 28,485	12%
130%	$ 228,263	$ 73,116	$ 6,911	$ (28,966)	8%

Table 22.6 Capital Cost Sensitivity

	After Tax Sensitivity - Capital Costs (K USD)				
% of Base	Undisc. CF	NPV 5%	NPV 8%	NPV 10%	IRR
70%	$ 559,257	$ 364,303	$ 277,943	$ 229,936	28%
80%	$ 526,503	$ 333,108	$ 247,615	$ 200,160	24%
90%	$ 493,748	$ 301,913	$ 217,286	$ 170,383	21%
100%	$ 460,993	$ 270,718	$ 186,958	$ 140,606	19%
110%	$ 428,239	$ 239,523	$ 156,630	$ 110,829	17%
120%	$ 395,484	$ 208,328	$ 126,301	$ 81,052	15%
130%	$ 362,729	$ 177,133	$ 95,973	$ 51,275	13%

Figure 22.1 PEA Sensitivity Graph



23.0 ADJACENT PROPERTIES (ITEM 23)

The author has no information to report regarding adjacent properties that is relevant to the resource estimate and PEA described in this report.

24.0 OTHER RELEVANT DATA AND INFORMATION (ITEM 24)

The author is not aware of any other data or information relevant to the mineral resource estimate and PEA described in this report.

25.0 INTERPRETATION AND CONCLUSIONS (ITEM 25)

The authors have reviewed the data from the Strong and Harris project. Based on the work completed or supervised by the authors, it is the opinion of the authors that the project data are of sufficient quality for the modeling, estimation, and classification of copper, zinc, and silver resources disclosed in this report. Furthermore, the authors are unaware of any significant risks or uncertainties that could reasonably be expected to affect the reliability of the current mineral resources.

The deposit was discovered in the late 1960s and was subsequently drilled by multiple companies through the early 1970s, most recently Superior Oil, who had other mineral interests in the district at the time. There has been no historical production at Strong and Harris.

The Strong and Harris project copper, zinc, and silver deposit is characterized as distal copper skarn, presumably related to a porphyry-type system. Strong and Harris mineralization occurs as lenses emplaced more-or-less parallel to layering in favorable lithologic units, especially in the Earp Formation and in the diabase sill and surrounding Horquilla Limestone, where intersected by feeder faults.

The project database includes the data from 152 historical core holes drilled between 1954 and 1992 by various operators, for a total of 130,679 feet of drilling. These holes have an average down-hole depth of 860 feet. Excelsior has not drilled additional holes. In 2021, Excelsior conducted a re-sampling program at Strong and Harris to increase the silver assays in the database and verify the historical assay results

The proposed sulfide flotation process has been developed from a combination of limited historical testwork done on district material and analogous successful flotation operations on copper-zinc sulfide ores undertaken elsewhere. Once proper parametric studies have been completed, the proposed approach is expected to be appropriate for Strong and Harris.

The process of leaching and recovering copper and zinc is more conceptual than that for flotation. Both copper heap leaching and copper recovery using SX-EW are widely practiced and provide the basis for this part of the process. Zinc recovery via SX-EW is also established but is much less common. The conceptual part of the leach-SX-EW process is the binary heap leaching of both copper and zinc with the sequential solvent extraction of copper, followed by the recovery of zinc in a parallel zinc SX circuit.

Metal recovery estimates for the PEA were derived from the limited historical testwork and Excelsior's ISR operations in the district. The PEA considers a seven-year, open-pit mining scenario using $3.50 per pound copper and $1.28 per pound zinc prices, contractor mining, throughput of oxide and mixed leach material at 7.2 million tons per year, and mixed and sulfide material to the flotation plant at up to 1.8 million tons per year.

Project capital costs are estimated to total of $326.5 million in initial capital and $37.4 million in sustaining capital. This totals $363.8 million through the life of mine. The majority of the sustaining capital is for the flotation plant construction which occurs in year two.

The PEA economic evaluation results in approximately 54 million tons processed, operating costs of $1.76 per pound of equivalent copper, and average annual production of 62 million pounds of copper and 82 million pounds of zinc. This production is estimated to generate $325,466,000 pre-tax NPV (8%), $186,958,000 after-tax NPV (8%), an IRR of 19% and 3.1 year payback on initial investment. The PEA is preliminary in nature and 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 resources. There is no certainty that the conclusions reached in the PEA will be realized. Mineral resources that are not mineral reserves do not have the demonstrated economic viability.

25.1 Strong and Harris Project Opportunities

Exploration potential for additional bulk-tonnage mineralization at the Strong and Harris project remains significant. Most of the modeled mineralization is open down-dip and, in several areas, along strike as well, which creates the opportunity to expand the presently defined resources that are potentially minable by open-pit methods

The present leach process can be characterized as a straightforward acid leach. However, other studies suggest that the extraction of both copper and zinc can be enhanced if microbial species such as *Thiobacillus ferrooxidans* and/or *Thiobacillus thiooxidans* are also present in the leach liquor.

When flotation is used to treat the transition material, the resulting tailings will likely contain both acid-soluble zinc and copper mineralization. Agitation leaching of these tailings will likely extract these metal values, which could be recovered in the SX-EW circuit erected as part of the heap leach operation.

25.2 Strong and Harris Project Risks

The current understanding of metallurgical performance, characteristics and costs are at best conceptual. The PEA relies on older metallurgical data undertaken using outdated process methods and on more modern, unpublished analogue studies known to the authors. These are considered reasonable at this stage and provide estimates for recovery consistent with other projects. Further metallurgical work is a requirement to move past this stage and there is a risk that recoveries could vary a lot from those used here and reagent consumptions could be different. The recovery of copper, then zinc, in a sequential SX-EW operation represents the biggest uncertainty.

26.0 RECOMMENDATIONS (ITEM 26)

As discussed in Section 25.0, the PEA results for the Strong and Harris project are positive, and there is potential to expand the extents of mineralization of economic interest; the project therefore warrants significant additional investment. Drilling should be a significant component of future expenditures, including infill drilling, to obtain samples for the ongoing metallurgical program, and step-out drilling, focused on both expanding the existing limits of the current project resources and testing targets peripheral to the resources. The authors recommends at least 20,000 feet of infill drilling, 6,000 feet of drilling for metallurgical programs, and at least 6,000 feet of additional step-out and/or exploration drilling for the next phase of development at Strong and Harris. The drilling should support a subsequent resource update.

Accessibility of historical drill holes should be evaluated by Excelsior. If possible, down-hole surveys should be collected to increase the spatial precision of the drill holes.

Test work programs should be developed for both sulfide flotation to recover copper, zinc and silver, and acid leaching to recover copper and zinc. This work should be initiated by collecting representative samples of the oxide, transition and sulfide materials that include life-of-mine composites as well as mineralized material that reflects the three expected grade ranges for the Strong and Harris resources. The composites would be subjected to analytical and mineralogical characterization, with the results used to guide the development of the two processing routes. For the flotation approach the key will involve optimization of recovery to separate copper and zinc concentrates at acceptable grades. For the leaching program, recovery and subsequent separation of copper and zinc will be the main focus, while working to minimize the impact of the expected high level of acid consumption. Optimization of both processing scenarios for all the types of material should continue as the Strong and Harris Project advances.

The continued collection of specific-gravity data from the proposed core drilling programs is highly recommended. The author recommends comprehensive collection of specific-gravity data for all assay samples in upcoming development work.

Geological modeling should be improved with the new drilling data and before the resource model is updated. Modeling should focus on increasing the spatial precision of geological controls on mineralization.

Geotechnical investigations need to be conducted for pit-slope stability, the heap-leach pad, tailings impoundment, waste rock disposal sites, and borrow areas for clay. Following these investigations and analyses of results, preliminary design layouts should be advanced from conceptual designs to refine facility locations and construction estimates.

Estimated costs for the recommended work program outlined above are presented in Table 26.1. This program has an estimated total cost of $4,910,000. The estimated drilling costs are all-inclusive, as they include Excelsior’s labor costs, access and drill-pad construction costs, assaying, etc., in addition to the contractor costs. In addition to the technical programs, the costs include land holding fees, environmental permitting costs, and project-site general and administrative costs.

Table 26.1 Cost Estimate for the Recommended Program

Item	Estimated Cost US$
Exploration Core Drilling (6, 000 feet)	$720,000
Infill Core Drilling (20,000 feet)	$2,500,000
Metallurgical / Infill Core Drilling (6,000 feet)	$720,000
Geological Modeling	$50,000
Land Holding Costs	$20,000
Metallurgy	$500,000
Geotechnical Studies	$200,000
Resource Update and Technical Report	$200,000
Total	$4,910,000

The authors believe that the Strong and Harris project is a project of merit and warrants the proposed program and level of expenditures outlined above.

27.0 REFERENCES (ITEM 27)

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28.0 DATE AND SIGNATURE PAGE (ITEM 28)

Effective Date of report: September 9, 2021

Completion Date of report: October 20, 2021

"Jeffrey Bickel" ("signed" and "sealed")
Jeffrey Bickel, professional designation

Date Signed:
October 20, 2021

" Michael M. Gustin" ("signed" and "sealed")
Michael M. Gustin, professional designation

Date Signed:
October 20, 2021

"Thomas L. Dyer" ("signed" and "sealed")
Thomas L. Dyer, professional designation

Date Signed:
October 20, 2021

"Robert Bowell" ("signed" and "sealed")
Robert Bowell, professional designation

Date Signed:
October 20, 2021

29.0 CERTIFICATE OF QUALIFIED PERSONS (ITEM 29)

JEFFREY BICKEL

I, Jeffrey Bickel, C. P. G., and Registered Geologist (Arizona) do hereby certify that I am currently employed as Senior Geologist by: Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502, a division of RESPEC.

I am the co-author of the report entitled "*Estimated Mineral Resources and Preliminary Economic Analysis, Strong and Harris Copper-Zinc-Silver Project, Cochise County Arizona*", prepared for Excelsior Mining Corporation, with an Effective Date of September 9, 2021. I take co-responsibility for Sections 1, 2, 3, 4, 5, 6, 7, 14, 25 and 26, and full responsibility for Sections 8, 9, 10, 11, 12, 20, 23, 24, and 27 of the Technical Report subject to those issues discussed in Section 3.0.

I graduated with a Bachelor of Science degree in Geological Sciences from Arizona State University in 2010. I am a Certified Professional Geologist (#12050) with the American Institute of Professional Geologists.

I have worked as a geologist continuously for over 10 years since graduation from university. During that time I have been engaged in the exploration, definition, and modeling of multiple copper skarn deposits in North America, and have estimated the mineral resources for such deposits.

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.

I last visited the Strong and Harris project on several occasions between February 12 and March 26, 2021.

I worked as a Senior Geologist for Excelsior from 2010 – 2020. I am independent of Excelsior Mining Corp. 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.

I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

As of the Effective Date of this report, 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.

Dated this 20th day of October, 2021

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

Signature of Qualified Person
Jeffrey Bickel

CERTIFICATE OF QUALIFIED PERSON

MICHAEL M. GUSTIN, CPG

I, Michael M. Gustin, CPG, do hereby certify that I am currently employed as Senior Geologist by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Geology from Northeastern University in 1979 and a Doctor of Philosophy degree in Economic Geology from the University of Arizona in 1990. I have worked as a geologist in the mining industry for more than 40 years. I am a Licensed Professional Geologist in the state of Utah (#5541396-2250), a Licensed Geologist in the state of Washington (#2297), a Registered Member of the Society of Mining Engineers (4037854RM), and a Certified Professional Geologist of the American Institute of Professional Geologists (CPG-11462).

2. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101"). I have previously explored, drilled, and evaluated copper and polymetallic deposits similar to Strong and Harris in Arizona, Nevada, Idaho, Utah, and Mexico, and I have participated in independent mineral resource estimations in accordance with NI 43-101 guidelines for such deposits in Arizona, Nevada, and Utah. I certify that by reason of my education, affiliation with certified professional associations, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

3. I have not visited the Strong and Harris project.

4. I am co-responsible for Sections 7, 8, and 14 of this report titled, "*Estimated Mineral Resources and Preliminary Economic Analysis, Strong and Harris Copper-Zinc-Silver Project, Cochise County Arizona*", prepared for Excelsior Mining Corporation, with an Effective Date of September 9, 2021 (the "Technical Report"), subject to my reliance on other experts identified in Section 3.0.

5. I have had no other involvement with the property or project that is the subject of the Technical Report other than that directly associated with the completion of the Technical Report.

6. I am independent of Excelsior Mining Corporation and all of its subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

7. As of the effective date of this Technical Report, 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.

8. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this this 20th day of October, 2021.
"Michael M. Gustin" ("signed" and "sealed")

Michael M. Gustin

CERTIFICATE OF QUALIFIED PERSON

THOMAS L. DYER, P.E.

I, Thomas L. Dyer, P.E., do hereby certify that:

(1) I am currently employed as Principal Engineer at Mine Development Associates, whose address is 210 S. Rock Blvd., Reno, NV 89502.

(2) I am a co-author of the report entitled "*Estimated Mineral Resources and Preliminary Economic Analysis, Strong and Harris Copper-Zinc-Silver Project, Cochise County Arizona*" prepared for Excelsior Mining Corp. with an Effective Date of September 9, 2021.

(3) I graduated with a Bachelor of Science degree in Mine Engineering from South Dakota School of Mines and Technology in 1996. I am a Registered Professional Engineer in the state of Nevada (#15729) and a Registered Member (#4029995RM) of the Society of Mining, Metallurgy and Exploration.

(4) I have worked as a mining engineer for more than 25 years since my graduation. Relevant experience includes providing mine designs, reserve estimates and economic analyses of precious- and base-metals deposits and industrial minerals deposits in the United States and various countries of the world. During this period I have worked as Chief Engineer of an operating heap leach and mill gold mine in Nevada.

(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 visited the Strong and Harris project and Excelsior's offices on site on March 19th, 2021.

(7) I take responsibility for Sections 1.6, 15, 16, 18, 19, 21 (except for 21.1.3 and 21.2.2) and Section 22 of this report, subject to those issues discussed in Section 3. I take joint responsibility for Sections 1, 24, and 25.

(8) I am independent of Excelsior Mining Corp., and all of their respective subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

(9) I have had no prior involvement with the property that is the subject of this report.

(10) I have read National Instrument 43-101 and those portions of this report for which I am responsible have been prepared in compliance with that Instrument.

(11) As of the effective date of the technical report, to the best of my knowledge, information, and belief, the technical report, or part that I am responsible for, contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this this 20th day of October, 2021
"Thomas L. Dyer" ("signed" and "sealed")
Signature of Qualified Person

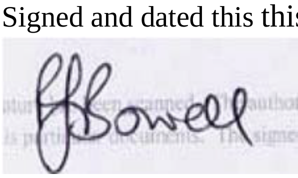
CERTIFICATE OF QUALIFIED PERSON

ROBERT JOHN BOWELL

I, Robert John Bowell, BSc PhD CChem CGeol EGeol, do hereby certify that:

1. I am Corporate Consultant (Geochemistry) of SRK Consulting (UK) Limited, Churchill House, Churchill Way, Cardiff CF10 2HH, UK.
2. I graduated with an honours degree in Geology and Chemistry (class i) in 1987 from Manchester University and a PhD in Geochemistry from Southampton University in 1991. I am a chartered chemist of the Royal Society of Chemistry, a Chartered Geologist of the Geological Society of London and a Registered European professional Geologist of the European Federation of Geologists in good standing in Europe in the areas of Chemistry and Geology.
3. I have worked as a professional Geochemist, Geologist and Geometallurgist for a total of 28 years. My experience includes in situ leaching, heap leaching and geometallurgical testwork on oxide, transitional and sulfide base metal ore types.
4. 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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5. I am responsible for Sections 1.4, 13, 17, 21.1.3 and 21.2.2 of this report titled, "*Estimated Mineral Resources and Preliminary Economic Analysis, Strong and Harris Copper-Zinc-Silver Project, Cochise County Arizona*", prepared for Excelsior Mining Corporation, with an Effective Date of September 9, 2021 (the "Technical Report"), subject to my reliance on other experts identified in Section 3.0.7.
6. I visited the site on the 27th to 29th of September 2021. I have had no prior involvement with the property that is the subject of the Technical Report.
7. I have current involvement with Excelsior Mining Corporation advising on the Gunnison ISR project.
8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Signed and dated this this 20th day of October, 2021



"Robert Bowell" ("signed" and "sealed")

Dr. Robert John Bowell, C.Chem., C.Geol., FIMMM

APPENDIX A

Listing of Unpatented Mining Claims, Strong and Harris Project

CLAIM NAME AND NUMBER	BLM Serial #	TOWNSHIP, RANGE, SECTION
		Mr Twn Rng Sec
BEE R2	403669	14 0150S 0220E 024
BEE R1	403670	14 0150S 0220E 024
BEE R3	403671	14 0150S 0220E 024
BEE R4	403672	14 0150S 0220E 024
BEE R5	403673	14 0150S 0220E 024
BEE R11	403674	14 0150S 0220E 024
BEE R12	403675	14 0150S 0220E 024
BONANZA	403676	14 0150S 0220E 022
BUMBLE BEE	403677	14 0150S 0220E 023
E-5 FRACTION	403692	14 0150S 0220E 013
ECHO NO 1	403693	14 0150S 0220E 024
ECHO R2	403694	14 0150S 0220E 024
ECHO R3	403695	14 0150S 0220E 024
ELEPHANT	403696	14 0150S 0220E 023
LAST CHANCE	403710	14 0150S 0220E 027
LAURA J	403711	14 0150S 0220E 024
PORTLAND	403728	14 0150S 0220E 023
PRIMROSE	403729	14 0150S 0220E 023
PRIMROSE BEE	403730	14 0150S 0220E 023
S-10	403732	14 0150S 0220E 023
S-12	403733	14 0150S 0220E 023
S-14	403734	14 0150S 0220E 023
S-16	403735	14 0150S 0220E 023
S-18	403736	14 0150S 0220E 023
S-26	403737	14 0150S 0220E 024
S-28	403738	14 0150S 0220E 024
S-30	403739	14 0150S 0220E 024
S-32	403740	14 0150S 0220E 024
S-34	403741	14 0150S 0220E 024
ASHLEY	416211	14 0150S 0220E 024
J-TRAVASSOS	416212	14 0150S 0220E 024
N-TRAVASSOS	416213	14 0150S 0220E 024
SUMMERTIME	416214	14 0150S 0220E 023
SUNSET	416215	14 0150S 0220E 023
T-ACKEN	416216	14 0150S 0220E 024
WILDFIRE	416217	14 0150S 0220E 023
BIRD DOG 1	AMC451034	14 0150S 0220E 014
BIRD DOG 2	AMC451035	14 0150S 0220E 014

CLAIM NAME AND NUMBER	BLM Serial #	TOWNSHIP, RANGE, SECTION
BIRD DOG 3	AMC451036	14 0150S 0220E 014
BIRD DOG 4	AMC451037	14 0150S 0220E 014
BIRD DOG 5	AMC451038	14 0150S 0220E 014
BIRD DOG 6	AMC451039	14 0150S 0220E 014
BIRD DOG 7	AMC451040	14 0150S 0220E 014
BIRD DOG 8	AMC451041	14 0150S 0220E 014
BIRD DOG 9	AMC451042	14 0150S 0220E 014
BIRD DOG 10	AMC451043	14 0150S 0220E 014
BIRD DOG 11	AMC451044	14 0150S 0220E 013
BIRD DOG 12	AMC451045	14 0150S 0220E 013
BIRD DOG 13	AMC451046	14 0150S 0220E 013
BIRD DOG 14	AMC451047	14 0150S 0220E 013
BIRD DOG 15	AMC451048	14 0150S 0220E 013
BIRD DOG 16	AMC451049	14 0150S 0220E 013
BIRD DOG 17	AMC451050	14 0150S 0220E 013
BIRD DOG 18	AMC451051	14 0150S 0220E 013
BIRD DOG 19	AMC451052	14 0150S 0220E 013
BIRD DOG 20	AMC451053	14 0150S 0220E 014
BIRD DOG 21	AMC451054	14 0150S 0220E 014
BIRD DOG 22	AMC451055	14 0150S 0220E 014
BIRD DOG 23	AMC451056	14 0150S 0220E 014
BIRD DOG 24	AMC451057	14 0150S 0220E 014
BIRD DOG 25	AMC451058	14 0150S 0220E 014
BIRD DOG 26	AMC451059	14 0150S 0220E 013
BIRD DOG 27	AMC451060	14 0150S 0220E 013
BIRD DOG 28	AMC451061	14 0150S 0220E 013
BIRD DOG 29	AMC451062	14 0150S 0220E 013
BIRD DOG 30	AMC451063	14 0150S 0220E 013
BIRD DOG 31	AMC451064	14 0150S 0220E 013
BIRD DOG 32	AMC451065	14 0150S 0220E 014
BIRD DOG 33	AMC451066	14 0150S 0220E 014
BIRD DOG 34	AMC451067	14 0150S 0220E 014
BIRD DOG 35	AMC451068	14 0150S 0220E 014
BIRD DOG 36	AMC451069	14 0150S 0220E 014
BIRD DOG 37	AMC451070	14 0150S 0220E 014
BIRD DOG 38	AMC451071	14 0150S 0220E 013
BIRD DOG 39	AMC451072	14 0150S 0220E 013
BIRD DOG 40	AMC451073	14 0150S 0220E 013
BIRD DOG 41	AMC451074	14 0150S 0220E 013
BIRD DOG 42	AMC451075	14 0150S 0220E 013
BIRD DOG 43	AMC451076	14 0150S 0220E 013
BIRD DOG 44	AMC451077	14 0150S 0220E 014
BIRD DOG 45	AMC451078	14 0150S 0220E 014
BIRD DOG 46	AMC451079	14 0150S 0220E 014

CLAIM NAME AND NUMBER	BLM Serial #	TOWNSHIP, RANGE, SECTION
BIRD DOG 47	AMC451080	14 0150S 0220E 014
BIRD DOG 48	AMC451081	14 0150S 0220E 014
BIRD DOG 49	AMC451082	14 0150S 0220E 013
BIRD DOG 50	AMC451083	14 0150S 0220E 013
BIRD DOG 51	AMC451084	14 0150S 0220E 013
BIRD DOG 52	AMC451085	14 0150S 0220E 013
BIRD DOG 53	AMC451086	14 0150S 0220E 013
BIRD DOG 54	AMC451087	14 0150S 0220E 013
SURPRISE NO 1	AMC452780	14 0150S 0220E 014
SURPRISE NO 3	AMC452781	14 0150S 0220E 014
SURPRISE NO 5	AMC452782	14 0150S 0220E 014
SURPRISE NO 7	AMC452783	14 0150S 0220E 014
SURPRISE NO 9	AMC452784	14 0150S 0220E 014
SURPRISE NO 19	AMC452785	14 0150S 0220E 013
SURPRISE NO 21	AMC452786	14 0150S 0220E 013
SURPRISE NO 22	AMC452787	14 0150S 0220E 013
SURPRISE NO 23	AMC452788	14 0150S 0220E 013
SURPRISE NO 37	AMC452789	14 0150S 0220E 014
SURPRISE NO 38	AMC452790	14 0150S 0220E 014
SURPRISE NO 39	AMC452791	14 0150S 0220E 014
SURPRISE NO 40	AMC452792	14 0150S 0220E 014
SURPRISE NO 46	AMC452793	14 0150S 0220E 013
SURPRISE NO 47	AMC452794	14 0150S 0220E 013
SURPRISE NO 48	AMC452795	14 0150S 0220E 013
SURPRISE NO 55	AMC452796	14 0150S 0220E 014
SURPRISE NO 56	AMC452797	14 0150S 0220E 014
SURPRISE NO 57	AMC452798	14 0150S 0220E 014
SURPRISE NO 58	AMC452799	14 0150S 0220E 014
SURPRISE NO 64	AMC452800	14 0150S 0220E 013
SURPRISE NO 65	AMC452801	14 0150S 0220E 013
SURPRISE NO 66	AMC452802	14 0150S 0220E 013

Listing of Patented Mining Claims and Fee Lands, Strong and Harris Project

Parcel 5

Acorn, A-Number One, A-Number Two, Chicago, Cochise, Copper Thread, Johnson, Little Johnnie, Rough Rider, Tenderfoot, and United Fraction patented lode mining claims, Mineral Survey No. 4314

Parcel 6

Blue Lead, North Star, Little Bush, Copper Chief, Southern Cross, Blue Lead Extension, Dwarf, and Esmeralda patented lode mining claims, Mineral Survey No. 3242 Anaconda, Last Chance, Delta, and Sara patented lode mining claims, Mineral Survey No. 1525

Parcel 10

Peabody patented lode mining claim, Lot 39, Mineral Survey No. 286

Parcel 19

Clondike, Blue Jacket, Keystone, Blue Bell, Copper Bell, Dewey, True Blue, and Ross patented lode mining claims, Mineral Survey No. 1717

Parcel 20

382681 v2 Hillside, Pittsburg, and Teaser patented lode mining claims, Mineral Survey No. 3306

Fee Lands

The following parcels of fee land are all situated in Township 15 South, Range 22 East, G&SRB&M, Cochise County, Arizona

Parcel 3

Section 24: Lot 16

Parcel 4

Section 23: Lots 11, 12, 13, and 16
Section 24: Lots 11, 12, and 13 EXCEPT any portion lying within the South Half of the Southeast Quarter of the Northwest Quarter (S1/2SE1/4NW1/4) and the East Half of the Southwest Quarter (E1/2SW1/4) of Section 24, Township 15 South, Range 22 East, G&SRB&M conveyed by Special Warranty Deed dated January 26, 1987 from Cyprus Mines Corporation, Grantor, to David A. Rae, Grantee, recorded in the Cochise County records as Document No. 870102364.