Preliminary Economic Assessment Update Technical Report for the Malku Khota Project Department of Potosi, Bolivia

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Prepared for
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by

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CONTENTS

TABLES

FIGURES

UNITS

The Metric System has been used throughout this report, unless otherwise stated. Tonnes are reported in metric tonnes of 1,000 kilograms. Copper, zinc, and lead grades are in weight percent. The primary monetary units in this PEA are the U.S. Dollar. Silver grades are in grams per metric tonne (g/t) with prices in U.S. dollars per troy ounce. Indium values are in parts per million (ppm) and prices in U.S. dollars per kilogram.

ABBREVIATIONS

The following abbreviations are used in this report.

Abbreviation	Unit or Term
AA	Atomic absorption
Ag	Silver
Ag_Eq	Silver Equivalent
Au	Gold
Bi	Bismuth
BLS	Barren Liquor Solution
^{o}C	Degrees Celsius
CMMK	Compañia Minera Malku Khota
Cu	Copper
cu	cubic
DFCs	Direct Field Costs
DFS	Detailed Feasibility Study
dia	Diameter
EMP	Environmental Management Plan
EPCM	Engineering Procurement and Construction Management
ESIA	Environmental and Social Impact Assessment
^{o}F	Degrees Fahrenheit
ft	Foot
g	Gram
Ga	Gallium
gal	U.S. gallon
G&A	General and Administration
gpm	Gallons per minute
GPS	Global positioning system
gpt or g/t	Grams per tonne
h	Hour
ha	Hectare
hr	Hour
hp	Horsepower
ILS	Intermediate Liquor Solution
In	Indium
in	Inch
kg	Kilogram
km	Kilometer
kWh	Kilowatt hour
l	Liter
l/s	Liters per second
lb	Pound (weight)
LDL	Lower detection limit
LG	Lerchs-Grossman

Abbreviation	Unit or Term
LOI	Letter of intent
LS	Lump sum
M	Million
m	Meter
mamsl	Meters above mean sea level
m2	Square meter
m3	Cubic meter
masl	Meters above sea level
m3/h	Cubic meters per hour
mm	Millimeter
m/s	Meters per second
MW	Megawatt
NGO	Non-Government Organization
NI 43-101	Canadian National Instrument NI 43-101
No.	Number
oz	Ounce(s)
Pb	Lead
PCMS	Process Component Monitoring System
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
PLS	Pregnant Liquor Solution
pH	Acidity
ppm	Parts per million
psi	Pounds per square inch
PVC	Polyvinyl chloride
QA	Quality assurance
QC	Quality control
QP	Qualified Person as defined by NI 43-101
SASC	South American Silver Corp.
Sb	Antimony
SG	Specific gravity
TC/RCs	Treatment and Refining Charges
t	Metric tonne (1,000 kg)
t/m3	Tonnes per cubic meter
tpd	Tonnes per day
tph	Tonnes per hour
Tonne	Metric tonne (1,000 kg)
Ton	Long ton (2,240 lb)
w/o	Waste-to-ore ratio
yr	Year
Zn	Zinc

1 SUMMARY

South American Silver Corp., (SASC) controls extensive exploration mining concessions at Malku Khota in the Department of Potosi, Bolivia through its wholly owned Bolivian subsidiary Compañia Minera Malku Khota S.A. (CMMK). CMMK holds mineral rights to 5,475 hectares at Malku Khota, covering 15 km of projected strike length along prospective silver mineralized sandstone units. Exploration of the concessions is at an advanced stage and, subject to satisfactory feasibility studies and environmental and social impact assessments; SASC intends to develop the Malku Khota property into a producing mine.

A Preliminary Economic Assessment (PEA) for Malku Khota was completed in March 2009 by Pincock Allen & Holt (PAH) based on resource estimates of November 2008 (also by PAH). The 2008 resource estimates and the 2009 PEA are superseded by this technical report.

This PEA update contemplates the construction and operation of a 40,000 tpd open pit acid-chloride heap leach operation. The PEA estimates that approximately 200 million tonnes of leach material would be mined over a 15 year mine life with production of 13.2 million ounces of silver per year for the first 5 years and over 10.5 million ounces per year for the life of the mine. Additionally, the mine is anticipated to produce approximately 80 tonnes of indium and 15 tonnes of gallium per annum. The mine would also annually produce several million pounds of by-product lead, copper and zinc contributing to the project's low operating costs which are projected to be in the lower quartile of primary silver producers.

Table 1-1 Operational and Production Summary

Plant Throughput	40,000	tonnes/ day
LOM Plant Feed	200	M tonnes
Strip ratio	2.23	
Mine life	15	Years
Initial Capital Cost	$411	million
Sustaining Capital	$209	million

	Silver	Indium	Gallium	Copper	Lead	Zinc
Recoveries	73.6%	81.0%	26.9%	84.8%	51.1%	62.0%
1st 5 years (averages)						
Grades	42.42 g/t	7.55 g/t	4.28 g/t	0.023%	0.084%	0,023%
Annual production (recovered metal)	13.2 M ozs	81 tonnes	15,184 kgs	5.64 M lbs	12.48 M lbs	4.42 M lbs
Life of Mine (averages)						
Grades	33.60 g/t	7.35 g/t	3.98 g/t	0.024%	0.085%	0.050%
Annual production (recovered metal)	10.5 M ozs	78.9 tonnes	14,198 kgs	5.87 M lbs	12.76 M lbs	9.04 M lbs
Life-of-Mine Recovered Metal	158 M ozs	1,184 tonnes	212,962 kgs	88 M lbs	191 M lbs	135 M lbs

Economic modeling for the updated PEA demonstrates a robust project with strong operating cash flow, high rates of returns, and low capital and operating costs, particularly on a cost per ounce basis. Three metal price scenarios were selected to show the project's sensitivity to varying metal prices using the same pit model, a Base Case scenario using the approximate 3 year trailing average price for metals, a Middle-Case with approximate 1 year trailing average metal prices and a Recent Case which reflects recent monthly average metals prices. At higher metal prices, updated mine models would likely increase total metal production.

Table 1-2 Financial Summary

Measure	Base Case	Mid-Case	Recent
Metal Prices			
Silver (US$/ounce)	$18.00	$25.00	$35.00
Indium (US$/kg)	$500.00	$570.00	$650.00
Lead (US$/lb)	$0.90	$1.00	$1.20
Zinc (US$/lb)	$0.90	$1.00	$1.10
Copper (US$/lb)	$3.00	$3.70	$4.30
Gallium (US$/kg)	$500.00	$570.00	$730.00
Average Operating Cash Flow	-	-	-
1st 5 years (per year)	$185 M	$287 M	$430 M
Life-of-Mine (per year)	$124 M	$208 M	$325 M
Net Cash Flow (undiscounted)	$1,261 M	$2,528 M	$4,298 M
NPV at 5% discount rate	$704 M	$1,482 M	$2,571 M
NPV at 8% discount rate	$505 M	$1,104 M	$1,942 M
Internal rate of Return (IRR %)	37.7%	63.0%	92.9%
Payback period	27 months	19 months	15 months
1st 5 years			
Silver cash costs before credits (US$/ounce)	$9.70	$9.70	$9.70
Silver cash costs after credits (US$/ounce)	$2.94	$2.01	$0.86
Life of Mine			
Silver cash costs before credits (US$/ounce)	$13.87	$13.87	$13.87
Silver cash costs after credits (US$/ounce)	$5.06	$3.85	$2.39

On the basis of the technical and economic evaluations in this PEA, it is concluded that the project demonstrates sufficient potential to advance to feasibility study.

This PEA includes inferred resources that are considered too speculative geologically to be categorized as Mineral Reserves but give an indication of the potential size of operations that may be possible on the deposit. However, there is no certainty that the results projected in the PEA will be realized and actual results may vary substantially.

1.1 Location

Malku Khota is situated in the eastern part of the Bolivian Altiplano at elevations between approximately 3,800 to 4,580 meters above sea level. The project is located 98 km east-southeast of Oruro, and 85 km south of Cochabamba in a relatively remote area, accessible by an improved dirt road linking the cities of Oruro and Cochabamba. Primary access is from Oruro via Bolivar, Sacaca, and Chiro Khasa.

The climate is typical for the Bolivian Cordillera Oriental and Altiplano with cool to moderate rainy summers and cool, dry winters. Scattered subsistence farm plots and seasonal pastures are found scattered throughout the Malku Khota project area and are utilized by local inhabitants on a seasonal basis.

Figure 1-1 Project Location Maps

Oruro has a capable supply of labor, equipment and service requirements for exploration and mining related activities. The majority of the day labor for road building, sampling technicians, core cutting and support work has been found among the 5 communal districts (Ayllus) in the project area. The project provides some of the only employment opportunities in the area.

1.2 Work since the 2009 PEA

Since the 2008 resource estimate and the 2009 PEA, work has advanced in a number of areas:

- Drilling has progressed to bring the total drilled at Malku Khota to over 40,000m. Resource estimates have been updated based on the additional drilling and are included in this PEA Update Technical Report.
- Considerable progress has been made with metallurgical testing including an extensive acid-chloride and cyanide leach testing program undertaken at SGS Lakefield through 2010. Preliminary geotechnical and hydrodynamic testing demonstrated satisfactory stacking characteristics and structural integrity of heap leach ores. Kinetic tests & acid optimization testing was started in 2010 and is ongoing. Additional cyanidation tests of selected samples were also completed in January 2011. Bench scale acid recovery amenability tests have also been successfully completed. Comminution tests to confirm crushing and grinding indices are in progress as is an engineering cost study into an Indium refinery.
- A number of mining and treatment options have been evaluated at scoping study level with the aim of narrowing the range of alternatives to be studied in the PFS. This work has considered alternatives to heap leaching including vat and agitated leaching, both of which have potential to achieve higher recoveries than can be achieved with heap leaching. The alternatives add milling and additional processing facilities and the studies are too preliminary to report at this time. They are however considered to be potentially attractive and will be progressed in the pre-feasibility phase.
- A preliminary hydrology and hydrogeology study of the property has been completed by Chilean water consultants and a power supply study has been undertaken by Bolivian power consultants. As well, SASC project personnel have visited Bolivia, Chile and Peru gathering information on local engineering and construction capability, and local cost factors for the capital and operating cost estimates. Groundwork has also been laid for environmental and social baseline studies to get underway in the pre-feasibility phase.

- As part of the Company's community relations strategy, community meetings are ongoing to help facilitate participation by the local indigenous communities that surround the Malku Khota project. The Company continues to look for ways to facilitate economic development in the area by employing local workers, utilizing local services and building and improving infrastructure.

1.3 Mineral Resources and Reserves

Drilling in 2010 was primarily focused on confirmation and in-fill drilling, with a focus on increasing the density of drilling in the areas that fall within the pit model. The updated resource estimate expands the Measured and Indicated resources by 60% from the 2009 estimate to 230 million ounces of silver with an additional Inferred resource of 140 million ounces of silver. The 2010 drill program successfully confirmed the geologic model with over 80% of the life-of-mine in-pit silver resources classified in the Measured and Indicated category.

Mineral Resources

Mineral resources have been estimated as of March 30, 2011 for this PEA update by Allan Armitage, Ph.D., P.Geo, of GeoVector Management Inc. The new resource estimate for Malku Khota, which takes account of additional drilling since the 2009 PAH resource estimate, is summarized in Table 1-3.at a 10 g/t silver equivalent cut-off grade.

Table 1-3 Malku Khota Estimated Resources

Resource	Ag (g/t)			In (g/t)		Ga (g/t)		Cu (%)		Pb (%)		Zn (%)	Ag_Eq (g/t)		
	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Tonnes	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Ozs
Limosna Zone															
Measured	10,781,799	40.0	13,853,996	6.3	68	2.8	30	0.01	1,836,252	0.13	30,150,033	0.04	9,405,115	54.4	18,854,749
Indicated	110,769,594	29.3	104,320,806	6.9	763	3.1	338	0.01	13,416,240	0.13	314,792,778	0.08	190,322,407	45.6	162,285,412
Measured + Indicated	121,551,393	30.2	118,174,802	6.8	831	3.0	368	0.01	15,252,493	0.13	344,942,811	0.07	199,727,522	46.4	181,140,161
Inferred	75,728,745	23.0	55,977,228	6.6	501	3.1	238	0.01	8,709,675	0.13	212,159,178	0.10	168,331,885	39.9	96,940,651
Wara Wara Zone															
Measured	10,551,499	26.1	8,845,445	2.7	29	5.7	60	0.02	4,037,605	0.03	7,928,699	0.004	899,715	36.6	12,411,255
Indicated	46,177,497	21.7	32,251,803	2.8	131	5.5	256	0.02	20,564,373	0.03	27,652,170	0.005	5,515,373	32.4	48,068,548
Measured + Indicated	56,728,996	22.5	41,097,247	2.8	160	5.6	316	0.02	24,601,978	0.03	35,580,870	0.005	6,415,088	33.2	60,479,803
Inferred	69,705,696	17.6	39,367,381	2.6	179	4.8	333	0.02	26,733,840	0.02	29,855,831	0.00	7,154,863	26.7	59,849,942
Sucre Zone															
Measured	9,656,149	34.2	10,620,046	9.5	92	5.0	48.3	0.04	8,073,966	0.05	10,586,634	0.03	5,934,260	53.7	16,668,881
Indicated	67,054,896	28.0	60,387,989	6.0	399	5.2	349.2	0.05	72,386,267	0.04	62,204,138	0.02	34,735,943	45.5	98,023,697
Measured + Indicated	76,711,045	28.8	71,008,035	6.4	491	5.2	397.6	0.05	80,460,233	0.04	72,790,772	0.02	40,670,202	46.5	114,692,579
Inferred	34,429,848	22.9	25,370,596	3.2	110	5.2	180.3	0.05	41,117,709	0.03	25,140,170	0.01	10,933,524	38.0	42,052,428
Low Grade Mineralized Halo															
Inferred	50,149,506	12.0	19,312,012	2.9	146	5.0	249.6	0.02	25,520,105	0.09	95,002,428	0.05	59,730,067	26.2	42,265,446
Total Malku Khota Project															
Measured	30,989,448	33.4	33,319,487	6.1	188	4.5	139	0.02	13,947,823	0.07	48,665,367	0.02	16,239,090	48.2	47,943,510
Indicated	224,001,987	27.3	196,960,598	5.8	1,293	4.3	943	0.02	106,366,881	0.07	404,649,086	0.05	230,573,723	42.5	306,119,818
Measured + Indicated	254,991,434	28.1	230,280,085	5.8	1,481	4.3	1,082	0.02	120,314,704	0.07	453,314,453	0.04	246,812,812	43.2	354,109,119
Inferred	230,013,794	18.9	140,027,216	4.1	935	4.3	1,001	0.02	102,081,329	0.07	362,157,607	0.05	246,150,339	32.5	240,292,377

Notes

1. The resource cut-off grade of 10 g/t silver equivalent is based only on the values of silver at $16/oz and indium at $550/kg.
2. The silver equivalent calculation in the above table includes value for Ag, In, Ga, Cu, Pb, and Zn using Base Case metal pricing above.
3. Estimated metal content does not include any consideration of mining, mineral processing, or metallurgical recoveries.

Mineral Reserves

No reserve estimate has been carried out for the PEA because the extent of mineralization is not considered sufficiently defined at this stage to create a reserve. This 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 reserves and there is no certainty that the preliminary assessment will be realized. Mineral resources that are not mineral reserves do not have economic viability.

1.4 Base Case – Acid-Chloride Heap Leach

The Base Case project is a 40,000tpd acid-chloride heap leach project.

- The acid-chloride leach process recovers silver along with indium, lead, zinc, copper, gallium and gold.
- The PEA is based on a conventional shovel and truck open cut mining operation with material hauled initially in 91 tonne trucks, phasing to 136 tonne trucks as the pit deepens. Material will be hauled to a heap leach pad that is located within 2km of the open cut. Waste will be hauled to dumps within 2km of the pit. In this study, mining operations assumed an owner-operated mining fleet.
- Over the 15 year life of mine, approximately 200 million tonnes of leach material will be mined at an average strip ratio of 2.23 to 1. Silver equivalent grades for the first 5 years of production average 58 g/t, with silver and indium grades of 42.42g/t and 7.55g/t respectively with the remaining value from copper, lead and zinc credits. Over the life of mine the silver equivalent grade is approximately 50 g/t, with silver and indium grades that average 33.6g/t and 7.35g/t.
- Run-of-Mine material will be crushed to 1/4" in a three stage crushing circuit and stacked in a heap leach pad where a leaching agent made up of hydrochloric acid mixed with salt and sodium hypochlorite will be applied. The heap leach will be designed as an On/Off pad to receive 40,000 tpd in a single 8 meter lift sized for a 120 day leach cycle. The leach pad design contemplates 6 terraced cells with a total footprint of 540,000 square meters. Pregnant solution will be collected in ponds located downslope of the leach pad before advancing to the metals recovery plant.
- Average leaching recoveries used in the design are 73.6% for silver, 81.0% for indium, 51.1% for lead, 62.0% for zinc, 84.8% for copper and 26.9% for gallium, based on the testing program undertaken at SGS Lakefield Laboratories.
- The metal recovery processes start with acid recovery and proceed through several steps of metals extraction and recovery in three main streams:
  - A cemented product containing silver, gold and copper will be further processed into a silver-gold-copper dore.
  - Indium-gallium hydroxide precipitate will be refined into pure indium and gallium ingots.
  - Separately precipitated lead and zinc sulfides will be refined off site as concentrates.
- Over 15 years of operations, Malku Khota is projected to produce 158 million ounces of silver, 1,184 tonnes of Indium, 191 million lbs of lead, 135 million lbs of zinc, 88 million lbs of copper and 212 tonnes of gallium.
- Using the natural terrain to advantage, residues will be disposed into lined facilities located downstream of the heap leach pads. Maximum use will be made of water recovered from the residue facility, which will be returned for re-use in the process plant.

- Water supply for the project will come from pit dewatering, regional flows and groundwater sources, which have been confirmed in preliminary investigations as sufficient to meet project needs.
- The project has an estimated maximum power requirement of 40MW, which will come from the Bolivian national grid approximately 20km from the site.

A number of mining and treatment options have been evaluated with the aim of narrowing the range of alternatives to be studied in the PFS. This work has considered alternatives to heap leaching including vat and agitated leaching, both of which have potential to achieve higher recoveries than can be achieved with heap leaching. This optimization work is considered to be potentially attractive and will be progressed along with further testwork to confirm crushing and grinding characteristics as part of the PFS level work on the project.

1.5 Facilities and Construction

South American Silver project personnel have met with engineering and construction contractors in Bolivia, Chile and Peru to assess local capability and cost factors for the capital and operating cost estimates. Site construction is expected to take around 18 months during which the construction workforce is expected to peak at over 1,000 people. Once in commercial production, Malku Khota will provide direct employment to over 400 employees and contractors, most of whom will be sourced from surrounding communities. Oruro has a capable supply of labor, equipment and service requirements for exploration and mining related activities.

Preliminary hydrology, hydrogeology and power studies of the property have been completed by experienced contractors in the region. Maximum use will be made of water recovered from the residue facility, which will be returned for re-use in the process plant. Using the natural terrain to advantage, residues will be disposed into lined facilities located downstream of the heap leach pads. Water supply for the project will come from pit dewatering, regional flows and groundwater sources, which have been confirmed in preliminary investigations as sufficient to meet project needs. Groundwork has also been laid for environmental and social baseline studies to get underway in the pre-feasibility phase.

1.6 Capital and Operating Costs

Initial Capital

Mining capital including pre-production capital and initial mining fleet are based on a project-specific pit design and production schedule with equipment selection and quantities based on equipment vendor's recommendations and engineer's experience of similar operations. Costs are based on budget pricing and engineer's cost database where appropriate. Pre-stripping of around 1.75 million tonnes is required. The initial mining fleet includes equipment that will be used to construct the heap leach pad base and for ongoing On/off heap leach materials handling during the production phase.

Process plant capital costs for crushing, leaching and metal recovery facilities are factored estimates based on process flowsheets, mass balances and equipment lists developed specifically for the project based on the metallurgical testwork. Project infrastructure costs for power supply, water supply and roads are budget estimates from consultant's reports and allowances based on recent costs for similar projects constructed in Bolivia.

Leach pads, ponds and residue facilities were estimated based on quantities derived from conceptual layouts overlain on the topographical plans of the site. Unit cost rates were applied using estimator's database. Around 3.5 million cubic meters of cut-to-fill is required during construction to achieve a terraced-level site for the heap leach pad. Costs for this portion are based on the work being undertaken by the mining fleet.

Indirect construction costs for freight, taxes and duties, construction camp and catering, EPCM, commissioning, spares and first fills, and Owner's Costs are included. These are typically factored on Direct Field Costs.

Capital Cost Summary

Mine pre-production	$3.8 M
Mining Fleet	$83.1 M
Process Plant & Infrastructure	$149,9 M
Leach Pads and Residues Facilities	$81.3 M
Indirect Costs	$93.3 M
Total Initial Capital	$411.4 M

Sustaining, Working Capital and Closure Provisions

Sustaining capital is estimated at $169 million in the mine, mainly for mining fleet additions and replacement, and $40 million in the processing facilities, mainly for On/Off heap leach materials handling and progressive residue facility construction, for a life of mine total of $209 million.

Working Capital requirements for the financial model are based on 90 days between production and receipt of payment. Allowances of $10 million for reclamation costs and $2 million salvage value at the end of mine life are included in the financial model.

Operating Costs

Operating Costs have been developed for each major area based on project-specific requirements. Mining and processing costs are based on year-round 24 hour/day operations. Power is estimated at $0.042 per kWh and diesel fuel at $0.51 per liter reflecting Bolivian costs. The project has an estimated maximum power requirement of 40MW, which will come from the Bolivian national grid approximately 20 km from the site. Operational manning is based on approximately 90% of labor being locally-sourced supported by a small expatriate contingent involved in management and training.

Mining costs are based on pit designs, mine production and waste dump schedules developed specifically for the project. Processing costs were developed for reagents on a cost per tonne (of production) basis using consumption rates arising from the metallurgical testwork and unit costs derived from engineer's database and Bolivian sources where possible. Estimated average life-of-mine costs are as follows.

Operating Cost Summary

Mining	$1.08	per tonne mined
Processing	$6.41	per tonne of leach material
G&A	$0.80	per tonne of leach material

Capital and operating cost estimates for this PEA Update are expressed in US dollars, based on Q1 2011 costs. Estimates have a scoping study accuracy range of ±35-50% and do not include provisions for inflation risks or future price escalation factors.

1.7 Project Economics

A discounted cash flow model has been developed for the project. In addition to those outlined above, the main parameters used in the model are summarized below.

Mine Production Rates and Grades

The table below is a summarized extract from the year-by-year mine production plan for the 40,000tpd acid-chloride heap leach Base Case.

Table 1-4 Base Case Mining Schedule by Year

Year	Ore Tonnes	Waste Tonnes	Strip Ratio	Ag (g/t)	In (g/t)	Pb%	Zn%	Cu%	Ga (g/t)
1	10,000,000	4,982,700	0.56	50.46	5.92	0.109	0.027	0.023	3.74
2	14,000,000	13,977,100	1.00	47.27	7.51	0.094	0.027	0.020	3.89
3	14,000,000	14,167,200	1.02	42.65	10.42	0.098	0.029	0.021	4.08
4	14,000,000	15,318,700	1.09	38.10	8.10	0.083	0.031	0.020	4.09
5	14,000,000	22,619,600	1.64	35.94	5.33	0.043	0.010	0.030	5.43
6	14,000,000	39,795,600	2.84	25.97	5.04	0.047	0.016	0.037	5.24
7	14,000,000	55,808,300	3.99	28.86	5.15	0.051	0.017	0.036	4.53
8	14,000,000	55,046,400	3.93	27.81	5.66	0.068	0.022	0.034	4.33
9	14,000,000	56,725,300	4.05	30.49	6.13	0.069	0.027	0.034	3.46
10	14,000,000	55,876,900	3.99	33.16	6.87	0.085	0.039	0.023	3.58
11	14,000,000	55,803,400	3.99	35.82	8.45	0.105	0.074	0.035	3.53
12	14,000,000	26,159,100	1.87	21.94	9.00	0.091	0.088	0.013	3.54
13	14,000,000	16,477,000	1.18	23.73	8.82	0.100	0.103	0.009	3.75
14	14,000,000	9,262,400	0.66	30.16	9.57	0.145	0.143	0.006	3.06
15	6,831,800	1,490,100	0.22	39.45	8.42	0.116	0.124	0.004	2.81

Metal Prices

Three metal price scenarios were used to show the project's sensitivity to varying metal prices using the same pit model. (At higher metal prices, updated mine models would likely increase total metal production.)

- a Base Case scenario using the approximate 3 year trailing average price for metals;
- a Middle-Case with approximate 1 year trailing average metal prices; and
- a Recent Case which reflects recent monthly average metals prices.

Table 1-5 Financial Model Metal Price Scenarios

Metal	Base Case	Middle Case	Recent Case
Silver (US$/ ounce)	$18.00	$25.00	$35.00
Indium (US$/ kg)	$500.00	$570.00	$650.00
Lead (US$/ lb)	$0.90	$1.00	$1.20
Zinc (US$/ lb)	$0.90	$1.00	$1.10
Copper (US$/ lb)	$3.00	$3.70	$4.30
Gallium (US$/ kg)	$500.00	$570.00	$730.00

Recoveries

Recoveries for pay metals reflect weighted average recoveries that take account of the different ore zones in the deposit based on mineral resource estimates described in Section 17 and metallurgical testwork results described in Section 16 of this report.

Metal	Acid-chloride Heap Leach Recovery
Silver	73.6%
Indium	81.0%

Metal	Acid-chloride Heap Leach Recovery
Lead	51.1%
Zinc	62.0%
Copper	84.8%
Gallium	26.9%

Metal Pay Factors, Treatment Charges, Refining Charges (TC/RCs) and Transport

The following pay factors and allowances for TC/RCs and transport have been used in the Base Case financial model

Metal	Pay Factor	TC/RCs and Transport
Silver	97%	$0.36/ oz
Indium	100%	$0.00/ kg
Lead	90%	$0.14/ lb
Zinc	85%	$0.14/ lb
Copper	95%	$0.24/ lb
Gallium	100%	$0.00/ kg

Gold Credits

Gold credits of $0.72/tonne are included in the financial model as described in Section 18.

Capital and Operating Costs

Capital and operating costs including sustaining and deferred capital, working capital and reclamation and closure provisions are included as summarized above and described in more detail in Section 18.

Key Financial Indicators

The following are outputs of the financial model for the acid-chloride Base Case.

Table 1-6 40,000tpd Acid-chloride Heap Leach - Key Financial Indicators

Measure	Base Case	Mid-Case	Recent
Metal Prices			
Silver (US$/ounce)	$18.00	$25.00	$35.00
Indium (US$/kg)	$500.00	$570.00	$650.00
Lead (US$/lb)	$0.90	$1.00	$1.20
Zinc (US$/lb)	$0.90	$1.00	$1.10
Copper (US$/lb)	$3.00	$3.70	$4.30
Gallium (US$/kg)	$500.00	$570.00	$730.00
Average Operating Cash Flow	-	-	-
1st 5 years (per year)	$185 M	$287 M	$430 M
Life-of-Mine (per year)	$124 M	$208 M	$325 M
Net Cash Flow (undiscounted)	$1,261 M	$2,528 M	$4,298 M
NPV at 5% discount rate	$704 M	$1,482 M	$2,571 M
NPV at 8% discount rate	$505 M	$1,104 M	$1,942 M
Internal rate of Return (IRR %)	37.7%	63.0%	92.9%
Payback period	27 months	19 months	15 months

Measure	Base Case	Mid-Case	Recent
1st 5 years			
Silver cash costs before credits (US$/ounce)	$9.70	$9.70	$9.70
Silver cash costs after credits (US$/ounce)	$2.94	$2.01	$0.86
Life of Mine			
Silver cash costs before credits (US$/ounce)	$13.87	$13.87	$13.87
Silver cash costs after credits (US$/ounce)	$5.06	$3.85	$2.39

Sensitivities

The financial model has been used to test the sensitivity of the project to a range of variables. The chart below represents the sensitivity of the 40,000tpd acid-chloride heap leach Base Case to changes in key variables by charting the impact of ±20% deviations from the Base Case set point on NPV (at 5% discount rate).

- The Base Case project is most sensitive to silver price and recovery, with each 1% change in either silver price or recovery impacting NPV by around $17M.
- The project is also sensitive to operating cost changes (mining, processing and G&A), with each 1% change impacting NPV by around $12.7M.
- Changes in processing costs (~$7.6M NPV change for each 1% change) have a greater impact than changes in mining costs (~$4.1M NPV change for each 1% change).
- NPV changes by ~$3.5M for each 1% change in capital costs.

The economic analysis in this study is appropriate for a Preliminary Economic Assessment but further studies will be required to demonstrate a higher degree of economic certainty.

1.8 Alternative Case – Cyanide Heap Leach

The project is also amenable to a cyanide leach process which would focus only on recovery of silver with gold and copper credits. An Alternative Case, treating 20,000tpd through a cyanide heap leach operation has been considered. Mining for the cyanide option would be similar to the Base Case except at 20,000tpd not 40,000tpd. Heap leaching would also be similar to the Base Case except at the lower treatment rate and using cyanide as the leaching agent not acid. Recovery of gold and silver would be achieved using the well-known Merrill-Crowe process. Tailings and other infrastructure would be similar to the Base Case but smaller due to the lower treatment rate and shorter mine life.

Table 1-7 summarizes key outputs of the financial model for the cyanide heap leach alternative.

Table 1-7 Cyanide Heap Leach Case – Key Financial Indicators

Measure	Base Case	Mid-Case	Recent
Silver Price (per ounce)	$18.00	$25.00	$35.00
Recovered silver (ounces)	108,818,628	108,818,628	108,818,628
Project cash flow ($US million)	$712	$1,454	$2,514
NPV at 5% discount rate ($US million)	$366	$796	$1,410
Internal Rate of Return	27.0%	44.4%	64.8%
Payback period	36 months	26 months	21 months
Life on Mine	19 years	19 years	19 years

Measure	Base Case	Mid-Case	Recent
Averages 1st 5 years			
Ag production (ounce/ years)	7,524,099	7,524,099	7,524,099
Ag grade (g/t)	50.65	50.65	50.65
Silver cash costs (per ounce)	$5.10	$5.10	$5.10
Cash flow pa (US$ million/ year)	$80.6	$132.1	$205.7

The value added by the indium, lead, zinc and gallium from the acid leach process is significant and allows for greater exploitation of the deposits; longer mine life, higher metal production and higher NPV. As well, the indium and gallium are regarded as strategic metals that give the project future upside potential. For these reasons the acid leach option is the preferred option and the cyanide heap leach option is considered a fallback option in the event that the acid-chloride heap leach option proves not to be viable. The cyanide case will remain open for further study in subsequent phases.

1.9 Project Development Plan

Subject to satisfactory outcomes through the process, SASC proposes to progress towards development of Malku Khota in a number of study phases that involve increasing levels of detail.

- On completion of the PEA, SASC proposes to undertake a pre-feasibility study (PFS) and to follow up with a detailed feasibility study (DFS) before advancing to construction.
- The main aim of the PFS will be to demonstrate technical and financial potential and to define the project that will be studied in a subsequent DFS. The DFS would describe the project in detail and ultimately provide the basis for the investment decision.
- Both the PFS and the DFS will include additional definition drilling to progressively promote Inferred Resources to the Measured and Indicated categories and ultimately to demonstrate sufficient Mineral Reserves in the DFS to support an investment decision.
- Further metallurgical testwork will also be required for both the PFS and DFS phases. SASC is currently considering a pilot plant to be constructed on site. The idea is that a pilot plant will be designed and partly procured in the latter part of the PFS in order to be constructed and operated during the DFS.
- Environmental and social baselining (data collection) will be carried out in parallel with the PFS with the aim of collecting at least 12 months of data to feed into the environmental and social impact assessment (ESIA) process.
- The ESIA process starts towards the end of the PFS and would ultimately be based on the project defined by the DFS. The ESIA ultimately leads to the development of the environmental management plans (EMPs) that are designed to mitigate the impacts. These processes eventually lead into the project permitting process.
- Rather than starting out at full capacity, it is possible that a starter “module” of 5,000-10,000tpd capacity would be built initially, stepping up to full capacity progressively over a number of phases. This concept will be examined in subsequent study phases.

Feasibility studies and the parallel environmental, social and permitting processes are inherently uncertain and it is not possible to express a forward-looking timetable with a high degree of accuracy. At this stage the Company proposes to start a pre-feasibility study (PFS) commencing in Q2-2011 and to follow up with a detailed feasibility study (DFS) in 2012 before advancing to construction.

1.10 Conclusions and Recommendations

The technical and economic evaluations in this PEA demonstrate that the silver-indium deposits at Malku Khota have sufficient value to support a decision to advance to PFS and, assuming satisfactory findings, then to advance to DFS. The decision to proceed to construction will, amongst other things, be contingent on the results of the DFS.

The essential objective of the PFS will be to confirm that a potentially viable project worthy of further examination in a DFS exists. This will involve identifying and evaluating obstacles to the project, collecting additional information and examining and selecting between development alternatives. The DFS will ultimately put the economic case for the project and be the basis for the environmental and social review process, the separate but related project permitting process, and financial commitment to its ultimate development.

It is recommended that the following areas are addressed in the PFS as they have the greatest potential to impact project outcomes.

- Carry out infill drilling of the deposits in the early stage of the PFS to ensure that sufficient Inferred Resources are promoted to Measured and Indicated categories to support the decision to proceed to DFS.
- Maintain a focus on metallurgical and materials testwork, particularly:
  - leaching testwork to improve the understanding of the relationship between recoveries and the different ore types in the orebody;
  - hydrodynamic and geotechnical testing of heap leach ores to verify the design assumptions based on preliminary tests carried out during the PEA;
  - metals recovery testwork to allow detailed design criteria to be developed for the design phase.
- Carry out the following work related to open pit mining:
  - Bring all leach pad material within the pit boundaries to Indicated or Measured resource categories in order to allow the tonnages to be used in the Pre-Feasibility study.
  - Undertake ARD testwork on all expected waste materials to ensure that no long-term waste storage issues exist.
  - Carry out geotechnical test work on all the pit areas as well as foundations of the proposed waste dump locations, considering different rock types, wall slope orientations, and potential faulting.
  - Develop rock strength parameters suitable for accurate blasting estimation and crusher design.
  - Assemble a comprehensive cost database for labor, explosives and equipment.
  - Carry out trade-off studies pertaining to the electrification of the pit for the shovels and drills for operating cost optimizations.
  - Carry out condemnation drilling beneath all waste dump locations to ensure mineralized material is not trapped beneath these areas.
- Investigate the following process enhancement opportunities:
  - Preliminary testwork indicates that leaching recoveries improve at finer material sizes suggesting that vat leaching or possibly agitated leaching might produce

higher economic returns than heap leaching. These options should be investigated as part of the PFS.

- Early-stage testwork suggests that significant recovery gains could be achieved at higher leach reaction temperatures. Higher solution temperatures are also a benefit in metals recovery circuits. This would allow more value to be extracted from the orebody and warrants further investigation.
- There is evidence that power can be generated economically by using Bolivia's cheap natural gas at a significant saving on power from Bolivia's electrical grid. This should be investigated further for its potential benefit to the project
- Acid consumption is a significant cost of operation and the potential to reduce costs and simplify logistics by making acid on site should be studied in the next stage.

- Continue to develop the cyanide case while it remains a viable alternative to the acid-chloride leach case noting that the competitiveness of the cyanide case or a blended cyanide/acid leach case increases as the silver price increases relative to other metals.
- Develop a detailed plan for a pilot plant to be built on site and operated during the DFS.
- Carry out PFS-level investigations and engineering studies into:
  - Important infrastructure aspects, specifically power supply, water supply, transport and logistics.
  - hydrological, hydrogeological and geotechnical aspects of the site to facilitate detailed design and cost estimating.
- Carry out a study to develop a better understanding of the market for the products that will arise from the project. The study will need to provide sufficient detail regarding the payment terms including "payability", TC/RCs, penalties and transport costs for trade-off studies to be undertaken.
- Advance environmental and social baselining to ensure that sufficient data is gathered) in advance of the ESIA process starting in earnest (minimum 12 months required).

2 INTRODUCTION

This report is classified as a Preliminary Economic Assessment (PEA) and is formatted and prepared in accordance with the Canadian National Instrument 43-101 (NI 43-101) "Standards of Disclosure for Mineral Projects". The report was prepared with contributions from nominated Qualified Persons (QPs) and input from South American Silver Corp. (SASC) personnel, contractors and consultants.

2.1 Background and Purpose

South American Silver Corp., (SASC), controls extensive exploration mining concessions at Malku Khota in the Department of Potosi, Bolivia through its wholly owned Bolivian subsidiary Compañia Minera Malku Khota S.A. (CMMK). CMMK holds mineral rights to 5,475 hectares at Malku Khota, covering 15 km of projected strike length along prospective silver mineralized sandstone units. Exploration of the concessions is at an advanced stage and, subject to satisfactory feasibility studies and environmental and social impact assessments; SASC intends to develop the Malku Khota property into a producing mine.

A Preliminary Economic Assessment (PEA) for Malku Khota was completed in March 2009 by Pincock Allen & Holt (PAH) based on resource estimates of November 2008 (also by PAH). The 2008 resource estimates and the 2009 PEA are superseded by this technical report, which takes account of work carried out since that time, including the following.

- Drilling has progressed to take the total meters drilled on site from 25,000m previously to over 40,000m for the current resource estimate.
- Resource estimates have been updated based on the additional drilling. The updated resources are included in this PEA.
- A significant metallurgical testwork program has been advanced to determine leaching characteristics of the ore and to test downstream metal recovery processes.
  - An extensive test program at SGS Lakefield through 2010 that included agitated acid leaching, bottle roll acid leaching, cyanide leaching, column leaching and diagnostic leach testing has been completed.
  - Preliminary geotechnical and hydrodynamic testing demonstrated satisfactory stacking characteristics and structural integrity of heap leach ores.
  - Kinetic soak tests & acid optimization testing of hanging wall samples was started in 2010 and will be ongoing through Q2 2011.
  - Flotation & QEMSCAN tests on hanging wall samples were started in late 2010 and will extend into early 2011.
  - Cyanidation tests of selected samples started at SGS in December 2010 were completed in January 2011.
  - ECO-TEC bench scale acid recovery amenability tests have been completed.
  - Comminution tests to derive crushing and grinding indexes will be undertaken at Hazen in Denver starting in early February 2011.
  - A scoping study into an Indium refinery is being undertaken by a consultant that has recently designed and constructed a similar plant in Peru.
- A number of mining and treatment options have been evaluated at scoping-level with the aim of narrowing the alternatives to be studied in the PFS.

  - o The size of the deposit at Malku Khota suggests that it will support a mining and treatment rate of 40,000tpd or more rather than the 20,000tpd used in the 2009 PEA.
  - o Acid-chloride leaching remains the preferred option (to cyanide leaching) because of its ability to recover not only silver but also indium, base metals (lead and zinc), gold and gallium.
  - o Vat leaching and agitated leaching have been considered as alternatives or add-ons to heap leaching. This work is at an early stage and further work is required before definite conclusions can be drawn.
- SASC project personnel have visited Bolivia, Chile and Peru gathering information on local engineering and construction capability, and local cost factors for the capital and operating cost estimates.
- A preliminary hydrology and hydrogeology study has been completed.
- A preliminary power supply study by a Bolivian power specialist has been completed.
- Work has been awarded to a La Paz-based consultant, to undertake environmental and social baselining in 2011.

The purpose of this technical report is to provide updated information on the Malku Khota project at a scoping level of detail taking account of the work undertaken since the 2009 PEA.

2.2 Qualified Persons

The Qualified Persons involved in the preparation of this report and the sections for which they are responsible are listed in the table below.

Table 2-1 List of Qualified Persons

Author	**Company**	**Section Responsibility**
Allan Armitage, Ph.D., P.Geo	GeoVector Management Inc.	17
Pierre Desautels, P.Geo	AGP Mining Consultants Inc.	12,13,14
Gordon Zurowski, P.Eng	AGP Mining Consultants Inc.	18
William Pennstrom	Pennstrom Consulting Inc.	16,18
Ralph Fitch	South American Silver Corp.	1,2,3,18,19,20,21,22,23
Felipe Malbran	South American Silver Corp.	4,5,6,7,8,9,10,11,15

3 RELIANCE ON OTHER EXPERTS

The authors of this report have relied primarily on information provided by SASC including:

- documents referred to in this report or listed in the Reference Section of this report,
- other data supplied by SASC or generated by its consultants and contractors;
- data, reports and opinions from prior owners and third-party entities (consultants);
- site visits to the Malku Khota project, personal inspection and review;
- assumptions, conditions and qualifications as set forth in the report;
- discussions with SASC personnel; and
- information found in the public domain.

Where possible, the information provided has been verified by comparison with other data sources or by field verification. Where checks and confirmations were not possible, all information provided is assumed to be complete and reliable within normally accepted limits of error. In the course of the review, the authors have not discovered any reason to doubt that assumption.

Independent title searches have not been conducted and the authors have relied upon SASC for information on the status of the claims, property title, agreements, permit status and other pertinent conditions.

Financial analysis and PEA coordination have been the responsibility of SASC.

4 PROPERTY DESCRIPTION AND LOCATION

The Malku Khota Project (Latitude 18.15º South, Longitude 66.21º West) is located in the Department of Potosi, Bolivia within the canton of Sacaca in the Province of Alonso de Ibanez, and in the cantons of Toracari and San Pedro de Buena Vista in the Province of Charcas. The project area is located approximately 98 km east-southeast of Oruro, and 85 km south of Cochabamba in a relatively remote area accessible by an improved dirt road that links Oruro and Cochabamba. The primary access is from the supply center of Oruro via Bolivar, Sacaca, and Chiro Khasa. The total property position controlled by SASC through its wholly owned subsidiary Compañia Minera Malku Khota (CMMK) consists of 5,475 hectares that extends 21 km in a northwest to southeast direction. Figure 4-1 shows the general location of the Malku Khota Project. The land is currently held by CMMK as Concessions (Cuadriculas). Relatively minor amounts of precious metal mining occurred in Spanish Colonial times (pre-1800) and small amounts after 1800.

4.1 Location

The Malku Khota silver property is accessed by a 162 km road from Oruro, or by a 138 km road from Cochabamba. CMMK currently holds mineral rights to an aggregate of 5,475 hectares that includes 15 km of projected strike length along favorable silver mineralized sandstone units. The cuadriculas can be maintained indefinitely by paying annual dues in January of each year. The fee is US$1.60 per hectare per year for the first five years, and rises to US$3.20 per hectare per year in the sixth year. The New Constitution requires that the concessions are exchanged for contracts with the government. The regulations defining this process are expected to be in place by the end of 2011.

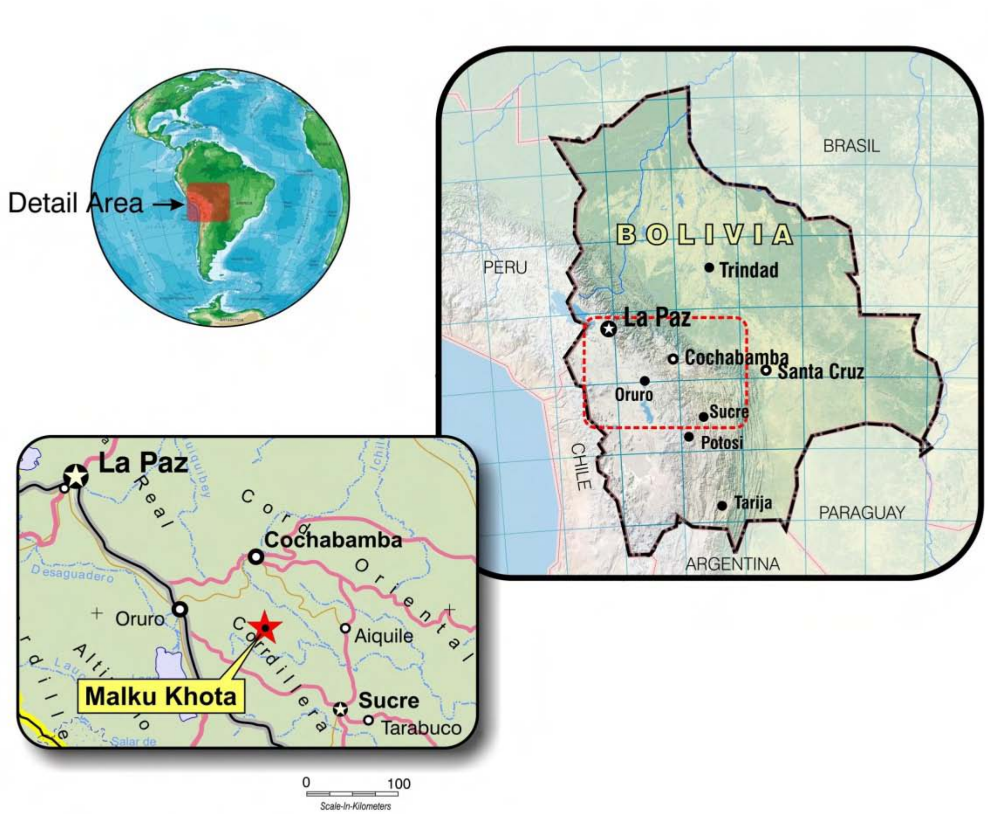
4.2 Tenure

In July 2003, General Minerals Corporation acquired the original concessions and the properties comprising the Kempff Option (Cobra and Daniel), through its indirect, wholly owned Bolivian subsidiary, Compañia Minera General Minerals (Bolivia) S.A., and in December 2003 transferred the property to CMMK. On March 27, 2007, CMMK made its final payment to purchase the concessions comprising the Kempff option with a retained 1% NSR). A list of claims is shown in Table 4-1.

Table 4-1 Cuadriculas Controlled by CMMK

Claim Name	Owner	ha
Norma	Compañia Minera Malku Khota S.A.	250
Cobra	Compañia Minera Malku Khota S.A.	125
Daniel	Compañia Minera Malku Khota S.A.	1,050
Takhaua	Compañia Minera Malku Khota S.A.	725
Alkasi	Compañia Minera Malku Khota S.A.	950
Jalsuri	Compañia Minera Malku Khota S.A.	125
Takhuani	Compañia Minera Malku Khota S.A.	1,150
Silluta	Compañia Minera Malku Khota S.A.	775
Antacuna	Compañia Minera Malku Khota S.A.	150
Viento	Compañia Minera Malku Khota S.A.	175
Total Area of Cuadriculas		**5,475**

Figure 4-1 Project Location Map



The royalty on the Cobra concession (125 hectares) and the Daniel concession (1050 hectares) is a 1 percent Net Smelter Return (NSR) payable to the former owners on all production. This NSR can be purchased at any time for US$500,000. Five of the 47 Cuadriculas comprising the Cobra and Daniel concessions have a prior water right issued under the old mining code which gave the holder a first right on the "water rights" such that CMMK would be required to have a further agreement with the holder of the water rights to use the water on these five cuadriculas. The Bolivian mining code of 1997, however, does not recognize these water rights.

In September 2006, CMMK acquired 925 hectares from SILEX Bolivia, S.A., and in April 2008, they acquired the 250 hectare Norma claim, resulting in a total of 5,475 hectares of mineral concessions. Property boundaries in Bolivia are located by UTM Coordinates. The property boundaries have not been surveyed; however, the known mineral deposits are located well inside the property boundaries.

A "Licencia Ambiental" permit from the Prefectura of Potosi, has been approved and is applicable for the exploration stage. A Base Line Environmental and Social Study is in preparation.

4.3 Environmental Liability and Permitting

SASC reports that there are no environmental liabilities associated with the project. Some alpine natural lakes near the site which contain fish may need additional monitoring to ensure that exploration activities do not jeopardize the water quality in these lakes. CMMK monitors the environment each six months to ensure that there is no environmental degradation. Surface rights are owned by local indigenous communities. CMMK makes agreements with the local communities prior to accessing and working on their land.

4.4 Property Status and Ownership

A new Constitution was enacted in February, 2009 which requires the creation of a new mining code. This code is presently being written. In the meantime the earlier mining code is being followed with some modifications. Thus mining activities continue to be regulated by Law 1777 as amended and the Mining Code, as amended, dated March 17, 1997. Mining is considered an activity of public importance in Bolivia, and it has preference over other industries in regard to the application of certain specific regulations.

In 1992, the Bolivian government approved an environmental law establishing a national environmental policy to protect the environment and to promote sustainable development, the preservation of biological diversity, and environmental education.

The primary permit required for mining operations is the environmental permit. Table 4-2 shows a summary of the permits and procedures discussed above.

Table 4-2 Summary of Permits and Procedures

These permits and procedures may change within the new mining code that is presently being written.

Subject	Permits, Licenses or Other Required Procedures
Mineral Rights	Application to the Mining and Metallurgy Ministry to get title
Land Use	Leasing from indigenous groups or buying or expropriation from other landowners
Water Use	Mining title gives right to use water within the perimeter of the mining property
Environmental	DIA (Environmental License) integrates all the environmental authorizations, permits, and protection requirements for mining activities
Hazardous Substances	The operator should include hazardous substances in the EIA and comply with norms given by environmental legislation and by the manufacturer
Controlled Substances	Handling, use, and disposal of controlled substances should be included in the EIA and requires permitting.
Explosives	Handling, use and disposal of explosives should be included in the environmental license, and a special permit should be applied for from the National Ministry
Construction Permits	Construction should be included in the ESIA, otherwise an environmental license is required. Within areas of Territorial Management, the respective authorization is required.
Roads	Road construction requires an Environmental License if the road was not included within the mining project. To improve roads of the National Net, an authorization is required from the respective Road Service.
Freight Transportation	Freight Law limits dimensions and load limits to freight transported by land. Special cargo exceeding those limits requires a Special Permit to be applied for before the National Department of Transport.

Subject	Permits, Licenses or Other Required Procedures
Imports and Exports	Imports and exports of all types of merchandise are free of permits except for goods that affect public health or national security. The importer or exporter must be registered at the Unique Register of Exporters, RUE.
Safety	All companies should enact Internal Work Regulations, where the safety rules should be established.
Social Security	Mining companies should cover short-term risks. The Retirement Law rules long-term risks.
Expatriate Labor	No restrictions for foreigners to work in Bolivia exist, but a working visa is required. The percent of foreigners working within a single company cannot exceed 15% of the total employees.

CMMK controls a significant block of land in the area around the Malku Khota Project. Figure 4-2 shows the perimeter of the property. The total surface covered by the Malku Khota group of properties is 5,475 hectares.

There are basically two alternatives for a mining concessionaire to obtain land surface rights for mining uses: acquisition through a purchase or lease agreement with the owner, or the expropriation from the owner. If neither of these two procedures can be applied, then servitude can be established.

The land surface rights that need to be acquired for the use of the project, will include the areas for the mine, plant, water wells, “heap leach” areas, camp, tailings pond, and runway. There are no dwellings or agricultural buildings within the presently known footprint of the mineralized silver-indium resource areas, and no one lives within these boundaries. The small community of Malku Khota (approximately 20 dwellings) is located within 1 kilometers of the mineralized footprint. The SASC project is currently in the exploration phase and the requirements for surface rights for mining operations have not yet been determined.

Bolivian legislation facilitates the granting of rights-of-way and easements for any infrastructure work that is considered of public need. If the works to be constructed are part of a mining project, then the pertinent parts of the Mining Code can be applied. If the construction work implies that a third party will provide a service, then the specific legislation will have to be applied. Bolivian legislation in general facilitates the acquisition of rights-of-way for infrastructure works that are of public need. These legislative requirements may change under the new mining code.

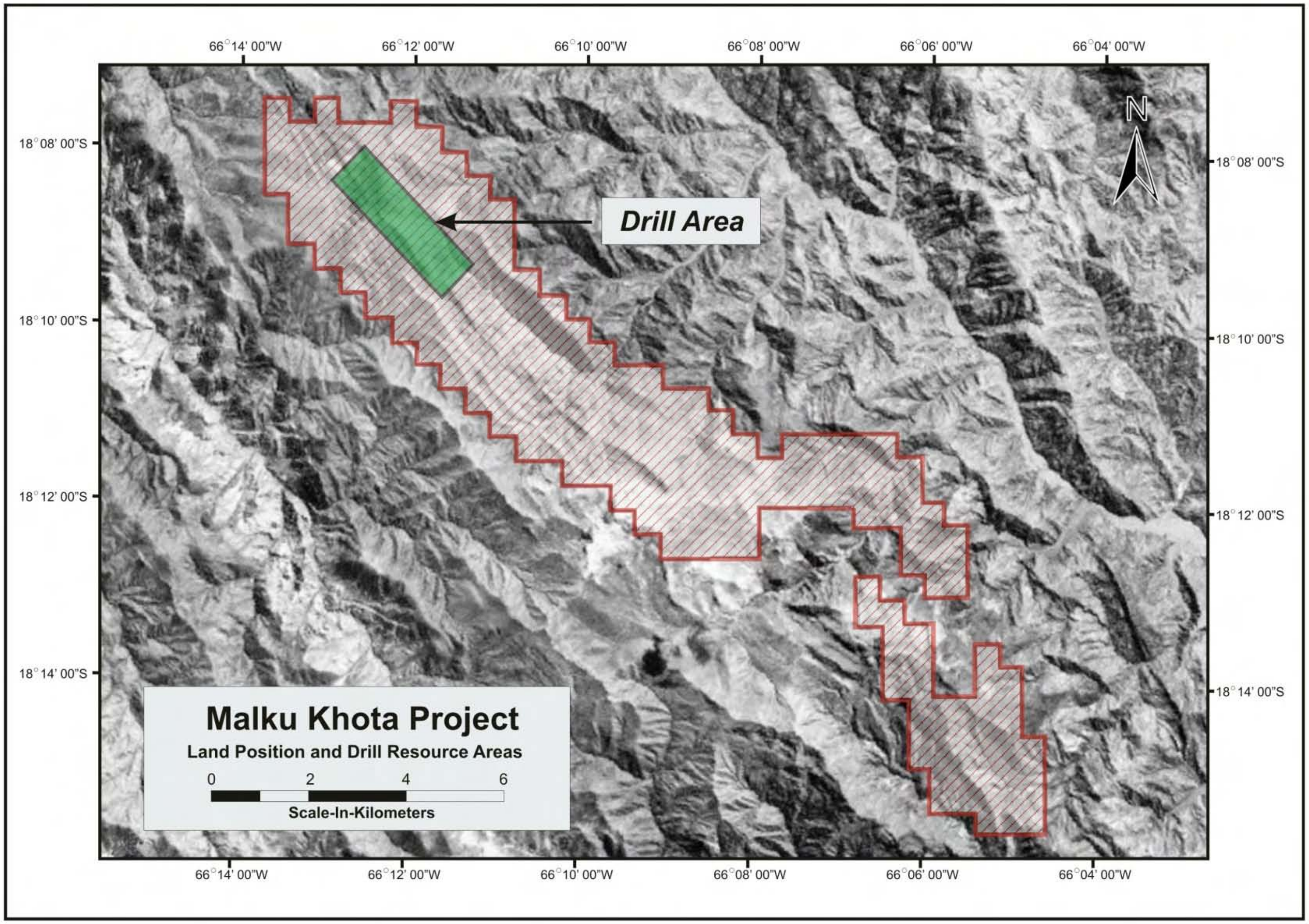
In the case of road construction under the concession system, the institution granting the concession becomes responsible for developing the required expropriations, which should be transacted according to the laws in force. The real estate incorporated in the concession will become public domain, and they cannot be alienated, mortgaged, or subject to liens of any kind separately from the construction. Expropriations that affect original community lands will be obtained by the granting entity from the National Institute of Agrarian Reform.

4.5 Agreements

The Bolivian Mining Law bans any mining activity within the limits of towns, cemeteries, public roads, etc. Therefore, the status of the land where the small community of Malku Khota is located will be defined. If the land is declared to be in the public domain and belonging to the municipality, then a Municipal Resolution from the Municipal Agent will be required. As stated by Articles 34 and 35 of the Mining Code, SASC could start using the land for mining activities as soon as this land is declared to be public domain.

In reference to the competence of community authorities to force individual owners of houses or land to be moved within the limits of their jurisdiction, the Municipality has the right to expropriate houses or buildings based on the public need (Organic Municipal Law, Chapter 21, Article 9).

Figure 4-2 Malku Khota Land and Drill Hole Areas



5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

This section is a verbatim extract of the 2009 PEA prepared by Pincock Allen & Holt.

5.1 Access

Access to the Malku Khota property is gained overland via improved roads from Oruro, via Bolivar, Sacaca, and Chiro Khasa. There is also access via improved dirt roads from Cochabamba, via Sakani. This main road links Oruro with Cochabamba and provides reliable access for regular truck service for people and supplies along this route. The main road traverses the property to the west of the exposed mineralized zone. The town of Sakani, which has electric power and phone service, is situated approximately 12 kilometers to the northeast of the project area and is the location of the CMMK field office and the core storage and cutting facilities. In recent years electric power has been brought into the village of Malku Khota and the town of Chiro Khasa. Travel time from Oruro to the Malku Khota project is approximately four hours during the dry season and 5 hours during the rainy season (December to March). Travel time for the 138 kilometers between the property and Cochabamba is approximately 4 hours during the dry season.

5.2 Climate

The climate is typical for the Bolivian Cordillera Oriental and Altiplano, with cool to moderate rainy summers and cool, dry winters. Winters (May – August) are cool with temperatures that range from -2˚C at night to 10 to 12˚C during the daytime, and are generally dry, with snowfall common at the higher elevations. Summers (November – March) have moderate temperatures that range from 5˚C at night to 12 to 25˚C during the daytime. Rainfall is heaviest during late December through March, averaging approximately 125 millimeters (mm) in December, 170 mm in January, and lightest in June and July where rainfall averages approximately 20 mm per month. Scattered subsistence farm plots and seasonal pastures are found scattered throughout the Malku Khota project area and are utilized by local inhabitants on a seasonal basis.

5.3 Local Resources and Infrastructure

The Malku Khota property is relatively remote with dependable access primarily from Oruro, which has a capable supply of labor, equipment, or service requirements for conducting exploration and mining related activities. The town of Sacaca lies approximately 75 kilometers to the northwest along the main road and may be able to provide a source for local temporary labor, supplies, and accommodations to support an exploration program. The town of Sakani is located 12 kilometers to the north east of the property and is the closest town with electricity and phone service. Chiro Khasa is a local village approximately 25 kilometers south of the project area on the road to Sacaca and is a mid-way point between Oruro and Cochabamba, which trucks carrying supplies and people often use as a lay-over en route. The majority of the day labor for road building, sampling technicians, core cutting and support work has been found among the 12 local communities in the project area. The project provides some of the only employment source of income in the area and appears to have been successful in building relationships between CMMK (SASC) and the local communities.

The company (SASC) has a large land position of 5,475 hectares, which include mining rights. Under the 1997 Bolivian mining laws, these cuadriculas also give surface rights for mining activity, but under the new constitution arrangements also need to be made with the local communities and individuals claiming surface ownership rights. Optimization of significant mining operations may require additional land agreements going forward with the project to allow efficient waste disposal, low grade stockpiles and tailings impoundments.

5.4 Physiography

The Malku Khota property is located in the eastern part of the Altiplano, a portion of the Cordillera Oriental of Bolivia, and encompasses a north – south trending mountain range that lies between a broad structural valley to the west and a deep river valley to the east. The mountain range consists of barren rock and talus that rises to elevations ranging from 3,800 to 4,580 meters asl and with relief of up to 400 meters above the surrounding valleys. The majority of the ranges and valleys in the Cordillera Oriental reflect the north - south trending fold and thrust belt which have been subject to glaciation and dissection by erosion. The hill slopes in the project area are covered by grasses and sparse low brush, which are currently utilized for grazing and scattered farm plots by the local inhabitants living in several small villages within the property boundaries.

The known mineralization areas lie along the western edge of the range, which coupled with their near surface and surface exposure, would require very low stripping ratios from a mining perspective. Near the small village of Malku Khota, the rolling hills west of the project area consist of glacial moraine deposits, with two small alpine lakes, Laguna Malku Khota and Laguna Wara Wara located in glacial depressions. This area of low rolling hills has the potential for siting the future mine processing facilities.

The Malku Khota property area is dominated by high-altitude bunchgrass and scattered shrub vegetation interspersed with barren rock ridges. The average population density is between 2 and 12 people per square kilometer, giving it a designation of “moderately populated”. Many of the local inhabitants maintain houses in the villages as well as dwellings scattered near the fields and pastures. The villages of Malku Khota and Kalachaka lie within or immediately adjacent to the project site boundaries, but outside the zones of mineralization.

5.5 Sufficiency of Surface Rights for Mining Operations

The Malku Khota project is currently in the exploration phase. This PEA has determined that the conceptual open-pit mining, processing and waste disposal could be conducted within the existing boundaries of the property.

6 HISTORY

This section is a verbatim extract of the 2009 PEA prepared by Pincock Allen & Holt.

Mining in the Malku Khota area dates from Spanish Colonial times, pre-1800, so silver and gold mining has a several hundred year history in the region. Prior to the involvement of CMMK, mining occurred on at least 11 separate high-angle structures scattered along the western face of the main ridge. These structures were explored on at least seven different levels and on the eastern side at the old Sucre workings since the early 1800s. Historical reports indicate that surface gold-silver veins included assays of between 2.0 and 47 gpt gold and 27 - 1,500 gpt silver from these narrow veins and structures that are generally less than 1.5 meters wide. Recent sampling form the disseminated silver mineralization included an assay of 0.9 gpt gold and 537 gpt silver over a width of 0.45 meters. This surface exploration was carried out by Geoexplorers Bolivia and Compañia Minera La Rosa between 1994 and 1995. Reports in the possession of CMMK from this period include results from approximately 100 vein and wall rock samples primarily from these historic workings.

Silver mineralization was discovered and mined during the early Spanish Colonial times and these workings are exposed in the Kallampa, Frio, and Pique Pobre areas; however, very little is known about this period of mining activity and no written records have been found. Recent exploration of the underground workings at Pique Pobre was documented in the technical reports by Kurt Katsura (2004, 2006 and 2008). Underground mapping and sampling conducted in 2005 suggests that the old workings followed high-grade stratabound ore zones that lie just below the surface and appear to reflect a horizon of supergene silver enrichment that follows the current surface profile. This mining was conducted with very little waste rock removed from the workings and virtually all of the mined ore was taken away for processing, possibly at an old milling facility that has been identified near the Malku Khota village adjacent to the lake shoreline.

Very little is known about the historical ownership of the property. The very early mining was in the seventeenth century and it is not known if Geoexplorers of Bolivia (previously mentioned) had rights to the area when they did their reported work.

7 GEOLOGIC SETTING

7.1 Geology

The Malku Khota property area is located in the Andean Cordillera of Bolivia, which has been characterized as a classic example of a convergent continental plate margin (Dewey and Bird, 1970; and Mitchell and Reading, 1969). The Andean Cordillera consists of three segments, northern, central, and southern Andes; each of these segments has some similarities but also have distinctly different Mesozoic and Cenozoic geologic histories. The Malku Khota Project is located in southwestern Bolivia, within the central portion of the Andes and near a major westward oroclinal bend in the cordillera. The central Andes in Bolivia consist of three distinct and contiguous provinces: the Cordillera Occidental, Altiplano, and the Cordillera Oriental, listed in order from west to east. These three provinces are crosscut by the Central Volcanic Zone (Thorpe, and others, 1982) which is the largest of the three active volcanic chains that comprise the Andean Cordillera. The Malku Khota Project lies within the Cordillera Oriental. Two previous 43-101 Technical Reports by Kurt Katsura (available at www.SEDAR.com) have detailed maps, sections and geologic descriptions, some of which have been incorporated into this document.

The Cordillera Oriental is a polygenetic Phanerozoic fold and thrust belt that consists of Paleozoic deep marine and platform facies sediments that are overlain by Mesozoic marine, carbonaceous platform and delta facies rocks. These sediments were deposited primarily on Precambrian basement rocks in a broad miogeosynclinal basin and were subsequently deformed during at least three tectonic and orogenic events: Caledonian (Ordovician); Hercynian (Devonian-Triassic); and Andean (Cretaceous-Cenozoic) (Sempere and others, 1990). The geologic source regions for the Paleozoic and Mesozoic sediments were the Precambrian Brazilian shield areas to the northeast and from the Proterozoic Arequipa massif of Peru-Chile to the west and southwest (Litherland, and others, 1989; Cobbing, 1985). There are no Paleozoic volcanic arcs or major suture zones in the central Andes, suggesting that the fold and thrust belt are an integral part of a passive margin of the Pangean continental margin (Cobbing, 1985). Following the Hercynian deformation period, peralkaline volcanism and the emplacement of granitoid plutons in the central Andes appears to be associated with extensional tectonics, local basin rifting, and the beginning of an active subduction regime along the western edge of the South American continent (Pitcher and Cobbing, 1985). Subduction and the generation of calc-alkaline volcanism commenced during the Jurassic and have continued uninterrupted until the present. During the Tertiary period, primarily during the Oligocene to early Miocene Incaic phase of the Andean orogeny, the Cordillera Oriental fold and thrust belt formed and rocks were thrust westward over the foreland basins of the Altiplano. This period of deformation is also temporally associated with increased volcanism and the development of the central volcanic zone.

In Bolivia, the central volcanic zone consists of two calc-alkaline volcanic arcs that converge near the southern end of the Altiplano. The western branch is the best developed arc and forms the high crest of the Cordillera Occidental, along the Chilean–Bolivian border. The eastern branch of the central volcanic zone occurs along the western margin of the Cordillera Oriental fold and thrust belt and contains a few, and locally large, Miocene to Pliocene dacitic to rhyolitic ignimbrite fields, scattered Miocene domes, remnant stratovolcanoes, and shallow intrusions. The dacite intrusion located at the southern portion of the Malku Khota property is correlated with this age and style of volcanic activity.

The rocks exposed in the Malku Khota Project area consist of Paleozoic sediments that are unconformably overlain by Jurassic to Cretaceous sediments and are locally in thrust fault contact with Jurassic to Paleocene terrestrial and lacustrine sandstones and red bed sediments. A series of North-South trending normal and thrust faults and broad synclinal and anticlinal folds define the ranges and valleys of the Cordillera Oriental, which is characterized by undulating ridges that alternate with broad synclinal valleys and are punctuated by steep dip-slopes and escarpments.

7.2 Regional Geology

The geology of the Malku Khota project area is shown in Figures 7-1 to 7-4 at the end of this section. The oldest rock units exposed in the greater Malku Khota area consist of sandstones, siltstones, and shales of the Silurian-age Uncia and Catavi Formations. The Paleozoic rocks crop out beyond the main area of mineralization and form the valleys and ridges to the east and west of the main project area. These units are not observed to be mineralized in the project area.

In nearby areas, Paleozoic and Jurassic rocks are unconformably overlain by the Cretaceous Tarapaya and Miraflores formations, which consist of sandstones, siltstones and tuffaceous units; and limestones, calcareous sandstones, mudstones, and marls, respectively (Troeng and Riera Kilibarda, 1996). These rocks have not been identified in the immediate project area, but are present to the north and south where they are locally associated with intra-basin basalt flows. The basalt flows are interpreted as evidence of local rifting.

In the Malku Khota Project area, the Paleozoic rocks are unconformably overlain by the Jurassic Ravelo Formation, which consist of white, yellow and red, fine to coarse-grained sandstones of both marine and Aeolian origin. The Aeolian sands exhibit distinct crossbedding structures and are locally intercalated with siltstones and conglomerate lenses (Troeng and Riera Kilibarda, 1996). CMMK geologists have identified three sandstone units within the Ravelo Formation, the lower one is the Chiru Khasa sandstone, the middle one is the Wara Wara sandstone and the upper one is the Malku Khota sandstone. The upper two sandstones are the primary hosts for the silver-indium mineralization at the Malku Khota project (Mateo and Caceres 2011).

In the Malku Khota Project area the Ravelo Formation is unconformably overlain by the middle Cretaceous Kosmina, Aroifilla and Chaunaca Formations which consist of sandstones, siltstones, mudstones and marls with a distinctly reddish color due to abundant iron oxides (Troeng and Riera Kilibarda, 1996 and Mateo and Caceres 2011). Studies carried out in the Potosi basin shows that the Aroifilla and Chaunaca Formations were deposited in an almost continuous enclosed lacustrine system that was subject to large variations in hydrology independent from any significant marine contribution. The hydrology of the basin and sedimentation were dominated by changes in the evaporation/precipitation ratio. Major evaporitic phases occurred during periods of lake contraction which resulted in the development of brine ponds trapped at depth in the lacustrine system (upper Aroifilla and middle Chaunaca Formations). The crystallization and precipitation of evaporate minerals occurred in a subaqueous environment and as interstitital growth both in dry peripheral mud flats and below the sediment surface during periods of collapsing water tables. Wide ranges of water chemistry redox conditions are reflected in the isotopic composition of the sulfates deposited in these rock units, and the source ions in the precipitates were within the basin with some possible contributions of reduced sulfur from volcanic sources.

With widespread occurrence throughout the Cordillera Oriental, the Aroifilla and Chaunaca Formations may in part be laterally time-correlative and represent the evolving facies of the lacustrine and basin depositional system. In the Malku Khota project area, the “red bed” sediments of the Aroifilla Formation unconformably overlie the sandstone units that host the main zone of disseminated silver mineralization. In other parts of the region the Ravelo Formation is overlain by the Tarapaya and Miraflores formations which are in turn overlain by the Aroifilla Formation. Geologists mapping in other areas have noted a major unconformity between the Aroifilla Formation and the underlying Tarapaya and Miraflores formations where they are in contact, and some have suggested that this indicates a tectonic shift and a major paleogeographic change in the Central Andes from a regime dominated by compression to an extensional regime (Jaillard and Sempere, 1991; Martinez and Vargas, 1990). The Miraflores Formation is not present in the immediate Malku Khota project area.

In other areas the Aroifilla and Chaunaca Formations are unconformably overlain by the El Molino Formation, which consists of limestones, calcareous sandstones, and marls (Troeng and Riera Kilibarda, 1996). These sediments were deposited during wetter periods (Chaunaca and El Molino Formations), and reflect the expansion of the lacustrine system which resulted in decreased salinity and less evaporitic influences, carbonate sedimentation (shell coquinas, oolite/oncolite deposits, stromatolitic and thrombolitic accumulations, organic-rich laminated carbonates, etc.) and a diversification of flora and fauna. Towards the end of the depositional period of the El Molino Formation, episodic prevalence of sodic alkaline waters resulted in the formation of analcime-rich laminates, and a possible influence from marine incursions into the lacustrine basin (Sempere, and others, 1997).

A dacite intrusion of probable Miocene to Pliocene age crops out to the south of the main project area and is tentatively correlated with the eastern branch of the central volcanic zone in Bolivia. The dacite intrudes and deforms both Paleozoic and Mesozoic sediments. No apparent mineralization has been observed to be directly associated with this particular intrusive body or in the adjacent sediments. Another intrusive body has been hypothesized to underlie the Malku Khota area of the project and is characterized by the prevalence of steeply dipping East-West veins which have been historically mined for gold, bismuth and silver.

The regional structural analysis of fold and thrust structures in the Cordillera Oriental indicate that they are bivergent to the west and to the east and likely merge into a detachment fault system at a maximum depth of 10 to 15 kilometers. Pre-Cretaceous folds and faults with N/NW to NW oriented axes were either superimposed by N/S-oriented faults or reactivated with a left-lateral strike-slip movement (Mueller and others, 1998). Restoration of erosional rates coupled with tectonic displacements and balanced on cross-sections suggest that in southern Bolivia the basal detachment of this tightly folded kink zone links up with a blind thrust fault or detachment that dips westward and may bear some geometric relationship to the original configuration of the early Paleozoic basins and their margins (Kley, 1996).

A major fault zone in the project area places Paleozoic rocks in thrust contact with the Jurassic and Cretaceous formations lies just west of the property. Pliocene to Pleistocene uplift occurred throughout the Cordillera Oriental, and this resulted in the present erosional pattern following the North-South trend of fold and thrust structures (Walker, 1949). Quaternary and Pleistocene glaciation are responsible for stripping and exposing the resistant sandstone outcrops and creating the depressions that are currently occupied by the Laguna Malku Khota and Laguna Wara Wara and the adjacent glacial moraine capping the nearby hills.

7.3 Project Geology

The geology at Malku Khota consists of exposures of the Ravelo, Kosmina, Aroifilla, Chaunaca and El Molino Formations as previously described. These Units were deposited within a NW striking rift zone developed in the late Jurassic related to the major period of Continental Drift. Geologic mapping in the project area has focused primarily on the rocks that are related to the silver-indium mineralization which include the Ravelo, Kosmina, Aroifilla and Chaunaca Formations. These Formations have been subdivided as shown below.

The Ravelo Formation rocks of Jurassic age are the primary units hosting mineralization at Malku Khota. CMMK geologists have identified three local sandstone units in the Ravelo Formation that host disseminated silver-indium mineralization; these are defined as the Chiru Khasa, Wara Wara and Malku Khota sandstone units. These sandstones all strike approximately NNW and dip to the West at 60 to 80 degrees.

Ravelo Formation

At Malku Khota this Formation has been subdivided into 4 Units, described below:

Chiru Khasa Unit - Lower Jurassic

This unit represents the basal sediments within the Jurassic sedimentary basin. These rocks, in the project area, are in fault contact with the Palaeozoic on the eastern side of the property. The sediments are composed of a series of fine to medium grained rhythmically banded sandstones with intercalations of clay containing occasional quartz grains. The rock is typically pink in color due to varying concentrations of hematite and other iron oxides. This unit is approximately 650 meters thick.

Wara Wara Unit – Upper Jurassic

This unit conformably overlies the Chiru Khasa sandstone and often forms the crests of the ridges in the area. This unit includes a series of rhythmically banded fine and course quartz sandstones within which are very fine intercalations of white clay that are typically less than 2 cm thick. This clay gives a white appearance to the rock. The proportion of clay bands increases from the base to the top of the Unit. The sandstone also exhibits strong cross-bedding on a large scale, with individual sets measuring up to several meters thick. The bedding is cut-off on the upper surface of the cross-bedding sets indicating that the beds are “right-way-up”. The Wara Wara Unit is subdivided into Upper and Lower units.

Lower Wara Wara Unit – Upper Jurassic

The rhythmic sandstones with intercalated white clay layers within the Lower unit can be distinguished by their compact nature and their pinkish color due to the quartz grains being cemented with hematite and jarosite. Also, pronounced convoluted bedding over thicknesses of several meters, believed to have been formed by early stage compaction and dewatering is present. This Lower unit is approximately 850 meters thick.

Upper Wara Wara Unit – Upper Jurassic

This continuation of the rhythmic sandstones with intercalated clay layers starts with a marker horizon consisting of a 6 metre thick coarse brown sandstone. This unit continues to exhibit cross-bedding but the convoluted bedding is absent. This sandstone is typically cleaner than the one in the Lower unit and includes white quartz grains. Ridge tops are often composed of this sandstone unit in the Project area. The unit is approximately 40 meters thick.

Malku Khota Unit - Upper Jurassic

This sandstone sequence is more massive with a higher degree of silicification and forms the prominent scarp which extends the full length of the drilled area in a north-south direction. The base of the sandstone unit includes a very coarse quartz sandstone with grains up to 2 mm with some lithic fragments up to 5 mm. This unit which is approximately 100 meters thick changes from coarse at the base to fine at the top. The quartz grains are typically well rounded and “sacaroidal” in nature. The finer grained portion of the unit is typically a hard, compact brownish quartzite.

Kosmina Formation

At Malku Khota this Formation is divided into two Units, described below:

Lutita Edwin (Basal Intermedia Unit) – Lower Cretaceous

This blue green possibly tuffaceous shale which is approximately 4 to 6 meters thick, unconformably overlies the Malku Khota Unit. The shale probably represents a tuffaceous shale representing a volcanic event within the larger rift zone. This shale exhibits a weak foliation which may have been caused by fault slippage.

Intermedia Unit - Lower Cretaceous

This Unit which includes the Lutita Edwin is part of the Kosmina Formation. At Malku Khota the lower part of the Kosmina Formation is missing. The Unit is characterized by a basal sequence of brown and white fine grained sandstones which include 4 to 6 bands of intercalated green and brown shales. These shaley partings are typically 10 to 20 cm thick. The remainder of the unit continues with similar sandstones with the addition of 20 to 30 cm thick bands of red sandstones cemented with iron and manganese oxides. The upper part of the Unit includes a higher percentage of the red sandstones which are less consolidated. The Unit is 90-150 meters thick with the thickest part occurring at Limosna.

Aroifilla Formation - Red Sandstone Unit – Upper Cretaceous

This sequence of strikingly red sandstones are 150 to 300 meters thick and unconformably overlie the Intermedia Unit. The Unit is divided into three parts. The lower part is characterized by the presence of a fine silty red sandstone, the middle by the presence of a fine grained brown sandstone and the top by the presence of green shales 10-15 cm thick including intercalations of gypsum.

Chaunaca Formation – Upper Cretaceous

At Malku Khota only the Upper portion of this formation is present and it lies unconformably over the Red Sandstone Unit. The Unit which includes a sequence of multicolored siltstones and shales with evaporitic intercalations of gypsum is 40-50 meters thick. These siltstones can be red, brown, green or purple.

El Molino Formation – Upper Cretaceous

This Formation is comprised of Oolitic limestone with layers of calcareous sandstone and intercalated gypsum. The rocks are typically multicolored including browns, greens and yellows. The Formation is conformable with the Chaunaca Formation and is typically 200 meters thick.

Quaternary to Recent Sediments

Portions of the area are covered by Pleistocene to Holocene glacial moraine which includes fine to very coarse material up to boulder size. The most recent deposits include fine grained material deposited in the glacial lakes and thin soil cover.

Volcanics

The most prominent volcanic activity is located 8 km south of the resource area (Limosna) and consists of a sequence of pyroclastics and dacitic columnar jointed lavas associated with a volcanic dome. It is thought that these volcanics are 20 to 24 million years old (Troeng & Riera, 1996). The dacite is leucocratic with crystals of potassic feldspar, biotite and quartz eyes. No relationship between this volcanic activity and mineralization has been observed.

7.4 Geologic Structure

The rocks exposed in the Malku Khota project area consist of Jurassic to Cretaceous sediments that are unconformably overlain by Cretaceous terrestrial and lacustrine red-bed sediments, evaporates and limestones. All rock units strike approximately N30W and dip to the West at 60-80 degrees except in the area to the West of the resource area where some rocks dip to the East due to folding. The Jurassic and Cretaceous Units within the resource area are bounded in the East and West by N-S trending thrust faults. To the East, the Palaeozoic rocks are in thrust contact with the Jurassic Chiru Khasa Formation and to the West the limestones of the El Molino Formation are in thrust contact with the Palaeozoic. On the western side of the project area the El Molino limestones are folded into N-S trending synclines and anticlines. These folds are not apparent in the more brittle Malku Khota and Wara Wara Formations.

The resource area rocks are cut by a consistent set of NW-SE trending left lateral faults that occur one to three kilometers apart. Offset can be tens of meters to several hundred meters. These fault zones can be wide and include breccias, silicification and veinlets of barite. Nearby rocks are sometimes bleached possibly due to fluid flow along these structures. A second set of fractures and faults occur throughout the area and strike between 50 and 110 degrees and dip towards the North at 50-90 degrees. These structures and fractures are associated with mineralization believed to be related to the later hydrothermal event which is best developed in the Wara Wara and Sucre zones. This series of faults and fractures are the most visually prominent when looking at the ridge that forms the resource area. .

Figure 7-1 Drill Hole Locations

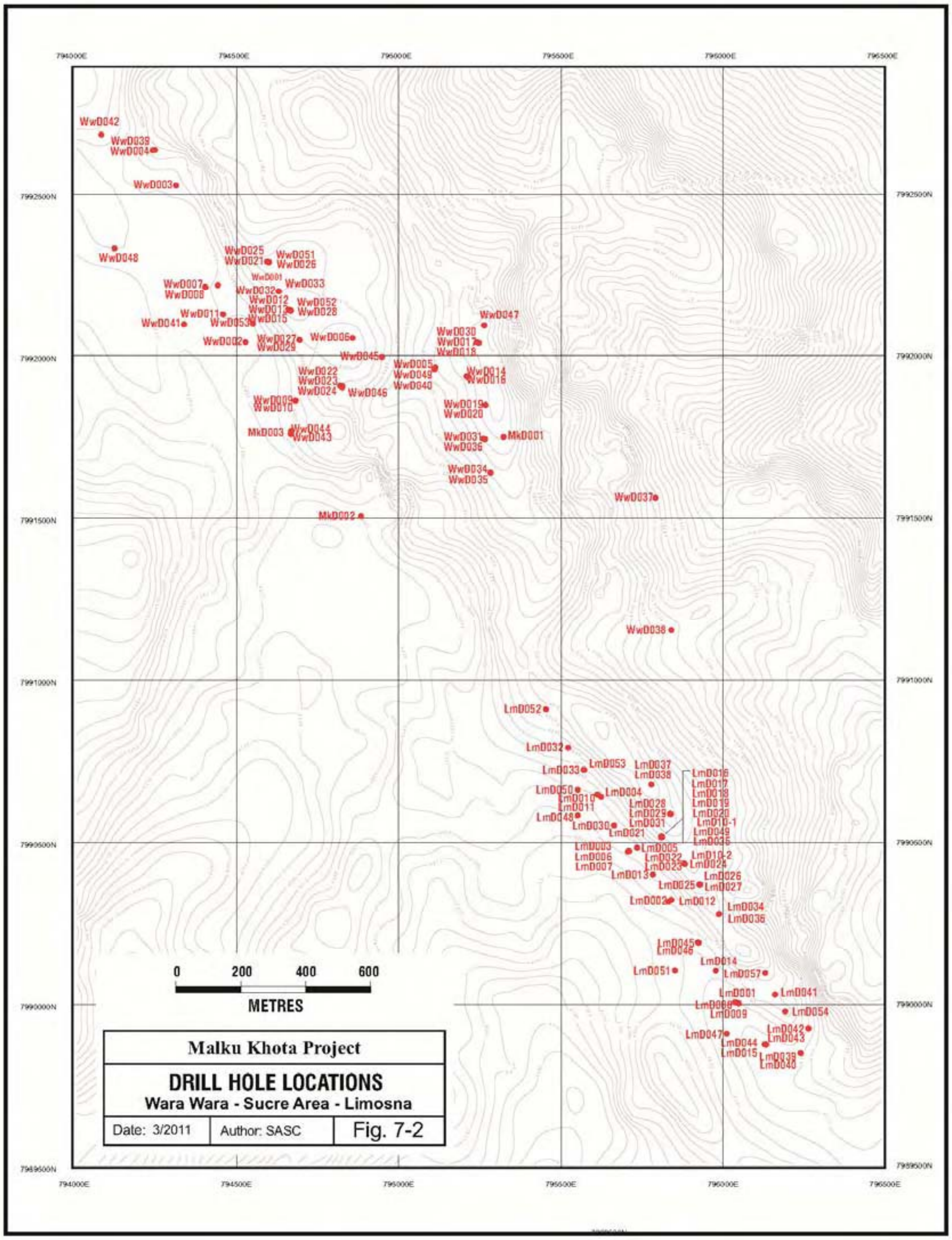


Figure 7-2 Geology Wara Wara-Sucre Area

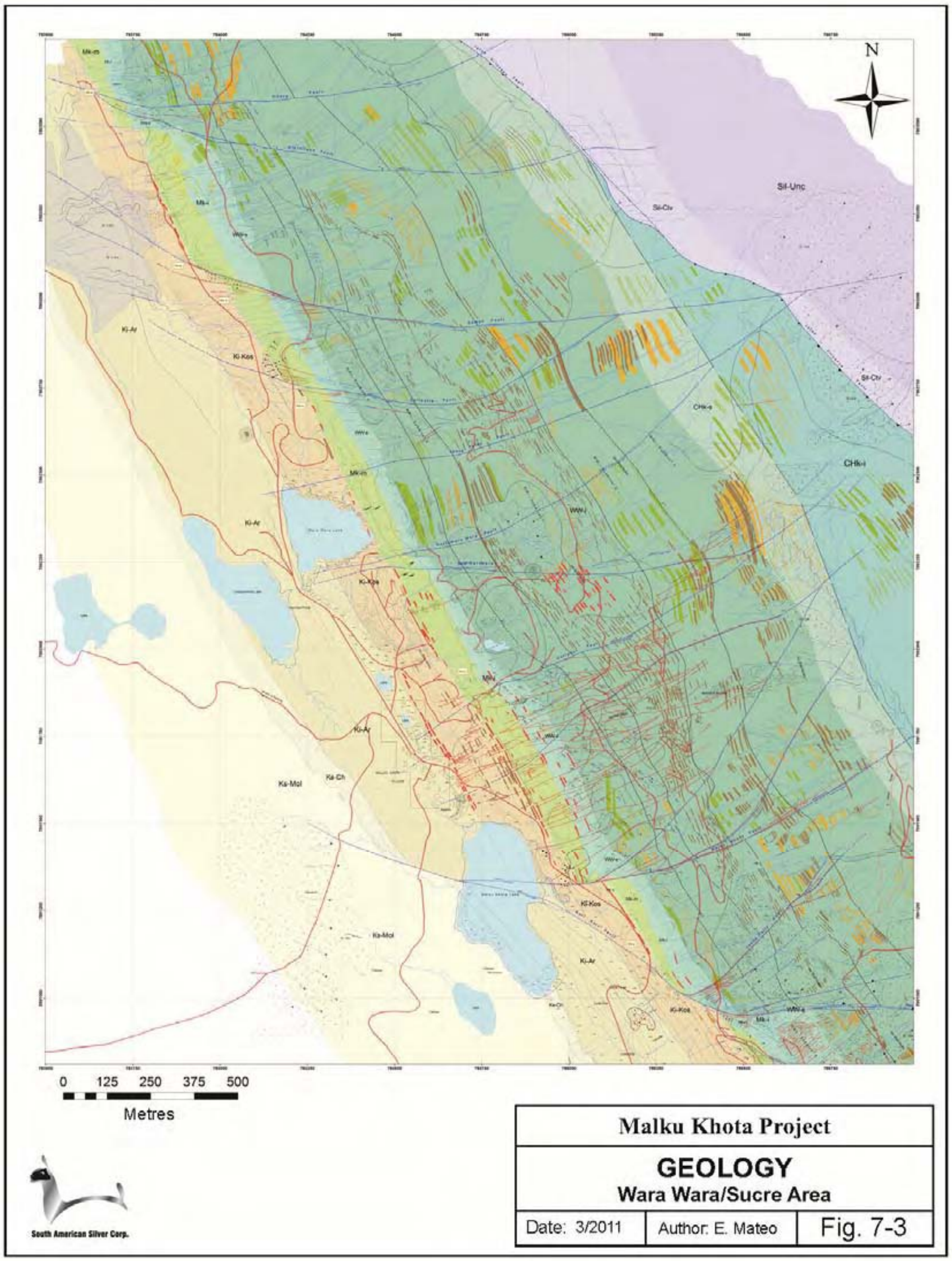


Figure 7-3 Geology Limosna Area

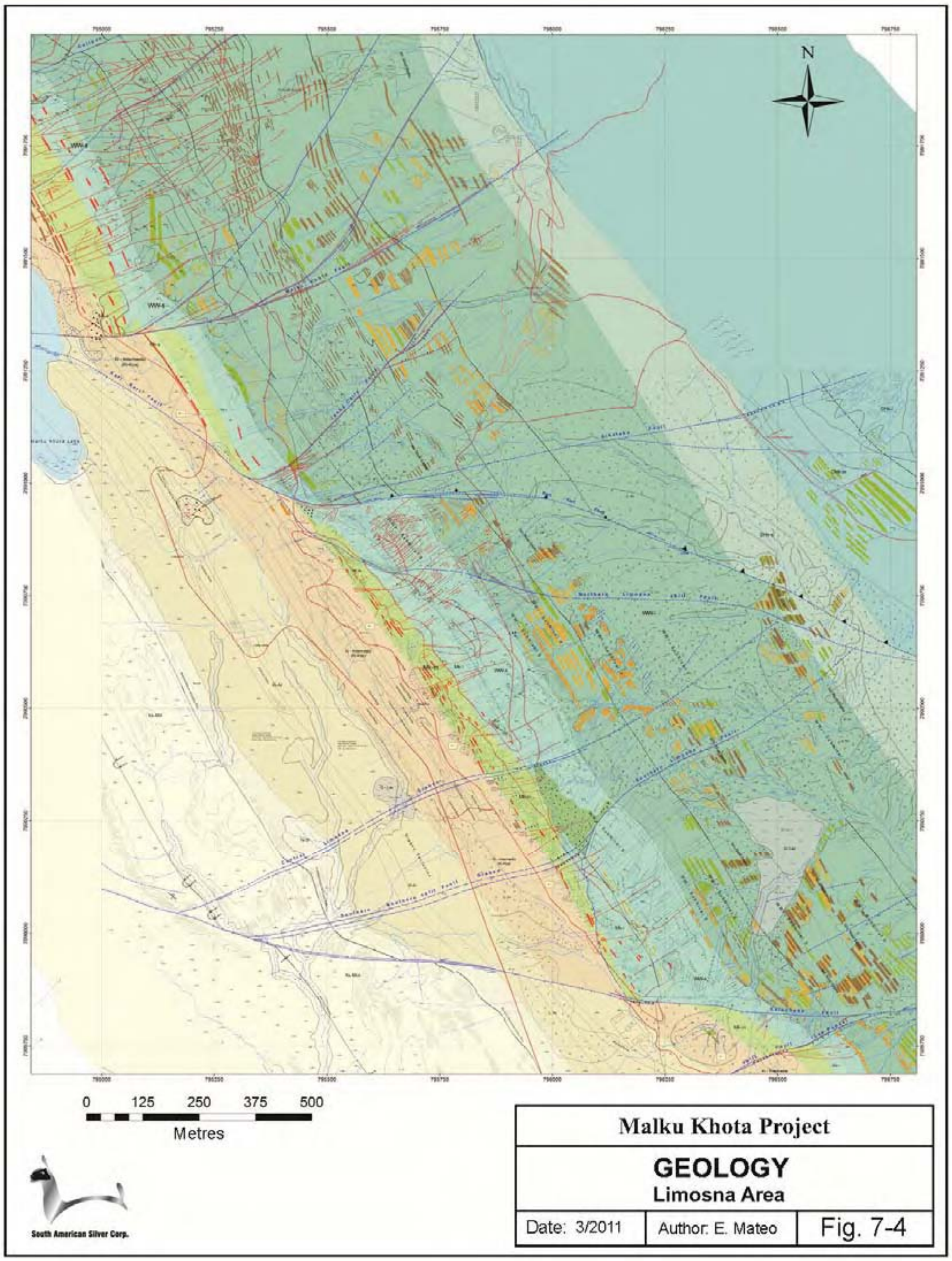
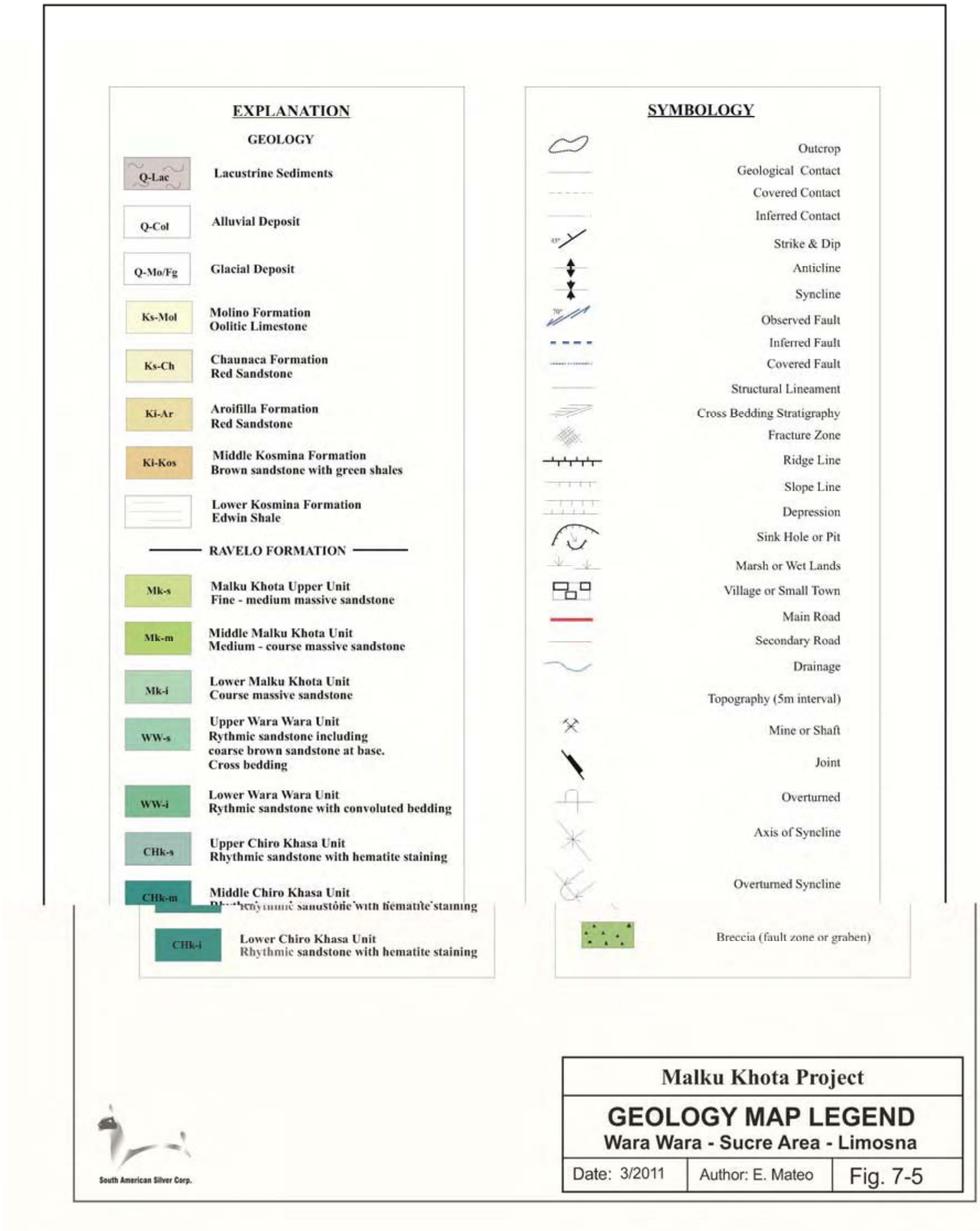


Figure 7-4 Geology Map Legend



8 DEPOSIT TYPES

Geologic mapping and geochemical sampling of outcrops and the results from 121 core holes drilled on the property since 2004 indicate that the Malku Khota property has the potential to host widespread silver mineralization within by sandstone units that extend more than 15 kilometers along strike within the project land position.

Two styles of mineralization have been identified at the Malku Khota property that are associated with extensive silver mineralization. Historic production and previous exploration have focused primarily on high-grade strata-bound lenses within the sandstone units and on narrow gold-bismuth veins that crosscut the earlier stratabound disseminated silver mineralization. Early mineralization is believed to be SEDEX in nature related to rift development between the Jurassic and Cretaceous Periods. Mineralization associated with this phase includes barite, lead, zinc and some silver. A later hydrothermal event related to a hypothesized intrusive hosted "gold system" brought in minor gold mineralization together with new and redistributed silver, lead, zinc, copper and indium/gallium mineralization. Fluid inclusions show that the early mineralization was deposited at temperatures of up to 200-230 degrees and mineralogy that includes tourmaline overgrowths and rare earth minerals suggests temperatures in the 500-600 degree range for the hydrothermal event.

9 MINERALIZATION

The mineralization at Malku Khota is confined primarily to relatively shallow water to aeolian sandstone sequences and is believed to be sedimentary exhalative in origin. Drilling and sampling from 121 core holes and previously reported channel, chip and underground samples indicate that the property hosts a large tonnage of silver mineralization. Mineralized zones range up to 200 meters in true width and extend to the limits of drilling along strike, a distance of approximately 4 km. The SEDEX related disseminated silver mineralization is associated with silver, lead, zinc, and barite. A later hydrothermal event introduced gold and bismuth and additional silver, base metals, indium and gallium. Earlier mineralization may have been partially redistributed by the hydrothermal event.

The majority of the disseminated silver mineralization at Malku Khota is hosted within the sandstones in the Malku Khota and Upper Wara Wara Units. The silver minerals identified in outcrop, tunnels and drill holes are mostly in the form of sulfides (acanthite), complex sulphosalts, oxides, iodides, and bromides often including iron, lead, and antimony in their structures. Ore Minerals include acanthite, jamesonite group minerals (including owyheeite), fizyelite (Pb,Ag,Sb sulfide), Pb,Sb,Ag,Cd sulphosalt, silver, cerargyrite minerals, silver oxide, pyrite, galena, sphalerite, tetrahedrite and tennantite (one sample), copper sulfide (only in a few samples), lollingite (one grain), native bismuth, bismuthinite, arsenopyrite, greenockite, barite, valentinite (Sb ox), stibioconite (Sb ox), bindheimite (Pb,Sb ox), scorodite (AsOx), massicote (Pb ox), plattnerite (Pb ox), tripuhyite (Fe,Sb ox), eriochalcite (Cu chloride), stetefeldite (AgSb hydrate), stibnite, iron oxides, zinc oxide. Sulfides and oxides have a very small grain size and are commonly intermixed. Only under the SEM or with X-ray diffraction could they be determined. Acanthite is distributed in cracks in the sulphosalt, and in microfractures in quartz and barite (Hansley 2011). Metal oxides were clearly derived from the sulfides, the latter occurring as remnants in oxides.

Specific indium minerals have not been found but there is mounting evidence that the indium occurs within the minerals jamesonite and acanthite. SEM analyses in 2006 and chemical assays in 2007 strongly suggest that indium is present in very small amounts in jamesonite and acanthite. One piece of evidence for this conclusion is based on SEM-energy dispersive X-ray analyses done at the Hazen Research lab. One of the samples analyzed was LMD-001020. The SEM system was adjusted to bring out the indium peak and to avoid the interfering cadmium and silver peaks. Using this restricted analytical set-up, some SEM spectra showed a small peak in the Pb,Sb,Fe sulphosalt mineral (jamesonite?) and in acanthite that may be indium. Other evidence are the chemical assays: The sample (LMD01005) with high indium also has high lead, antimony, and cadmium. These data strongly suggest that jamesonite or owyheeite contains indium. Zinc is not present, so sphalerite can be ruled out. Electron microprobe analyses point to the same conclusion. A second series of studies by SGS lakefield in 2009 using BSE and Elemental X-ray mapping of sample SGS08-2 show a positive correlation between silver (acanthite), indium and lead suggesting a similar relationship to that found by Hansley and Hazen Research.

Secondary minerals within the sandstones include specular hematite, hematite, crandallite, kaolinite, illite, strontianite, chlorite, carbonate cement, calcite, rutile, primary and secondary monazite, primary and secondary tourmaline, ferroan carbonate, muscovite, plumbogummite ($PbAlPO_4$), anglesite, anatase.

Many of these secondary minerals including the specular hematite, secondary tourmaline and monazite attest to the much higher temperatures associated with the later hydrothermal event which is best seen in the northern portion of the resource area in the Wara Wara and Sucre zones. The ferroan carbonate is also thought to be an alteration product associated with the hydrothermal event that is believed to be genetically related to an ”intrusive hosted gold system”. The latter speculation is based on the occurrence of the cross-cutting E-W gold/silver bearing veins, the coincident, large somewhat circular bismuth in rock anomaly and the surrounding ferroan carbonate alteration. The hypothesized intrusion has not yet been discovered.

The depositional Sequence of the ore and other secondary minerals shows an alternation of secondary silica stages and sulfides in pores. Proceeding from a quartz grain into the center of a pore, a typical sequence would be as follows: quartz grain, pyrite, jamesonite and(or) owyheeite (contains silver), barite, one or two quartz overgrowths, iron and antimony, lead and silver oxides, jamesonite, chert, chert or chalcedony ending with cerargyrite (silver-bearing) and more oxides in the center of a pore. The barite and two stages of quartz overgrowths are not always present. This sequence indicates an initial fluid geochemistry alternating between alkaline (silica) and acidic (sulfides) and progressively becoming more silica saturated (quartz to chert). Tiny grains of silver, acanthite, silver chloride, freibergite, and rarely gold occur in vugs in quartz and quartz overgrowths. This strongly suggests that a leaching event occurred before precipitation of these elements and minerals.

Most silver precipitated as a lead-antimony-silver-iron-cadmium sulphosalt (now partly oxidized) in alternating bands with silica during the early post depositional history of this sandstone and then was redistributed as a halogen mineral or the native metal during the later hydrothermal event. This is well demonstrated by the paragenesis seen in sample MK17550 which shows: Quartz overgrowths – antimony, lead, and silver sulfides – microcrystalline silica – metallic sulfides – chalcedony – chalcedony – oxidation of sulfides and redistribution of silver (cerargyrite).

Three areas of strata-bound silver mineralization have been identified, the Limosna, Wara Wara and Sucre zones. This mineralization occurs within a strike length of approximately 4 kilometers and widths of 20 to 200 meters but is open along strike and to depth. The original surface channel samples exceeding 10 gpt silver defined an area of approximately 320,000 square meters and silver values within this zone vary from 10 to a few highs exceeding 1,000 gpt silver. A total of 115 drill holes have been completed comprising 40,992 meters through December 2010 have been drilled in the resource area. The deepest drill holes show that the mineralized zone is still present to a depth of up to 500 meters below the crest of the Limosna Ridge. Exploration is currently focused on infill drilling of the resource to upgrade it to measured and indicated. Further exploration drilling is also planned to follow the high grade mineralization found in each of the zones to depth, to explore the area between Wara Wara and Limosna and to explore the footwall to the main mineralized zones where evidence of additional bands of SEDEX style mineralization have been found at the surface.

10 EXPLORATION

10.1 Geochemical Surface Sampling

During 2004 and 2005, General Minerals Corporation completed initial reconnaissance of its original 4,125 hectares property that included geological mapping and the collection of 1,120 chip samples across the silver-bearing sandstone units of the Malku Khota and Wara Wara Formations. These continuous chip samples were taken as a series of 32 “traverse” lines perpendicular across the width of the sandstone units, with the spacing between each “traverse” ranging from 50 to 800 meters apart. This sampling project covered an approximate 15 kilometer strike length of the sandstone units and resulted in the definition of an area of approximately 3,500 meters long by 800 meters wide that exhibited anomalous silver, gold, bismuth and base metal values. Within this area, there is a well-defined zone of 3,450 by 263 meters in which anomalous silver values of approximately 0.5 to 1.0 ounce per tonne were found, including 228 meters that averaged 40 gpt silver. Approximately fifty crosscutting gold-bismuth veins were also identified in this mineralized area.

In 2005, SILEX Bolivia, S.A. (SILEX) entered into a partnership with General Minerals Corporation. In June 2005, SILEX completed a substantial program of surface and underground sampling that confirmed the existence of large widths of disseminated silver mineralization at the surface and in historic underground workings. A total of 1,111 surface and underground samples were collected and the initial surface program focused on Cerro Limosna where SILEX defined anomalous silver values hosted by sandstones along a strike distance of approximately 1.4 kilometers long and varying in width from approximately 30 to 180 meters true width. Additional underground mapping and channel sampling within old tunnels, primarily in the Pique Pobre area, returned a composite with average values of 395 gpt silver over a projected width of 130 meters in the sandstone units. Much of the silver mineralization was observed to begin at or near the surface and follows a dip slope to the west with very little overburden present in many of the areas. During 2005, SILEX continued with the program of surface channel sampling and identified two additional target areas, referred to as the Sucre and Wara Wara areas. The surface sampling programs identified approximately 320,000 square meters in the Limosna and Wara Wara and Sucre areas that averaged greater than 10 gpt.

Geochemical sampling, trench and underground sampling were used to assist and guide the exploration effort but none of the sample results were used in the resource model.

11 DRILLING

11.1 Core Drilling

From May 2007 to December 2010 CMMK completed a total of 42,704 meters of drilling in 121 diamond core holes, which include the drill holes completed by Silex in 2005 and the six exploration holes completed to the South of the resource area. This drilling tested three zones in the resource area called Limosna, Sucre and Wara Wara as shown in Figure 7-1. Drill holes were initially drilled from the base of the ridge formed by the Malku Khota and Wara Wara sandstones, from West to East cutting across the N30W striking stratigraphy and the typically N60E striking vein sets. These holes were typically drilled towards 103 degrees at declinations of 40-60 degrees. In 2008 roads were built along the tops of the ridges such that drill holes could be designed to penetrate the mineralization in the 150 meter high ridge which earlier drilling had drilled underneath. Drilling from the ridge line was orientated both to the East, across strike and to the West sub-parallel to strike and dip. These later holes although not ideally orientated gave useful assays of the upper portion of the ridge which proved important due to the high grades near the surface. These holes although sub-parallel to bedding were at a high angle to the interpreted horizontally distributed secondary enrichment that was interpreted to exist in the upper 100 metres of the ridge. Due to the rugged nature of the topography it was not possible to develop sufficient roads and pads to drill all the holes in an easterly direction.

The results from the core drilling program confirm that silver mineralization is present and in grades of economic interest in the three areas that were tested by drilling and are comparable to the relative widths and grades found in the surface sampling program. Details of the results of the drilling are described in more detail in the section discussing the resource estimation.

11.2 Underground Mapping and Sampling

During 2005, SILEX conducted underground mapping and sampling within old tunnels in the Kallampa, Frio, and Pique Pobre workings in the Limosna area. Results from the Pique Pobre workings showed that high-grade mineralization is still present and this zone returned a composite average value of 395 gpt silver over a width of 130 meters across the bedding in the Malku Khota sandstone unit. The samples collected were channel samples taken from underground exposures a few meters to approximately 50 meters below the surface. The observed mineralization starts at surface and there is no overburden.

Geochemical sampling, trench and underground sampling were used to assist and guide the exploration effort but none of the sample results were used in the resource model.

12 SAMPLING METHODOLOGY AND APPROACH

Much of the information contained in this section was extracted in part from the PAH report and edited by the author.

12.1 Silex 2005–2006

During the SILEX program in 2005–2006, surface and drill samples were collected and supervised by qualified personnel. Samples were transported to Oruro to the preparation laboratory. A portion of the surface samples were delivered to Alex Stewart laboratory (Stewart Group Laboratory) and the remainder samples, including all drill cores, were delivered to ALS Chemex. At these facilities the material was dried, crushed, split, and pulverized. The resulting pulps were analyzed at the Alex Stewart laboratories in Mendoza, Argentina, or one of ALS Chemex laboratories either in Lima, Perú, or in Vancouver, BC.

12.2 CMMK 2007–2010

PAH reported that the procedure used was developed by CMMK, the Bolivian subsidiary of SASC, and was designed to minimize the possibility of sampling and assay errors in the sampling program.

Samples were collected under the supervision of the geologist in charge of the mapping and sampling program, who ensured the quality of the samples taken and confirmed that the samples were correctly numbered.

The following is a description of the sampling procedures conducted by CMMK, observed by Ross Conner of PAH during the drilling program noted on his site visit of November 2008. The sampling procedures did not change much since the November 2008 site visit, and minor edits were made to this procedural description by this author to reflect these changes as a result of the more recent November 2010 site visit.

At the drill rig, the core is recovered by the driller and placed into wooden boxes. The driller is responsible for placing the core in the box, marking the intervals with wooden markers to designate footage in meters, and numbering the box.

The core boxes are collected daily by a CMMK driver and brought to the village of Sakani, where they are unloaded and stacked under the supervision of a CMMK geologist who re-marks each box with the DDH number and downhole direction, and conducts a visual inspection of the core.

Each box is then examined by the geologist and is logged on a pre-printed paper form for structures, fractures, and rock quality controls (RQC). The core is then passed to the senior geologist and is logged for rock type, contacts, alteration, mineralization, and veining. At this stage, the senior geologist selects the intervals to be sampled. The individual sample interval is marked on the core box with a red felt marker, and a sample tag is stapled to the box below the marker as shown in Figure 12-1.

Figure 12-1 Core Box Ready for Sampling

Once the core has been completely logged by the senior geologist, each core box and contents is photographed and then stacked for splitting by diamond saw. The core is split for sampling by a technician using a diamond saw. The saw uses a continuous stream of fresh water for cooling fluid. Once split, both halves of the core are placed back into the core box and the CMMK geologist takes the sample from the split portion, labels it, bags it, and prepares a sample submittal sheet for ALS Chemex labs in Oruro, Bolivia.

A CMMK driver delivers the bagged samples twice a week to the ALS Chemex preparation facility in Oruro. At the prep laboratory, the samples are dried, crushed, and pulverized, and a pulp extracted and sent to Lima for analysis.

All samples are analyzed for silver and indium. The majority are also analyzed for gold and 48 additional elements using ICP.

PAH reported that surface channel and chip samples were typically 2 to 4 m long with a maximum of 10 m and minimums of less than 2 m where local geologic features were being investigated. Sample lines were spaced at approximately 100 to 200 m intervals and oriented to cross the host sandstone approximately at right angles to the strike direction. This detailed sampling was carried out in an area measuring approximately 4 km long by 800 m wide.

Underground tunnel samples were typically from 2 to 3 m in length with local geologic features sampled at less than 2 m. These samples were collected prior to the November 2010 site visit, and the sample sites were not visited since none of the surface channel samples or underground tunnel samples were used in the resource estimation.

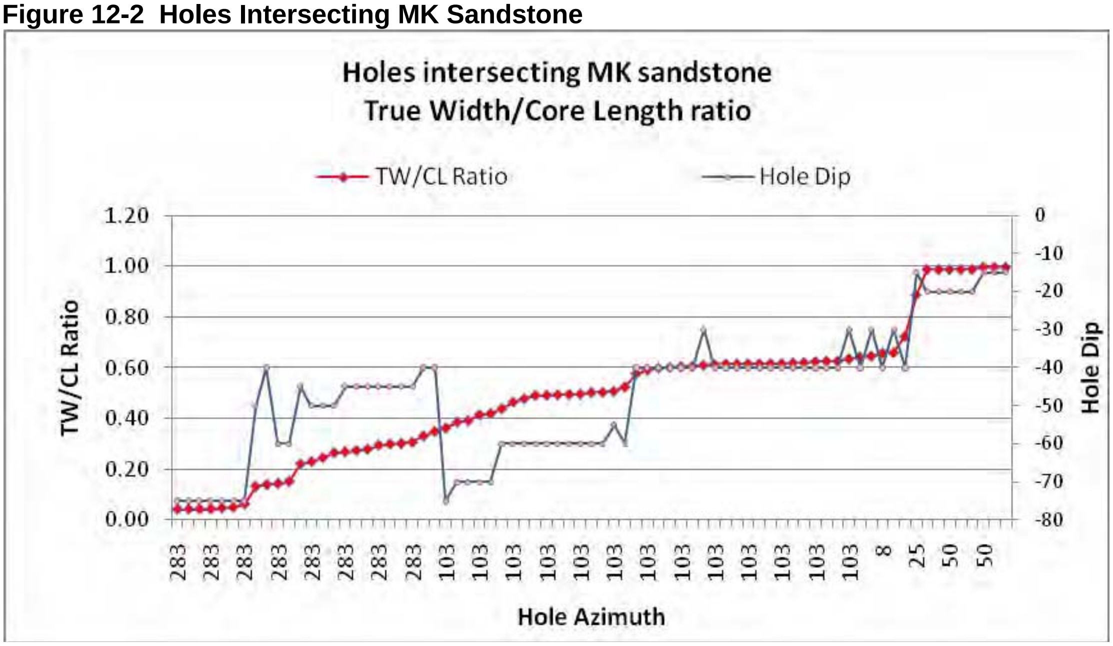
Drill core is split with a diamond saw and typically sampled at 2-m intervals or less due to local geologic features. Sample length averaged 1.67 m with a third quartile of 2.0 m and first quartile of 1.5 m. Core recovery was excellent, with near complete recovery except in rare structural zones. PAH's review of core photographs confirms this observation, and the author noted during the site visit that recovery was excellent in the holes reviewed. The width of the high-grade silver mineralized intercepts typically exceeds 10 m true width so the sampling interval of 2 m or less is considered by the author to be appropriate.

The drill holes were irregularly spaced along an approximate 3.2 km strike length in the sandstone host rock. Drill hole spacing in the Limosna and Wara Wara Zones is on a nominal 100 to 200 m grid spacing, but is highly irregular at depth because the holes were drilled in fans.

There are two main orientations for the mineralization on the property. The bulk of the mineralization is oriented at 323 azimuths with a dip of 78 degrees to the southwest. A second style of mineralization, located primarily in the Sucre area is oriented between 40 to 60 degrees azimuth dipping at approximately 80 degrees in a north-northwest direction. Because of this, CMMK drill fences crossed the main strike of the deposit at a 45 degree angle in an attempt to intersect both styles of mineralization.

AGP examined the 75 holes (out of 112) that completely or partially crossed the MK Sandstone unit in order to assess the drill angle versus the azimuth of the main mineralized horizon. The ratio between the true width and the core length of the intersection was calculated and plotted on a chart displayed in Figure 12-2. A high ratio indicates that the drilling angle is more oblique to the zone. A ratio of 1.0, while ideal, is rarely seen since holes are typically drilled in fans originating from a single set-up. A ratio of 0.4 or less is considered less than ideal, since the holes are drilled sub-parallel to the mineralized zone and holes showing a ratio of less than 0.2 should be avoided. The data shows that 27 out of the 75 drill holes examined (36%) were drilled sub-parallel to the main mineralized horizon with 11 of those with a ratio less than 0.2 (14%) and intersected the mineralization at a low angle.

Figure 12-2 Holes Intersecting MK Sandstone



During the site visit it was noted that the topography is steep and at high elevation, making it difficult for CMMK to build drill pads that allow consistent drilling on an oblique angle to the zone. AGP still recommends that CMMK take every effort possible to reduce or eliminate drilling sub-parallel to the zone.

13 SAMPLE PREPARATION, ANALYSES & SECURITY

SASC has conducted systematic programs of drilling, sampling, and analysis since it optioned the Malku Khota property as explained in Section 4.

13.1 Silex 2005–2006

During the Silex program, samples were dried, crushed, pulverized and split before being transported to the Alex Stewart laboratories in Mendoza, Argentina or the ALS Chemex laboratories in Lima, Peru or in Vancouver, BC. Samples were assayed for silver by gravimetric analysis and ICP, and gold was analyzed by gravimetric means. Associated elements including lead, zinc, bismuth, antimony, and barium, among others, were analyzed by ICP. Two types of ICP were performed on the majority of samples: one using a 3-acid digestion and the other a more rigorous 4-acid digestion. The laboratory results showed that the latter digestion tended to give higher values by about 10%, which was not considered by Silex to be significant at the exploration stage of project development. The Alex Stewart laboratories in Oruro and Mendoza are both ISO 9001:2000 certified.

13.2 CMMK 2007–2010

Samples collected prior to 2007, were transported by CMMK personnel to the ALS Chemex sample preparation laboratory in Oruro, Bolivia, which did not have an ISO certification at the time. The prepared pulp samples were then analyzed at ALS Chemex laboratory in Vancouver, an ISO 9001-2000 certified laboratory. The analysis for gold was by fire assay (FA) using a 30 gram sample split. Other elements are assayed by ICP with a 24 element package. Five percent of the samples were re-analyzed for control purposes at ACME Laboratories in Santiago, Chile, and/or Vancouver, BC.

The 2007–2010 program followed similar procedures, with sample preparation conducted by ALS Chemex in Oruro, Bolivia, and analysis conducted in their ISO 9001-2000 laboratory in Lima, Peru. Forty-eight elements were analyzed by the ICP MS61 method using a four acid digestion. Silver, lead, zinc, and copper values greater than the limit of detection were reanalyzed by the ALS Chemex AA62 method using a 4-acid digestion. Silver assays greater than 1,500 gpt were analyzed by the 30 gram FA-GRAV method. Gallium and indium were included in ICP MS61. Gold was analyzed by fire assay using a 30 gram split using either the ICP21, AA25 or ST44 methods. No gold over limit values were encountered in the data reviewed.

When the results were received they were checked for their geologic reasonableness by the site project manager, and the field locations were cross-referenced with assay sheet sample numbers to check accuracy. In SASC's Denver offices, the assay certificates were again checked against the geologic sample log index and input to their digital database.

PAH, as part of their due diligence, reviewed CMMK's internal procedures for analytical data input and record keeping at their Denver offices, and found them to meet appropriate standards for data input, security, and checks. AGP did not review the procedure in place at the Denver office and relies on the observations made by PAH.

13.3 Security

CMMK maintain a field office core storage area located in the village of Sakani approximately 12 km northeast of the project area. All core is stored in a locked storage shed within a fenced compound. The core cutting facility is located within the same fenced area, inside a building constructed specifically for that purpose. During the site visit by the author, it was apparent that CMMK has full chain of custody for the samples that are collected on site.

13.4 Assay Quality Control

Quality assurance/quality control (QA/QC) for the drilling programs consisted of inserting field "duplicate" samples and a blank pulps. No analytical standards are inserted in the sample chain, and the author strongly recommends CCMK add this protocol to the QA/QC procedures. SASC should investigate the creation of custom made low- and high-grade standards using Malku Khota mineralized material, since a quick search for a commercially available standard did not reveal any source that included indium.

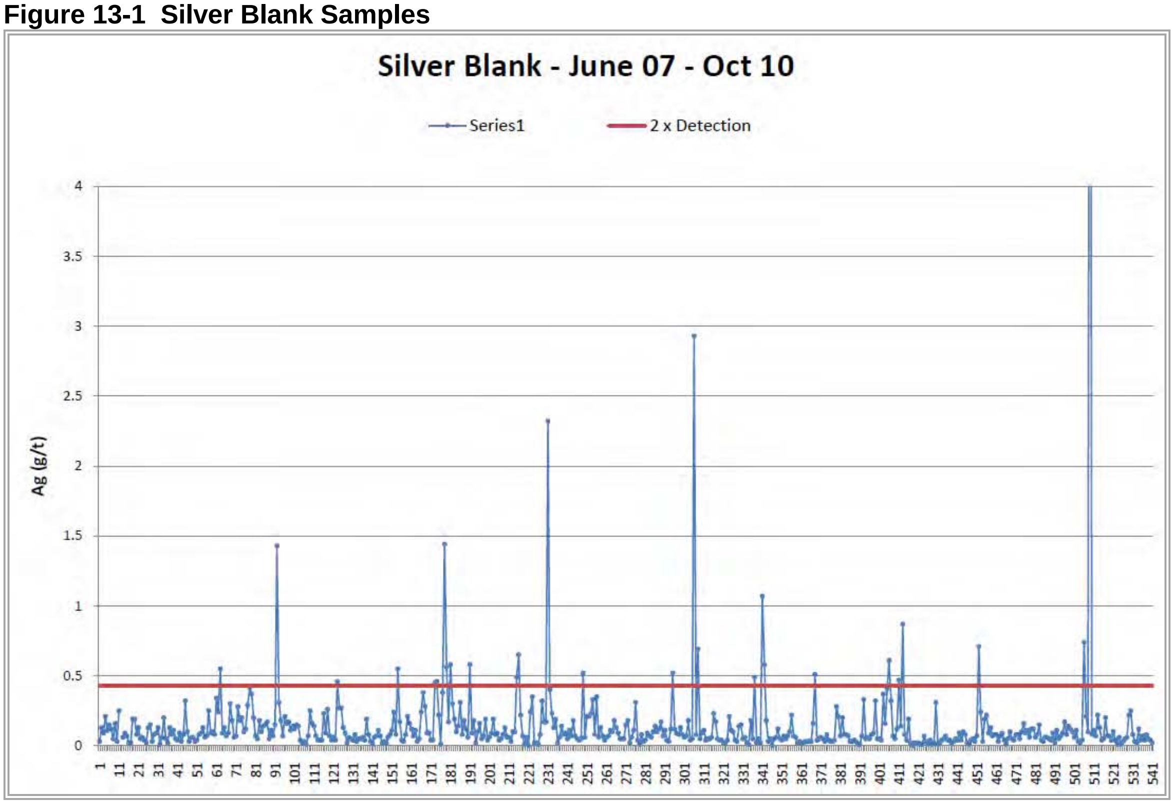
Insertion rate for the field duplicates was approximately one every 20 m in the sample runs with one blank sample submitted in every batch of 40 samples.

13.4.1 Blanks

Blank material consists of a pulp that originated from the low-grade source on the Malku Khota property.

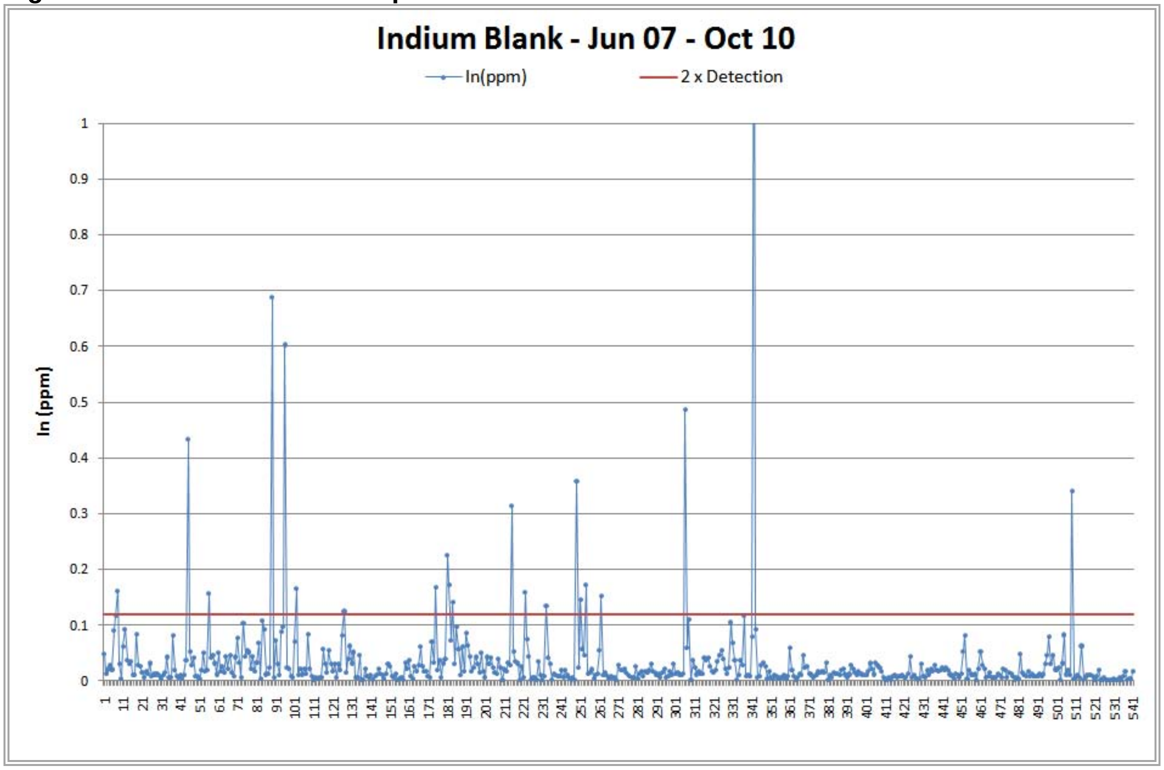
Blanks are inserted into the ALS Chemex sample stream to determine if contamination has occurred during sample preparation or analysis. The material used by CMMK is not a true blank and averages 0.13 ppm silver with a standard deviation of 0.21 after removing one outlier. Using twice the standard deviation of the silver data set, 26 failures out of a total of 541 blanks occurred amounting to 4.8% of the blank submitted (Figure 13-1).

Figure 13-1 Silver Blank Samples



For indium the average grade is 0.031 ppm with a standard deviation of 0.059 (Figure 13-2). Using twice the standard deviation of the indium data set, 20 failures occurred out of a total 541 blanks amounting to 4.8% of the blank submitted.

Figure 13-2 Indium Blank Samples



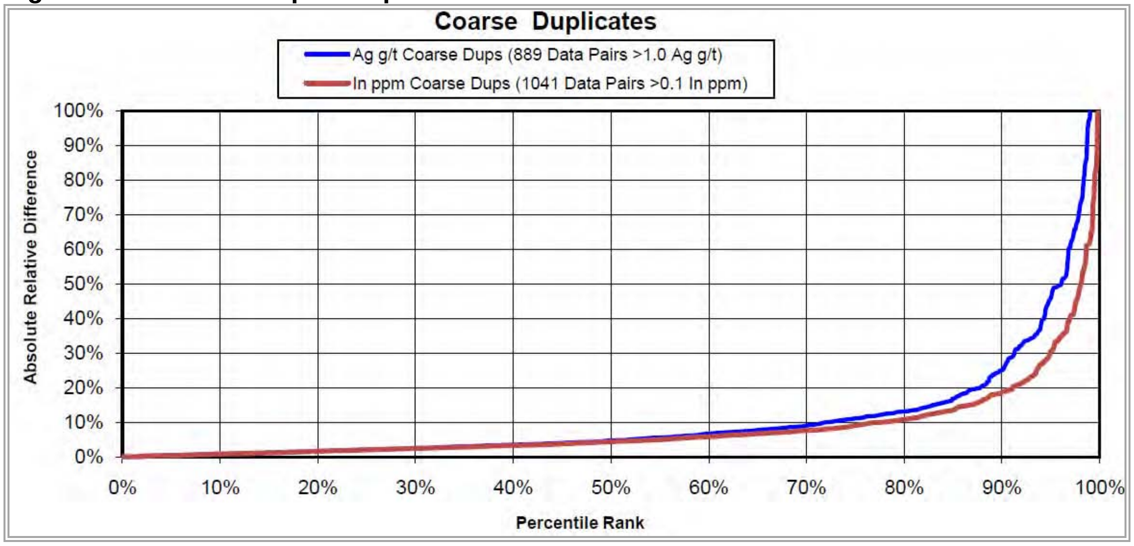
AGP notes that since the material is not a true blank the term "failure" is loosely applied in this context, and for that reason AGP recommends CMMK using a different crushable material for blanks that consistently returns assays below the detection limit.

13.4.2 Field Duplicates

Field duplicate consist of 1/2 core samples that have been crushed by a 2 lb sledgehammer over a steel plate and riffle split into two portions. One portion bears the original tag, and the second is marked with the duplicate tag. Field duplicates are inserted to verify the precision of the analysis.

For field duplicates it is generally accepted that 90% of the data should be within ±20% of the original value. For silver, 90% of the data greater than 1 g/t are within ±23% of the original value. For indium, the data shows that 90% of the assays above 0.1 ppm are within ±18%, as shown in Figure 13-3

Figure 13-3 Coarse duplicate percentile rank

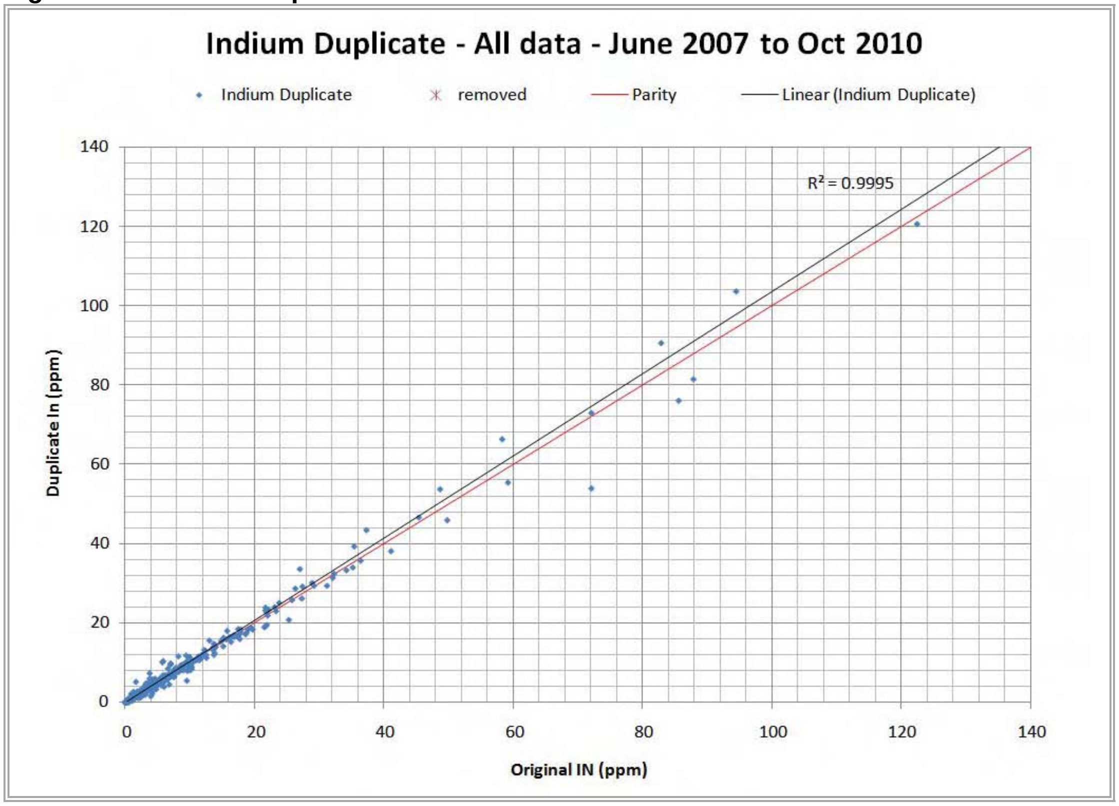


A slight bias is noted; however, the R2 value of the regression line is very good at 0.99, as shown in Figure 13-4.

Figure 13-4 Silver Coarse duplicate

For indium the regression line is similar; a slight bias is noted with a R2 value of 0.99, as shown ion Figure 13-5.

Figure 13-5 Indium Duplicate



13.5 Opinion on Adequacy of Sample Testing

In 2009, PAH commented on the adequacy of the sampling at the Malku Khota deposit. From the review of the literature and documentation on the project, PAH found acceptable results from analytical work done by previous operators who collected their samples according to accepted standards and practices at the time of their campaigns. For the more recent campaigns (CMMK, SILEX and SASC), PAH was satisfied with the adequacy of the sample preparation and analysis but noted some evidence of analytical bias in the results obtained from duplicate and blank analysis.

AGP's review of the QA/QC echoes the findings from PAH. Laboratory standard material should be included in the QA/QC protocols. The blank material currently used at the site should be replaced with material that consistently assays less than the laboratory detection for both silver and indium.

The current procedure used by CMMK for the preparation of duplicates will yield better results than the 1/4 core duplicates used at other operations, since the material originates from the same half-core samples and is mixed. An air source should also be provided in the preparation area of the coarse duplicate samples to allow for cleaning of the riffle box and steel plate used in the crushing operation in order to avoid the possibility of cross contamination between duplicates.

AGP believes that the sampling and analysis at the Malku Khota deposit was generally conducted using standard practices, providing reasonable results. AGP believes that the resulting data is suitable for resource estimation purposes.

14 DATA VERIFICATION

CMMK and SASC geological staff have made a strong commitment to the geological and assay database and have, as far as is possible, produced a database that is complete and well documented.

14.1 Summary of validation by PAH

The property was originally visited by Mr. K. Katsura of PAH as part of the 2009 NI 43-101 due diligence. During his visit, independent check samples were collected and processed at ALS Chemex laboratory in Oruro, Bolivia, the same laboratory used for the CMMK samples. Results from the check samples were found to be comparable to the original values. In addition to the site visit, PAH validated the assay database and reported that in the existing Malku Khota drill hole database provided by SASC, very few errors were identified. PAH also conducted several checks of the existing database compared with older maps and records, which included spot checks of assay certificates and data records. PAH at that time found no errors on the 2004–2008, certificates relative to the existing database.

14.2 Site visit by AGP in 2010

Mr. Pierre Desautels visited the Malku Khota deposit between 11 and 13 November 2010, accompanied by Gordon Zurowski, principal mining engineer at AGP, Mr. Ralph Fitch, Executive Chairman & Director, Felipe Malbran, Vice President of Exploration and Phillip Brodie-Hall, Vice President of Project Development, all of SASC. Two members of the Board of Directors of SASC were also present, Mr. Peter Harris and Ms. Tina Woodside. Three drill rigs were currently active during the site visit, and core logging and sample collection activities were in progress.

The 2010 site visit entailed the following:

- overview of the geology and discussion of possible dump and tailing locations was held at the CMMK office in La Paz.
- current exploration program design (drill hole orientation, depth, number of holes, etc.)
- surveying (topography and drill collar)
- visit of the core logging facility and camp
- discussion of sample transportation and sample chain of custody and security
- core recovery
- QA/QC program (insertion of standards, blanks, duplicates, etc.)
- review of diamond drill core, core-logging sheets and core logging procedures. The review included commentary on typical lithologies, alteration and mineralization styles, and contact relationships at the various lithological boundaries.
- density sample collection.

Other than the Limosna and Wara-Sucre deposits, none of the other targets or regional exploration areas were visited.

During the 2010 visit, AGP collected 3 half-core character samples and retained full custody of the samples from the project site to La Paz, Bolivia, where the samples were shipped to AGP's head office in Barrie, ON. The samples were inspected and then shipped to Activation Laboratories Ltd., in Ancaster, Ontario. The main intent of analyzing these samples was to confirm the presence of silver, indium, copper, zinc, lead, and gallium in the deposit by an

independent laboratory not previously used by CMMK. Samples were crushed, split, and pulverized with mild steel, leaving a 100 g pulp (Activation prep code RX-2). Samples were analyzed for silver, copper, lead, and zinc via a 4-acid digestion followed by ICP-OES. For indium and gallium, samples were analyzed via a sodium peroxide fusion, acid dissolution followed by ICP/MS.
From the assay results shown in Table 14-1, AGP concluded that the general range of values returned by the character samples collected during the site visit correspond well with those reported by CMMK.

Table 14-1 AGP Character Sample Results

Sample	AGP	SASC	AGP	SASC	AGP	SASC
Sample Number	00939	87846	00940	81541	00941	88694
Ag (ppm)	116	127	56	56.3	25	27.4
Cu (%)	0.052	0.111	< 0.001	0.004	0.038	0.047
Zn (%)	0.042	0.053	0.018	0.020	0.007	0.019
Pb (%)	0.345	0.377	0.118	0.116	0.072	0.068
Ga (ppm)	1.4	1.3	2.9	2.9	5	4.7
In (ppm)	2.8	2.7	0.5	0.6	23.3	25.1

CMMK uses a sample tag system consisting of three parts where the main part goes in the sample bag, the second is stapled to the core box, and the third part is retained. Location of specific gravity samples were indicted on the core box with a felt marker.

At Limosna, geologists responsible for logging the core can roughly estimate the high/low grade of the core in the field by the presence of darker golden brown beds. At Wara-Sucre the core is silicified, rendering the visual estimation of the grade difficult. A Niton XL3T portable XRF analyzer is available for exploration purposes, but seldom used in the core logging procedures.
The core is continuously sampled from the collar of the hole to the toe.

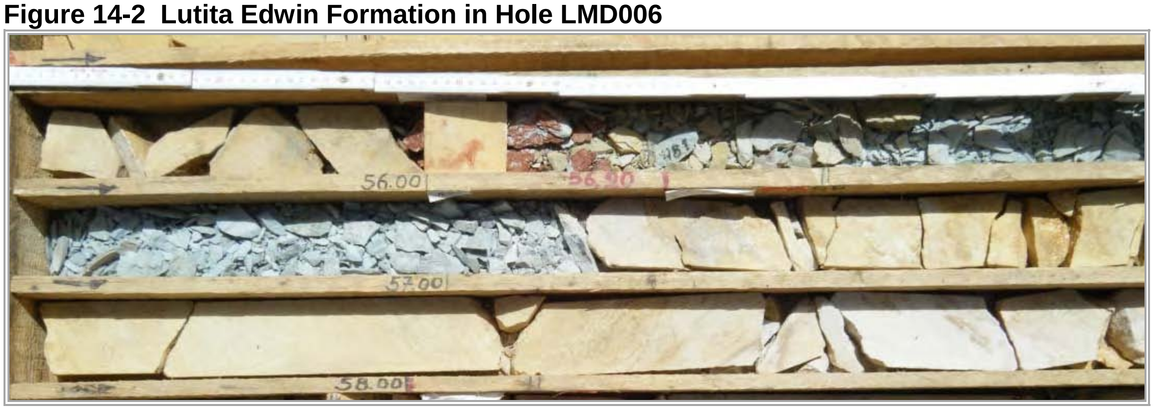
Malku Khota (MK) sandstone is generally a massive unit lacking the cross-bedding features commonly observed in the Wara Wara (WW) sandstone units. This lack of cross-bedding features is not always obvious at the scale of the drill core; however, a sharp increase in the aluminum assay is a good indicator of the contact between the MK and WW sandstone. Coarse-grained quartz banding often occurs in the vicinity of the contact.

Sandstone in the core was found to be coarse-grained and very porous, as illustrated in Figure 14-1. Since petrographic studies of selected samples collected from mineralized intervals of surface samples show that silver minerals have been deposited in the pore space between quartz sand grains, there is a possibility that silver is lost during the drilling and core cutting process. CMMK routinely assays the material left by the saw kerf in order to evaluate the material lost in the sample preparation.

Figure 14-1 Typical Porosity at Limosna LMD048 @ 468.30 m

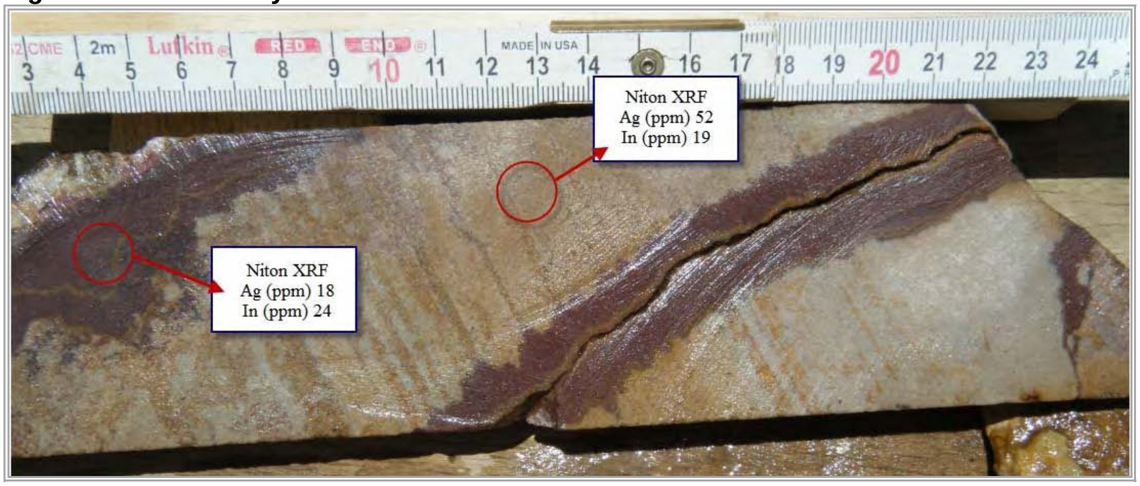
A marker bed, the "Edwin Shale," or Lutita Edwin, is a tuffaceous shale unit, light greyish blue in color unit, occurring at the contact between the Malku Khota sandstone and the overlying Intermediate Unit sediments. With the exception of gallium, mineralization is almost fully restricted to the footwall of this marker bed.

Figure 14-2 Lutita Edwin Formation in Hole LMD006



The Wara Wara – Sucre deposit is characterized by dark brown cross-bedded zones referred to as Sucre-style mineralization. The bands show evidence of bleeding into the sandstone unit. Three samples were selected and analyzed using the Niton XRF unit. For each sample the unit was pointed at the Sucre-style mineralization and then pointed at the surrounding sandstone. Results were inconclusive, with silver and indium occurring at various proportions in both styles of mineralization on the samples selected, as shown in Figure 14-3. Because of the possibility of having a significant effect on the methodology used for the resource model, AGP recommends CMMK to analyze a number of samples with the portable XRF analyzer in an attempt to establish if the grade in the cross-cutting structure is statistically different than the grade in the surrounding sandstone.

Figure 14-3 Sucre-Style Mineralization in Hole WwD014



Specific density procedures were reviewed during the site visit along with the equipment used for the determination. CMMK discontinued the collection of specific gravity samples after hole LmD047 at Limosna and hole WwD039 at Wara-Sucre. AGP recommends CMMK to re-instate this practice for any holes targeted at extending the resources, especially holes drilled to the North and south of the known mineralized horizon.

Figure 14-4 displays a series of photographs taken during the site visit.

Figure 14-4 Site Visit Photos

Drill rig on LMD056

Core cutting in progress

HoleWwD022 casing cap

Core storage facility

General view of the core facility

Blank Material

14.2.1 Database Validation

AGP carried out an internal validation of the drill holes in SASC's Malku Khota database used in the February 2011, resource estimate. Holes were selected for validation according to the following criteria:

- highest silver grade
- highest average grade
- distribution in the deposit
- representative selection based on the drilling year.

A total of 25 holes were partially or completely validated amounting to 4,260 individual samples out of a total of 22,684. All samples were validated against the electronic version of the certificate provided directly by the issuing laboratory without CMMK's intervention. The validation rate amounted to 23% of the drill holes and 19% of the assays.

Table 14-2 shows a list of the holes selected for validation.

Table 14-2 Hole Selected for Validation

Hole-ID	Assay Count in Database	Assay Validated	Fully Validated
LmD006	178	178	yes
LmD007	262	62	no
LmD022	165	165	yes
LmD023	150	150	yes
LmD024	182	182	yes
LmD035	190	140	no
LmD040	263	89	no

Hole-ID	Assay Count in Database	Assay Validated	Fully Validated
LmD045	269	269	yes
LmD048	296	296	yes
LmD049	150	150	yes
LmD050	345	345	yes
LmD051	294	294	yes
MkD001	139	3	no
MkD002	152	4	no
WwD006	87	3	no
WwD019	212	95	no
WwD020	264	118	no
WwD021	229	50	no
WwD024	321	321	yes
WwD035	318	81	no
WwD036	261	65	no
WwD040	220	220	yes
WwD041	268	268	yes
WwD047	358	358	yes
WwD048	354	354	yes
Total	**5,927**	**4,260**	

14.2.2 Collar Coordinate Validation

Collar coordinates were validated with the aid of a handheld Garmin GPSMAP model 60CSx. A series of collars were randomly selected and the GPS position was recorded. The difference with the Gems database was calculated in an X-Y 2D plane with the following formula:

$$\mathbf{X - Y\ difference = \sqrt{(\Delta East)^2 + (\Delta North)^2}}$$

As shown in Table 14-3, the results indicated an average difference in the X-Y plane of 4.4 m and -18.4 m in the Z-plane for the 6 hole collars where the instrument was located on the top of the casing. The calculated differences in the X-Y plane are well within the accuracy of the hand held GPS unit used. The instrument reported "accuracy" between 2.5 m to 3.8 m at most field locations surveyed, which is typically influenced by the number of satellites active at the time and day. Elevation difference is normal, as handheld GPS devices are notoriously inaccurate in elevation.

Table 14-3 Collar Coordinate Verification

Database Entry				GPS Point Recorded during Site Visit				Differences between Database Entry and GPS	
Hole-ID	**East**	**North**	**Elev.**	**GPS Point**	**East**	**North**	**Elev.**	**X-Y Plane (m)**	**Z Plane (m)**
LmD030	795666.7	7990551.9	4245.8	Lmd030	795668	7990553	4270	1.7	-24.2
LmD048	795554.0	7990583.0	4234.0	Lmd048	795559	7990585	4252	5.4	-18.0
WwD020	795268.4	7991851.4	4402.7	Wwd020	795265	7991855	4427	5.0	-24.3
WwD022	794826.0	7991904.0	4313.0	Wwd022	794820	7991912	4323	10.0	-10.0
WwD024	794827.2	7991909.2	4312.0	Wwd024	794828	7991911	4322	2.0	-10.0
WwD036	795262.8	7991746.8	4388.3	Wwd036	795262	7991749	4412	2.3	-23.7
LmD055	795816.0	7990010.0	4226.0	Lmd055	795820	7990017	4222	8.1	4.0
LmD056	795823.0	7990172.0	4227.0	Lmd056	795823	7990178	4229	6.0	-2.0

14.2.3 Down-Hole Survey Validation

The down-hole survey data was validated by searching for large discrepancies between dip and azimuth reading against the previous readings (Table 14-4). A total of 450 readings (the entire database) were evaluated. No holes were collared vertically, as the steepest dip is -82 degrees. The absolute differences in azimuth and dip and degree-per-meter changes were calculated from reading against the previous readings.

Any azimuth with a difference exceeding 10° combined with a change per meter in excess of 1° was considered suspect.

For dip measurement, any dip differences exceeding 1° combined with a change per meter in excess of 1° was flagged as suspect.

Table 14-4 Down-hole Survey Validation Results

For Angle Holes	Azimuth Diff.	Azimuth Diff. (ABS)	Dip Diff.	Azimuth Change per m	Dip Change per m
Min	-10	0	-7	-	-
Max	20.6	20.6	18.9	0.767	1.000
Average	0.153	1.710	-0.448	0.018	0.011
First Quartile	-1	0.325	-1	-	-
Median	0	1	-0.1	0.007	0.004
Third Quartile	1	2	0	0.020	0.013
97 percentile	6	6.639	2	0.077	0.038
99 percentile	10.959	10.959	4	0.190	0.052

Results indicated two azimuths and one dip reading required further validation. Following review, no data required correction, since the suspected data only influenced the last few meters of drill holes WwD038 and LmD049. All other entries were considered correct as the database entry coincided with the down hole survey instrument value.

14.2.4 Assay Validation

The validation against the electronic version of the certificates consisted of comparing the values on the certificate against the CMMK database entry.

Laboratory certificates for the holes selected were requested from ALS Chemex Laboratories Ltd. (CMMK's principal laboratory) in Vancouver, BC, via Felipe Malbran of SASC. ALS Chemex issued the requested certificates as a series of text files in CSV format directly to the AGP e-mail address without CMMK's interaction. A total of 4,822 assay results covering 25 drill holes were compiled from the certificates into an Excel spreadsheet and matched against the sample number in the CMMK drill database. A total of 562 QA/QC assays or surface samples did not find a matching number in the drill database, while the remaining 4,260 sample numbers were successfully matched.

Results show that out of the 4,260 samples reviewed, none were entered erroneously in the database. Two silver assays had the AG-AA62 value entered in the database (Ag by HF-HNO3-HClO4 digestion with HCl leach, ICP-AES, or AAS finish) despite the fact that a gravimetric finish was available. Assays below the laboratory detection limit were consistently entered in the database by reducing the detection limit value by a factor of 10. For example gold values <0.001 were entered in the database as 0.0001.

The assay validation covered 19% of the entire assay database. A total of 14 drill holes (13%) in the database were fully validated, and 11 additional holes were partially validated.

The error rate in the CMMK drill database was found to be non-existent (Table 14-5). The Qualified Person regards the sampling, sample preparation, security, and assay procedures as adequate to form a basis for resource estimation.

Table 14-5 Assay Validation Results

	Verified	Ag Errors	In Errors	Pb Errors	Cu Errors	Zn Errors	Ga Errors
Total validated	4,260	0	0	0	0	0	0
Total assays in DB	22,684	-	-	-	-	-	-
Percent checked/error	19%	0%	0%	0%	0%	0%	0%

15 ADJACENT PROPERTIES

This section is a verbatim extract of the 2009 PEA prepared by Pincock Allen & Holt.

South American Silver Corp. (SASC) controls 5,475 hectares of exploration mining concessions (Cuadriculas) through its wholly owned subsidiary Compañia Minera Malku Khota S.A. ("CMMK"). The block of concessions extends twenty-one kilometers in a northwest–southeast direction. Only an area amounting to approximately 20 percent of the concession area has had any drilling and mineral evaluation. SASC controls all of the significant mineral potential in the zone of permeable sandstones of the Wara Wara and Malku Khota sandstone units. No other significant mineralization is known outside of the immediate SASC controlled ground.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Metallurgical Testwork

SASC is developing an acid chloride based leach process to extract silver and indium from the Malku Khota deposit in Bolivia. Metallurgical studies on the Malku Khota Project were initiated by SASC in 2007. The testwork was undertaken by the Metallurgical Operations Group, SGS Minerals Services, Lakefield, Ontario (SGS) under the direction of Dr. David Dreisinger, Professor and Chair holder of the Industrial Research Chair in Hydrometallurgy at the University of British Columbia and Vice President, Metallurgy for SASC. Previous work reported under the Preliminary Economic Assessment of March, 2009 focused on two potential processing routes:

- Acid-chloride leaching
- Cyanide leaching

The current program continued to look into these two process options. The hydrometallurgical test program in 2008/2009/2010 consisted of:

- Agitated acid-chloride leaching of various samples;
- Bottle roll acid-chloride leaching of various samples;
- Column acid-chloride leaching of 09-1 and 09-2 samples;
- Bottle roll cyanide leaching of various samples;
- Column cyanide leaching of 09-1 and -09-2 samples;
- Diagnostic leaching (soak tests).

In addition to the leaching studies at SGS, there have been three other testwork programs initiated:

- Metal Recovery Work at SGS
- Hydrodynamic characterization studies at HydroGeoSense. Inc.
- Acid recovery testing at Eco-Tec.

The results of testing of the 2008, 2009 and 2010 metallurgical samples are reported by SGS in their documents titled:

- "The Recovery of Silver, Indium and Gallium from Malku Khota Samples – Report 1" dated April 14, 2008 (Project 11659-001) and;
- "The Recovery of Silver, Indium and Gallium from Malku Khota Samples – Report 2" dated July 22, 2009 (Project 11659-001) and;
- "An Investigation into Extraction of Silver and Indium from the Malku Khota Deposit- Final Report" dated December 7, 2010 (Project 12178-001).

This section of the report summarizes the results of the 2009/2010 testwork and is used in the current Preliminary Economic Assessment.

16.2 Metallurgical Sample Description

The following groups of samples were investigated in the current program:

- 08 series (08-1 to 08-6);
- 09 series (09-1 and 09-2);
- 10 series (10-1 and 10-2);
- 10 series (10-3 to 10-5);

- 10 series (10-6 to 10-17).

The 08 series was selected from intervals of core to represent the material in the deposit with high grades of Ag, In, Pb and Zn. Samples 09-1 and 09-2, as well as 10-1 and 10-2 were selected based on their physical behavior (hard versus soft lithology). Samples 10-3 to 10-5 were selected from the hanging wall of the deposit which is rich in gallium. The 10-6 to 10-17 samples were selected to represent the different ore zones within the deposit that can be identified in the resource model. The blocks within the block model can then be assigned a value based on the recovery of metals for that sample. Each of these samples was also tested at different crush sizes to study the effect of particle size on the leach kinetics.

These metallurgical samples were assembled at ALS Chemex in Oruru from minus 10 mesh material and at site from ¼, ½ and whole core under the direction of the senior geologist. The samples were secured in sealed containers and transported to ALS Chemex laboratory in Lima and SGS's laboratory at Lakefield. The samples were assayed by ALS Chemex and SGS. A comparison of the head assays reported independently from the two laboratories for the same metallurgical samples is presented in Table 16-1. This shows good correlation of the results. Minor differences can be attributed to the small size of the samples taken for assay at Lima. No assays were done in Lima for the 10-1 and 10-2 series samples as the whole core was sent to Lakefield.

16.3 Existing Testwork

The use of acid-chloride leaching (sulfuric acid, sodium chloride and sodium hypochlorite or hydrochloric acid, sodium chloride and sodium hypochlorite) has been studied in both standard bottle roll tests and column testing for precious metal, base metal and rare element extraction. The acid chloride solution is a potent leaching agent with sufficient capacity to extract indium, gallium, copper, zinc and lead as potential "pay metals" in addition to the extraction of gold and silver. The use of sodium cyanide to leach silver and gold has also been studied in standard bottle roll tests, column testing and stirred reactor tests.

Additionally, the acid-chloride leach process has been the subject of an intense program for developing a definitive test (a 'soak test') that can predict the extraction of key metals. The soak test involves taking a sample of MK material, grinding to target size and immersing in a "bucket" of solution containing acid-chloride-hypochlorite. The sample is soaked for an extended period with intermittent mild mixing and sampling. Metal extraction is calculated based on the solution analysis and final residues.

The metallurgical testing completed to date and reported here has provided the basis for evaluation of treatment options for the Malku Khota ore.

Table 16-1 Head Assay Correlations from Samples (Lima) and Leach Solution/Residue Assays (Lakefield)

Sample	Description	Assay	Au g/t	Ag g/t	Cu g/t	Ga g/t	In g/t	Pb g/t	Zn g/t	Bi g/t
08-1	LmD017 Approx 30 kg representing enriched Ag (8 samples)	Lima	< 0.01	140.57	842.90	1.47	5.6	2,779	306	33.40
		Lakefield	3.04	133	880	<2	5.2	2900	370	32
08-2	WwD020 Approx 32 kg representing enriched Ag and high In (9 samples)	Lima	0.051	270.16	878.6	8.01	59.3	1,627	93	192.77
		Lakefield	0.07	229	990	7	60	1700	160	140
08-3	LmD035 Approx 34 kg representing enriched Ag and high Pb (9 samples)	Lima	<0.01	794.31	804.6	1.43	7.2	6,774	600	271.38
		Lakefield	0.03	777	1100	<2	6.8	8000	690	270
08-4	LmD013 Approx 31.5 kg representing high In (9 samples)	Lima	<0.01	66.95	80.0	3.99	81.9	2,604	672	0.21
		Lakefield	<0.02	52	95	3	89	3000	740	3.3
08-5	WwD016 Approx 29.5 kg representing high Pb/Zn (8 samples)	Lima	0.017	10.04	50.3	4.80	22.4	7,613	12,394	0.14
		Lakefield	<0.02	12	65	4	32	10900	15000	0.7
08-6	LmD029 Approx 37 kg representing high Pb/Zn (9 samples)	Lima	<0.01	22.91	67.8	4.42	25.5	6,011	2,253	0.07
		Lakefield	<0.02	22.9	63	4	26	6700	2000	< 0.6
09-1	LmD006;7;8;10;17;19;41 Approx 147 kg representing hard material in the Limosna deposit (38 samples)	Lima	<0.01	100.3	250.6	2.6	15.7	3,682	318	12.07
		Lakefield	< 0.02	98.5	190	2	12	3600	290	7.9
09-2	LmD006;7;9;18;29;37;38;41;42 Approx 124 kg representing soft material in the Limosna deposit (29 samples)	Lima	<0.01	34.26	45.4	3.2	11.9	1,062	200	0.85
		Lakefield	0.03	31.9	40	2.9	11	820	180	< 0.6
10-1	New hole drilled at LmD017 - 018 representing hard material in the Limosna deposit (25 samples). Whole core to SGS	Lima	no assay	no assay	no assay	no assay	no assay	no assay	no assay	no assay
		Lakefield	0.02	108	570	2.1	3.2	1500	1.5	30
10-2	Twin hole drilled at LmD023 representing soft material in the Limosna deposit (13 samples). Whole core to SGS	Lima	no assay	no assay	no assay	no assay	no assay	no assay	no assay	no assay
		Lakefield	<0.02	30.5	97	3.5	2.9	1600	1.4	3.8
10-3	Lutita Edwin in Limosna: 8 samples located in the hanging wall with high Ga from 7 holes	Lima	0.006	12.37	40.8	25.5	8.1	6,341	4,838	0.50
		Lakefield	< 0.02	11.1	55	28	8.2	6300	4800	< 0.6
10-4	Intermedia Unit in Limosna: 14 samples located in the hanging wall with high Ga from 14 holes	Lima	0.007	5.39	29.8	29.6	18.1	1,045	2,836	0.64
		Lakefield	< 0.02	3.91	30	27	14	860	2100	< 0.6
10-5	Intermedia Unit in Warawara: 17 samples located in the hanging wall with high Ga from 9 holes	Lima	0.003	35.32	3,417.8	33.5	20.7	796	233	15.88
		Lakefield	< 0.02	29.7	3100	35	19	770	260	11

Table 16-1 (Cont) Head Assay Correlations from Samples (Lima) and Leach Solution/Residue Assays (Lakefield)

Sample	Description	Assay	Au g/t	Ag g/t	Cu g/t	Ga g/t	In g/t	Pb g/t	Zn g/t	Bi g/t
10-6	LmD007 11,25 kg Central Limosna, near Lutita Edwin (2 samples)	Lima	<0,01	97	25.35	2.13	3.25	764	249.5	0.05
		Lakefield	< 0.02	115	20.00	1.90	3	730	220	< 0.6
10-7	LmD008 11,5 kg South Limosna, near Lutita Edwin (4 samples)	Lima	0.01	43.3	24.35	2.96	10.2	611	385	0.07
		Lakefield	< 0.02	41.8	32.00	3.00	10	600	350	< 0.6
10-8	LmD007 11,75 kg Central Limosna, between two mantos (2 samples)	Lima	<0,01	30.9	49.35	2.07	15.48	412	86	0.06
		Lakefield	< 0.02	32	55.00	1.90	14	420	86	< 0.6
10-9	LmD009 11,65 kg South Limosna, between two mantos (3 samples)	Lima	<0,01	18.9	11.84	1.95	8.33	194	123	0.04
		Lakefield	< 0.02	21	15.00	1.70	8.3	240	130	< 0.6
10-10	LmD013 11,4 kg Central Limosna, manto in the contact between Malku Khota and Warawara sandstones (3 samples)	Lima	0.004	67.5	41.42	3.34	64.99	548	96	0.08
		Lakefield	< 0.02	71.2	48.00	2.70	69	650	76	< 0.6
10-11	LmD045 11 kg South Limosna, manto in the contact between Malku Khota and Warawara sandstones (3 samples)	Lima	<0,01	30.2	54.72	1.6	40.44	2160	621	0.01
		Lakefield	< 0.02	35.3	31.00	1.80	28	910	190	< 0.6
10-12	LmD007 11,4 kg Central Limosna, manto in the Warawara sandstone (4 samples)	Lima	<0,01	32	47.46	4.96	14.66	483	86	0.33
		Lakefield	< 0.02	38.5	50.00	3.70	15	930	300	< 0.6
10-13	LmD013 11,25 kg Central Limosna, manto Sedex in the Warawara sandstone (6 samples)	Lima	<0,01	67.4	98.52	2.98	3.95	2050	1029	0.08
		Lakefield	< 0.02	38	43.00	2.30	7.5	9600	4200	< 0.6
10-14	LmD045 11,7 kg South Limosna, manto Sedex in the Warawara sandstone (6 samples)	Lima	0.01	33.2	18.38	2.05	8.05	11601	6740	0.03
		Lakefield	< 0.02	37.7	25.00	1.90	10	15000	8800	< 0.6
10-15	WwD007 10,6 kg Internediate Unit at Warawara sector (4 samples)	Lima	<0,01	111.7	403.1	4.56	5.58	1017	24	17.04
		Lakefield	< 0.02	106	480.00	3.80	5.9	1200	30	19
10-16	WwD014 11,75 kg Sucre veins sector (1 sample)	Lima	<0,01	77.9	475	4.19	7.22	174	371	18.10
		Lakefield	< 0.02	41.8	410.00	4.00	9.8	150	280	28
10-17	WwD024 11,25 kg Sucre vein sector (3 samples)	Lima	0.018	229.6	2704.25	3.49	14.1	543	3116	74.35
		Lakefield	< 0.02	220	2600.00	3.40	14	610	2890	55

16.3.1 Acid – Chloride Leaching

To establish the general level of dissolution of metals, samples were tested in 1 m column tests at a crush size of P80 3/8". The initial samples for this testing were selected with respect to the apparent level of silicification, one hard sample and one soft sample. These samples were considered "end-member" samples regarding the characteristics of the ore in general, and therefore, do not necessarily represent the majority of the resource. These samples were labeled 09-1 and 09-2.

Table 16-2 Column tests on Sample 09-1 (3/8" material)

		Amt.	Assays (mg/L, g/t)			Extraction (%)		
Product		mL, kg	Ag	In	Fe	Ag	In	Fe
Head	Day	23	98.5	12	6600			
Day 1 PLS	1	27.2	44.8	10	1990	17.0	27.8	10.9
Day 2 PLS	2	25.4	74	16	3970	28.8	45.7	22.4
Day 3 PLS	3	26.3	86.5	18	5400	34.0	52.0	30.8
Day 4 PLS	4	40.1	96	21	6730	37.4	60.1	38.0
Day 8 PLS	8	24.5	107	24	8050	41.3	67.9	45.0
Day 9 PLS	9	22.0	103	24	8440	39.7	67.8	47.1
Day 11 PLS	11	24.7	106	25	8940	40.3	69.6	49.2
Day 14 PLS	14	32.3	107	26	9010	40.2	71.6	49.0
Day 17 PLS	17	25.0	116	27	9640	43.9	74.8	52.8
Day 21 PLS	21	28.1	111	28	9830	42.3	78.1	54.2
Day 24 PLS	24	25.5	127	28	10200	47.7	77.2	55.5
Day 28 PLS	28	24.8	124	29	10600	47.6	81.7	58.9
Day 31 PLS	31	49.3	130	30	11000	44.5	75.3	54.4
Day 35 PLS	35	25.1	129	30	11100	47.3	80.7	58.9
Day 39 PLS	39	58.6	137	30	11800	50.9	81.9	63.4
Day 43 PLS	43	30.6	132	30	11200	50.0	83.3	61.4
Day 51 PLS	51	39.0	89.8	25	9770	42.6	86.2	66.3
Day 54 PLS	54	39.7	100	25	9940	47.8	87.3	68.3
Day 62 PLS	62	33.5	105	26	10200	49.7	90.0	69.5
Day 69 PLS	69	31.5	120	27	11800	54.1	89.4	76.6
Day 78 PLS	78	35.5	128	29	11800	58.8	97.9	78.4
Day 85 PLS	85	47.0	125	28	12000	58.1	95.7	80.5
Final PLS	95	10382	121	26	12000	55.8	88.3	79.6
Final wash		6569	14.7	5.1				
Residue (calculated)		22.7	45.4	0.91	1840			
Column Crystals		0.035	1434	6.9				
Calc'd Head						109.3	14.9	7532
Extraction (Calc head - residue)						**58.9**	**94.0**	**75.9**
Mass Balance (out/in%)						111.0	124.2	114.1
Calc'd wt. Loss		1.2%						

Table 16-3 Column tests on Sample 09-2 (3/8" material)

		Amt.	Assays (mg/L, g/t)			Extraction (%)		
Product		mL, kg	Ag	In	Fe	Ag	In	Fe
Head		22	31.9	11	6800			
Day 1 PLS	1	26.5	8.43	4.7	910	12.3	19.0	5.7
Day 2 PLS	2	25.8	14.9	8.3	2120	21.4	33.1	13.1
Day 3 PLS	3	25.6	17.3	9.9	3380	24.6	39.0	20.6
Day 4 PLS	4	37.1	17.5	11	4390	24.6	42.9	26.5
Day 8 PLS	8	21.7	21.7	14	6060	30.5	54.6	36.5
Day 9 PLS	9	25.7	23.6	15	6350	34.1	60.2	39.4
Day 11 PLS	11	24.0	22.1	16	6380	30.7	61.6	38.0
Day 14 PLS	14	36.1	22.2	17	7000	31.6	66.9	42.6
Day 17 PLS	17	29.5	22	17	7340	31.0	66.4	44.3
Day 21 PLS	21	35.9	22.4	18	7330	32.1	71.4	45.0
Day 24 PLS	24	29.1	25	19	8010	35.8	75.3	49.1
Day 28 PLS	28	30.8	27	20	8710	38.7	79.5	53.5
Day 31 PLS	31	49.6	27	20	8840	37.5	77.0	52.5
Day 35 PLS	35	34.0	26.4	21	9090	36.9	81.2	54.3
Day 39 PLS	39	44.6	26.1	21.5	9050	36.7	83.5	54.4
Day 43 PLS	43	35.1	25.2	22	8910	35.2	84.8	53.2
Day 51 PLS	51	42.4	17.1	19	7860	29.9	90.6	58.0
Day 54 PLS	54	40.4	17.1	20	7860	30.4	96.7	58.9
Day 62 PLS	62	41.4	18.1	21	8080	31.5	99.5	59.4
Day 69 PLS	69	38.5	21	22	9090	35.5	101.9	65.1
Day 78 PLS	78	46.1	16.8	23	9200	28.8	106.3	65.9
Day 85 PLS	85	50.0	12.2	23	9380	21.1	104.2	65.8
Final PLS	95	8902	2.88	23	11000	6.8	103.6	76.1
Final Wash		10200	0.83	0.5				
Residue (calculated)		21.7	19.80	0.82	2418			
Column Crystals		0.09	472					
Pail Crystals		0.16	919	1.20	700			
Calc'd Head						30.5	11.0	7102
Extraction (Calc head - residue)						**36.0**	**92.6**	**66.4**
Mass Balance (out/in%)						95.7	99.9	104.4
Calc'd wt. Loss		1.3%						

A further series of samples, samples 10-6 through 10-17, was collected. These samples are deemed by SASC to be more representative of the resource. The 10 Series samples and the 09-1 and 09-2 samples were "soak" tested by the method described above. The soak test results for samples 09-1 and 09-2 are shown below.

Table 16-4 Soak Tests on Samples 09-1 and 09-2 (Coarse sizes)

Test ID	Sample	Crush Size 100%-	Duration day	Chge kg	Lixiviant L	NaOCl g/L	PLS In mg/L	PLS Ag mg/L	Residue In g/t	Residue Ag g/t	Dist. to Liquor In %	Dist. to Liquor Ag %	HCl kg/t
Head	09-1								12.0	98.5			
Soak 1	09-1	1"	20	1000	10	1	0.9	2.96	1.6	51.9	85.3	37.1	16
Soak 2	09-1	3/8"	20	1000	10	1	1.2	3.47	2.0	38.9	86.1	48.0	14
Soak 3	09-1	16m	20	1000	10	1	1.1	4.11	1.3	23.9	90.1	65.0	12
Soak 4	09-1	32m	20	1000	10	1	2.5	11.50	1.3	23.3	95.3	83.8	28
Soak 5	09-1	1"	49	1000	10	1	1.7	5.90	1.7	56.2	91.1	51.9	17
Soak 6	09-1	3/8"	49	1000	10	1	1.5	4.56	1.1	27.3	93.5	63.7	17
Soak 7	09-1	16m	49	1000	10	1	1.3	6.03	1.0	20.3	93.1	75.5	15
Soak 8	09-1	32m	49	1000	10	1	1.4	6.09	1.0	19.9	93.6	76.0	16
Head	09-2								13.5	27.1			
Soak 9	09-2	1"	20	1000	10	1	1.1	0.84	2.5	21.9	81.9	28.3	12
Soak 10	09-2	3/8"	20	1000	10	1	0.9	2.14	2.1	23.2	81.5	48.7	11
Soak 11	09-2	16m	20	1000	10	1	0.8	1.64	1.8	21.0	82.2	44.7	11
Soak 12	09-2	32m	20	1000	10	1	0.8	1.36	2.0	20.0	81.0	42.1	11
Soak 13	09-2	1"	49	1000	10	1	1.1	1.36	1.1	19.9	91.2	25.8	22
Soak 14	09-2	3/8"	49	1000	10	1	1.0	0.67	1.3	21.2	88.9	26.6	19
Soak 15	09-2	16m	49	1000	10	1	1.0	0.74	1.0	18.7	91.0	30.1	14
Soak 16	09-2	32m	49	1000	10	1	1.0	0.78	1.3	17.4	88.5	33.2	14

Further testing at very fine size shows continued improvement in silver extraction below 32 mesh.

Table 16-5 Soak Tests on Samples 09-1 and 09-2 (Fine sizes)

Test ID	Sample	Crush Size 100%-	Duration day	Chge kg	Lixiviant L	NaOCl g/L	PLS In mg/L	PLS Ag mg/L	Residue In g/t	Residue Ag g/t	Dist. to Liquor In %	Dist. to Liquor Ag %	HCl kg/t
Soak 09-1a	09-1	42m	21	200	2	1	1.2	7.90	1.3	18.5	90.6	81.6	16.8
Soak 09-1b	09-1	60m	21	200	2	1	1.2	8.90	1.3	14.4	90.7	86.6	19.3
Soak 09-1c	09-1	115m	21	200	2	1	1.2	9.20	0.9	9.5	93.3	91.0	20.5
Soak 09-2a	09-2	42m	21	200	2	1	0.7	1.36	1.7	16.8	81.2	45.9	9.9
Soak 09-2b	09-2	60m	21	200	2	1	0.9	1.82	1.5	13.4	86.2	58.6	13.1
Soak 09-2c	09-2	115m	21	200	2	1	0.9	2.18	1.5	9.5	86.1	70.4	14.6

The results indicate that In extraction is generally 80-95% and relatively insensitive to crush size from 1 inch down to 115 mesh. The silver extraction on the other hand has a strong correlation to size ranging from 37-91% for sample 09-1 and 25-70% for sample 09-2.

Table 16.6 compares the acid-chloride leaching behavior of the samples 09-1 and 09-2 for the different tests.

Table 16-6 Comparison of Tests on 3/8" material

Test	Sample	Extraction Ag	Extraction In
Column Test @ 95 days*	09-1	58.9	94.0
Bottle Roll Test @ 56 days	09-1	67	86
Soak Test @ 49 days	09-1	63.7	93.5
Column Test @ 95 days*	09-2	36	92.6
Bottle Roll Test @ 56 days	09-2	35	83
Soak Test @ 49 days	09-2	26.6	88.9

The SGS Report evaluates that silver extraction in the column test is similar to the extraction achieved using the Soak Test for similar size material and this is taken to support using the soak tests as a means of predicting metal dissolution in a heap. The following observations and conclusions can be made:

- The 3/8 inch crush size data of sample 09-1 led to silver and indium extraction after 49 days of 63.7 and 93.5% respectively. This compares well to the column results (K80 of 3/8 inch – 95 days) of 58.9% Ag and 94.0% In.
- Likewise the 3/8 inch crush size data of sample 09-2 led to silver and indium extraction after 49 days of 26.6% and 88.9% respectively. This compares reasonably well to the column results (K80 of 3/8 inch – 95 days) of 36.0% Ag and 92.6% In.
- Silver extraction from 09-1 has a strong dependence on crush size. It increases from 37% (1 inch) to 91% (115 mesh);
- Silver extraction from 09-2 shows an erratic pattern with a probable outlier at the 3/8 inch data point. Even at the finest crush size, silver extraction does not increase much beyond 70%;
- Indium extraction from samples 09-1 and 09-2 shows very little dependence on crush size;
- Generally it appears that the extended soak test duration of 49 days has a pronounced effect on recovery at the coarse crush sizes (3/8 inch and 1 inch in particular), but appears to have little effect within the fine crush size tests.
- Lead extraction from 09-1 shows a sharp increase between 3/8 inch and 1 inch, but little increase in extraction at finer crush sizes. Lead extraction from 09-2 show no dependence on crush sizes.

The other metals that were extracted in the acid-chloride column test are shown in the table below. High levels of base metal and some extraction of gallium have been achieved in the acid-chloride leach.

Table 16-7 Extraction of Cu, Pb, Zn, Ga

Element	Extraction From Column 09-1	Extraction From Column 09-2
Cu	91%	82%
Pb	88%	88%
Zn	75%	76%
Ga	46%	25%

The soak test procedure is continuing to be refined and has now been applied to samples 10-6 to 10-17 under conditions of variable grind size and leach time. The recovery of the various metals has been recorded for the different samples up to 64 days and is continuing. The results are presented by SGS in their correspondence with Dr. Dreisinger dated March 4, 2011. The samples 10-6 to 10-17 were selected to represent the different ore zones in the deposit and the recoveries were weighted according to the tonnages that each sample is estimated to represent in the deposit. The average recovery for Ag across the deposit by this method is 73.6% at 64 days and for Indium is 81%. The weighted recovery curves for different particle size for Ag and In are shown in Figure 16-1 and Figure 16-2 below. The recovery curves for silver shows that recovery continues to increase with time.

Figure 16-1 Weighted Average Leach Recoveries for Silver (Samples 10-6 through 10-17)

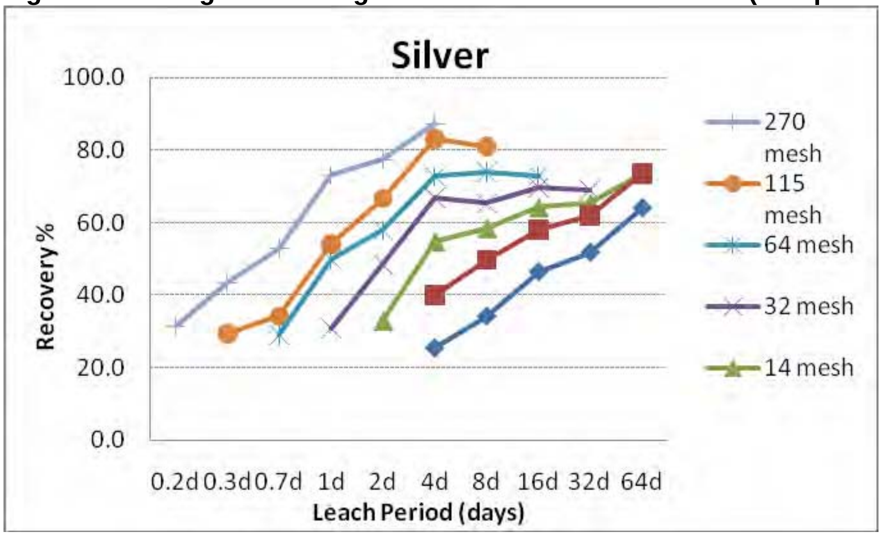
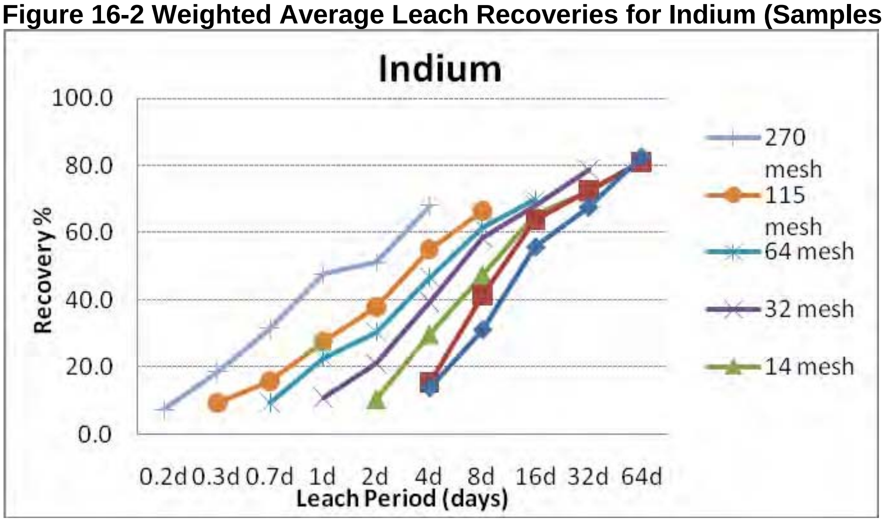


Figure 16-2 Weighted Average Leach Recoveries for Indium (Samples 10-6 through 10-17)



The weighted recovery of all pay metals for ¼" material (heap leach material) is shown in Table 16-8. The data from these acid-chloride soak tests were used as a predictor of metal extraction in a acid chloride heap leach for this PEA Update.

Table 16-8 Metal Recoveries from Acid-chloride leach of 1/4" material

Metal	Average Extraction
Ag	73.6%
In	81.0%
Pb	51.1%
Zn	62.0%
Cu	84.8%
Ga	26.9%

16.3.2 Cyanide Leaching

The cyanide leach testing can be summarized in the figures below (abstracted from the SGS Report – December 7, 2010) for column leaching of samples 09-1 and 09-2. These results demonstrated fast recovery of gold and reasonably fast recovery of silver. The results also confirmed that bottle roll tests would be good predictors of column performance for silver leaching.

Figure 16-3 Metal Extraction vs. Time from cyanidation of Sample 09-1 at P80 of 9mm (~3/8")

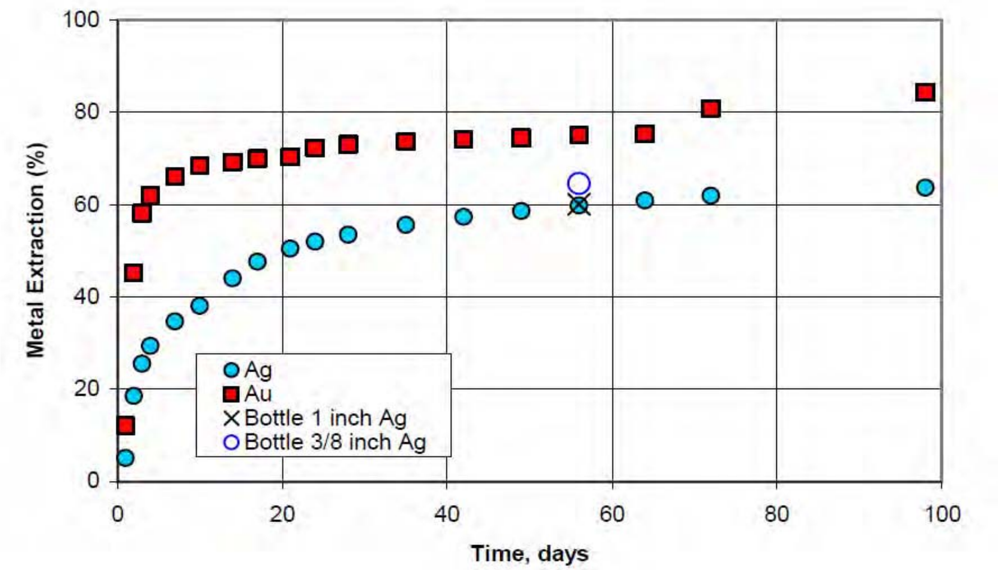
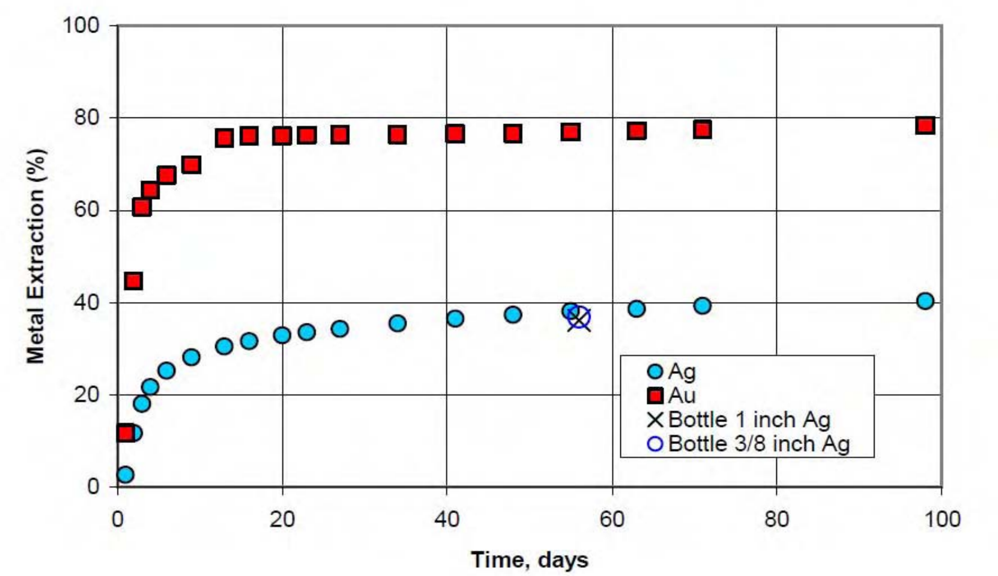


Figure 16-4 Metal Extraction vs. Time for cyanidation of Sample 09-2 at P80 of 9mm (~3/8")



A further series of cyanide leach tests was carried out for samples 10-1 to 10-17.

Table 16-9 Cyanide Leach Tests on the "10-series" Samples

Sample	Ag Extraction (%)	
	Coarse Bottle Roll Tests (-10 mesh, 28 days)	Fine Stirred Reactor Tests (-74 um, 72 h)
10-1	59.4	84.8
10-2	45.4	72.9
10-3	55.3	56.8
10-4	48.5	47.6
10-5	80.3	87.9
10-6	31.6	81.6
10-7	52.3	80.6
10-8	43.4	80.7
10-9	37.3	76.5
10-10	51.1	84.3
10-11	48.4	81.7
10-12	61.1	81.8
10-15	72.0	83.4
10-16	41.8	88.5
10-17	23.5	55.0
Average	50.1	73.9

The results confirm that cyanidation can be used to extract silver and gold from Malku Khota samples and that the extraction of silver is strongly influenced by the particle size for leaching.

16.3.3 Hydrodynamic Characterization Studies at HydroGeosense, Inc.

HydroGeoSense was contracted to perform preliminary geotechnical tests on samples of residue from the 1 m column, acid chloride leaching tests of samples 09-1 and 09-2 at the crush size of P80 3/8”. As discussed in Section 16.3.1, the 09-1 and 09-2 samples were selected with respect to the apparent level of silicification, one hard sample and one soft sample. These samples were considered “end-member” samples regarding the characteristics of the ore in general and testing these samples was considered to provide preliminary upper and lower values for the heap design criteria.

The essential outcome of the testing was that the two samples tested showed characteristics suitable for multi-level heap leaching. The report from HydroGeoSense states;

“The results from the stacking tests lead us to preliminarily conclude that the MK composites 09-1 and 09-2 would support a multi-lift leaching process with solution application rates of 12 l/h/m2 and less.”

This is an important result for the development program as heap leaching is the process scenario that is currently being considered for the project.

16.3.4 Metal Recovery Work at SGS

The recovery of gold and silver from cyanide solutions is straightforward and can be accomplished by carbon adsorption or zinc dust precipitation. As these are commonly applied to cyanide solutions, no development work has been performed on metal recovery from cyanide solution.

The recovery of the valuable metals from the acid-chloride leach solution is more complex due to the concentrated acid-salt solution matrix and the multiplicity of metals to be recovered. The work on metal recovery from solution has been very successful and leads naturally to a range of intermediate products for sale or further refining.

Silver and indium along with minor gold, copper, lead zinc and gallium values present in the pregnant leach solution can be recovered in the plant by the steps of (1) acid neutralization, (2) silver/gold/copper cementation, (3) indium/gallium removal, (4) hydrogen sulfide precipitation of PbS and (5) hydrogen sulfide precipitation of ZnS. All of these steps have been tested successfully at SGS.

The cementation of precious metals is accomplished by contact of the acid depleted leachate with scrap iron. The silver/copper/gold "cement" recovered at SGS analyzed 14.8 g/t Au, 18,645 g/t Ag, 7.19% Cu along with 44.2% Fe, 2.15% Pb, 2.17% As, 2.19% Sb.

The precipitation of indium and gallium is accomplished by raising the pH with sodium hydroxide solution to precipitate the metal hydroxides. The precipitate analyzed 0.58% In, 120 ppm Ga, 18% Al, 13% Fe, 3.99% Pb. The precipitate also contains As, Cu, Sb and Zn as impurities. This crude precipitate is typical of the first recovery of indium from a leachate and it is necessary to refine further to pure indium products.

The precipitation of lead and zinc can be performed sequentially using a source of sulfide ion (e.g. hydrogen sulfide or a sulfide chemical). The SGS results show that it is possible to obtain a high grade lead concentrate (analysis 75.4% Pb, 0.98% Fe and 120 ppm Zn) followed by a mixed lead-zinc concentrate (analysis 14% Zn, 61.6% Pb, 2.4% Fe). There is opportunity for more selective precipitation of lead and zinc but the initial testing was very promising.

16.3.5 Acid Recovery Testing at Eco-Tec

The Eco-Tec company has a technology for acid recovery from salt solutions. This technology is installed around the world for regenerating acid solutions used in the metal finishing industry as well as other applications. In the case of the acid-chloride leach technology under development, it is desirable to recover and re-use acid, prior to treating the solution for metal recovery. Note in the process described above, the first step in chemical treatment is acid neutralization. The cost of acid at MK necessitates minimizing the consumption of acid and hence the need to separate and recover acid prior to recovery of the pay metals.

Eco-Tec has completed a study of the recovery of acid from a synthetic solution provided by SGS. The solution was designed to mimic the column/heap leach solution composition. The Eco-Tec testing was very successful and indicated that a high degree of acid recovery could be achieved.

The Eco-Tec report dated January 2011 concludes;

- Acid recovery efficiency was measured as 90%. This indicates that 90% of free HCl in the feed stream processed by the APU (Acid Purification Unit) reports to the recovered acid stream. For design purposes a safety factor of a few percent would normally be applied to this value.
- Sodium separation efficiency was measured as 76%. This indicates that 76% of sodium in the feed stream processed by the APU reports to the de-acidified salt stream. For design purposes a safety factor of a few percent would normally be applied to this value.

These results confirm that the Eco-Tec acid recovery process may be applied to MK acid-chloride leach solutions. The individual pay metal deportment between the recovered acid solution and the salt permeate must still be studied further to minimize any recycle of pay metals back to the heap. This is the subject of ongoing discussions with Eco-Tec and will require further bench testing.

16.3.6 Indium Refining

The indium intermediate product from the acid-chloride process must be refined further to +99.99% purity. The nature of the product to be produced at MK (mixed indium, gallium, aluminum, iron hydroxide) is commonly recovered in the first step in the recovery flowsheet.

There are a number of indium refineries around the world that could refine the MK indium precipitate. For example, Teck has a refinery in Trail, B.C., Votorantim Metals has a refinery in Lima, Peru and Dowa Mining has a refinery in Japan. However, the cost of customized refining and the lack of sufficient capacity for the anticipated In production rate from MK require that an alternative to toll refining be developed.

The refining of indium is a specialized technology and will require testing of the specific refining process for MK indium product. The process will include an acid re-leach and precipitation (multiple steps to reduce the Fe and Al content of the precipitate) followed by solvent extraction of In and cementation of indium on aluminum plates.

16.3.7 By-Product Metal Recovery

The recovery values available in the metallurgical test work for the minor by-product elements of lead, zinc, gallium and copper are presented in Table 16-8. The parameters used to calculate the by-product metal credit value for gold are shown in Table 16-10.

Table 16-10 Calculation of Gold Credits

Grade	0.032	g/ tonne
Price	$1,050	/ ounce
Metallurgical Recovery	70%	
Overall Payability	95%	
Gold Credit	$0.72	/ tonne

16.4 Future Testwork

A number of studies that will be used in the PFS are either underway or planned for 2011.

Silver solubility tests

Silver solubility tests are underway in order to determine the solubility of silver as a function of key solution chemistry parameters (HCl, NaCl and other metal salt components) and temperature. The objective is to make sure that the leach solutions used to treat MK mineralization have sufficient capacity to extract silver without being concerned about the re-precipitation of silver.

Soak tests to establish the temperature-grain size relationship to recovery

The soak test method is a relatively simple and lower cost technique to determine the metal extraction versus particle size and temperature. The ore is placed in sealed buckets with the required solution composition and maintained at the target temperature by being placed in controlled temperature room or a water bath. Samples of solution (and if necessary, solids) are taken out of the soak test over time to determine metal extractions. The soak tests are currently being used to characterize samples from across the deposit. Additional soak tests are being used to determine the relationship between free acid in solution and metal extraction. The ultimate goal of the soak test work is to have a test procedure available to predict the ore response to the leaching process.

Metal recovery processing testwork.

The basic elements of the metal recovery flowsheet have been demonstrated. There is additional optimization and variability work to be conducted. Additionally, the treatment of intermediate products will be studied. Specifically, work will be done toward the following points;

- Optimization of the acid recovery and recycle process using Eco-Tec Recoflo system or other alternate processes with a view to maximizing acid recovery with minimum acid use and ensuring that all pay metals deport to the low acid solution.
- The use of cementation for silver, gold and copper recovery will be optimized. Alternate methods of reduction of ferric ion will be studied to determine if iron consumption in cementation can be reduced.
- The treatment of the silver, gold, copper cement to separate into a silver-gold dore for refining, a copper product for sale and by-products/waste products will be investigated
- The use of limestone to precipitate indium and gallium will be studied.
- The refining of the indium/gallium hydroxide precipitate will be investigated.
- The production of separate lead and zinc sulfide concentrates by sulfide precipitation will be examined carefully in order to maximize the value of the respective products.
- The removal of residual iron from solution will be studied in order to regenerate the leach solution after precious and base metal recovery.

Larger scale hydrodynamic and geotechnical testing

The preliminary studies by HydroGeoSense have confirmed that the MK mineralization is amenable to heap leaching. Further work will be performed on hydrodynamic (fluid flow) and geotechnical aspects (stability of material under and after leach process).

Hydrodynamic testing

Refer to Section 16.5.1 under the heading “Future Column Testing and Piloting”.

16.5 Processing Plant Design

The metallurgical tests are the basis for developing a conceptual design of a heap leach facility and recovery plant for the Malku Khota Project. As envisioned for this study, heap leaching would provide the lowest cost processing option for this low grade ore. Two heap leach process chemistries are possible:

- The developmental hydrochloric acid - chloride leach chemistry provides the greatest level of metal recovery for silver, indium and the minor component metals of gold, copper, zinc, gallium and lead.
- Cyanide heap leaching, a very well-known and practiced process chemistry, can extract silver and gold from the Malku Khota ore with a reasonable efficiency.

The operation scenario developed for this scoping study is the processing of 40,000 tons of ore per day by heap leaching in a hydrochloric acid - chloride leach solution. The testwork on the hydrochloric acid - chloride leach option has been greatly advanced since the PAH Report in March 2009, and this is presented as the most prospective scenario for project development.

16.5.1 *Scale-up Procedure from Test Work for Conceptual Heap Leach Design*

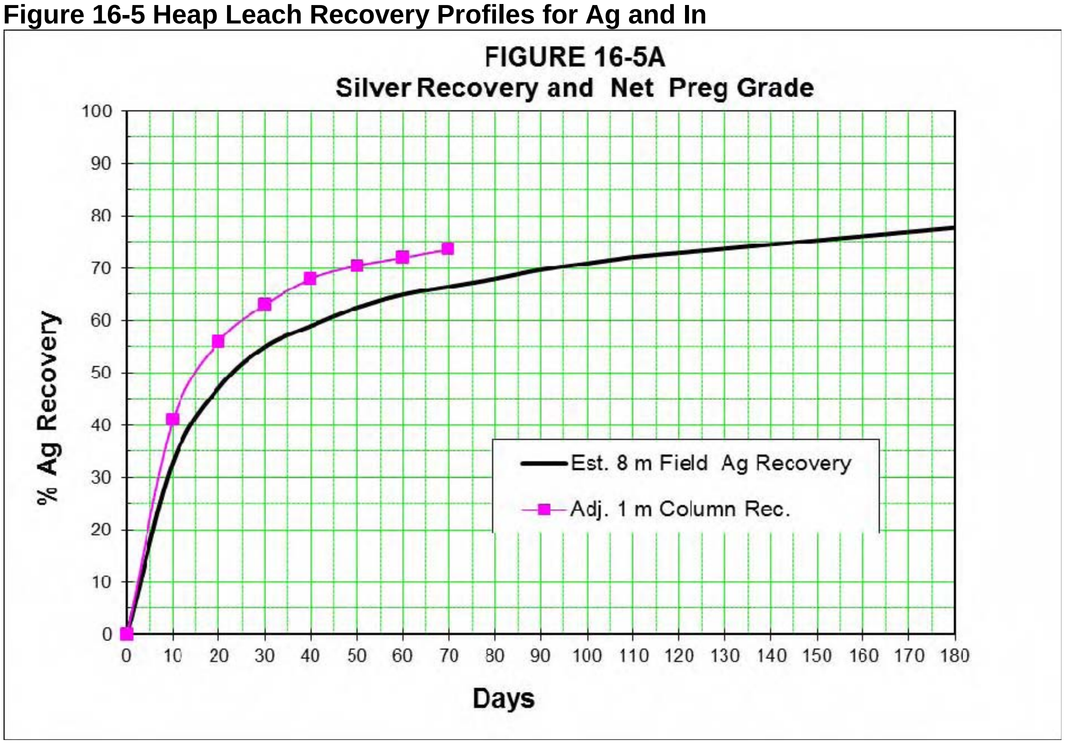
The following is taken from the memorandum by Randolph Scheffel, titled Malku Khota Project – Preliminary Acid Chloride Heap Leach Scale-up for PEA , dated: February 22, 2011:

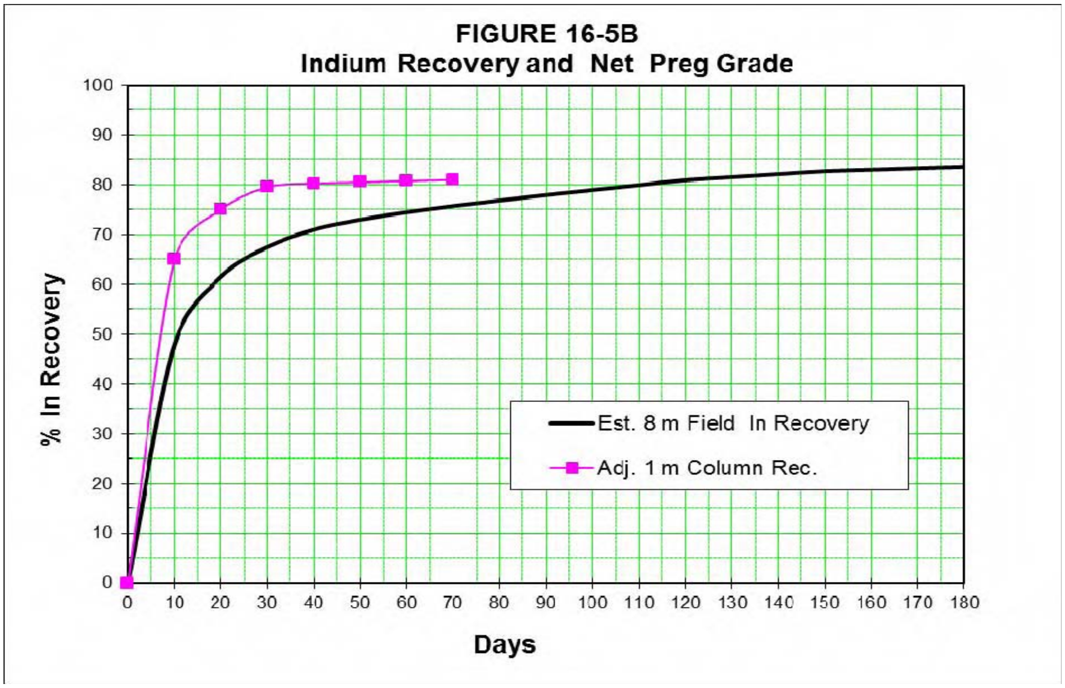
The scaled-up extraction profile is key to determining the solution management scheme to apply in the "conceptual" heap leach design. It will take full depth column testing to confirm the assumptions made herein:

- The leaching rate, not necessarily overall dissolution, experienced in the 09-1 1 m column test is used to develop the initial extraction profile.
- The targeted total dissolution, based on the weighted average ¼" Series 10 "soak" tests, is assumed in a 1 m column test to be achieved at day-70.
- The leach cycle must be increased to deal with the lower total solution application taking into regard the reagent delivery. In this case, because the acid, sodium and chloride concentration is so high relative to the silver and indium concentration in the ore, and assuming the gangue acid requirement can be quickly satisfied, it is assumed that it may be possible to achieve the level of dissolution in about half the time that might typically be used for a gold cyanide or copper acid heap leach. Thus, where typically scale-up from a 1 m column to 8 m would be by a factor of 4, the scale up for the acid chloride leach is taken as a factor of 2. It will take tall column testing to confirm this key assumption.
- It is also assumed the level of dissolution will be 6% below the 1 m dissolution at day-70 and this 6% is recovered in the doubling of the leach time.

Using the method described above, it is then possible to scale-up the 1 m column to an 8 m "commercial" heap. The result of this scaled-up leach profile is shown in Figure 16-5 below.

Figure 16-5 Heap Leach Recovery Profiles for Ag and In





With these two scaled-up 8 m "commercial" heap leach profiles, it is possible to put them into a simple mass balance that assumes these recovery profiles can be maintained across a small range of variation to application rate, ore grade, dry bulk density and ore depth.

Commercial Scaled-up Mass Balance

A mass balance can be developed off this fixed recovery profile accommodating a range of variations to application rate, rest/rinse leaching, etc. to arrive at a reasonable solution management scheme.

It was decided to limit the PLS flow for metal and acid recovery to 833 m^3/h if possible to best match the acid recovery units. Therefore the following mass balance suggests a combination of continuous application to supply the initial acid requirement followed by a rest/rinse leaching cycle.

Heap Leach Design Criteria

Lift Height 8.0m
Application Rate =9 L/h/m^2 for 20-30 days; 3.00 L/h/m^2 days 31-120
Density =1.65 t/m^3
Tons/m^2/lift =13.2,
Daily Tons =40,000 tonnes
Surface area required: Primary – 363,636 m^2

Ag mass balance:
Head Ag =31 ppm
Operating recovery =73% of Ag (possible additional dissolution with time)
PLS Grade 46.07 ppm Ag at 30 days
PLS flow to metal recovery 812 m^3/hr
Daily Ag production=904 kg Ag

The associated Indium mass balance is:
Head grade In =5.9 ppm
Operating recovery =81% of In (possible additional dissolution with time)
PLS Grade 9.7 ppm In at 30 days
PLS flow to metal recovery 812 m^3/hr
Daily production =191 kg In

The above mass balance using an initial application rate of 9 L/hm^2 for 30 days then converting to a 60-day rest and 30-day rinse irrigation scheme, also applying 9 L/hm^2 when rinsing, so no solution application piping changes are necessary, shows a PLS stream of ~818 m^3/h containing ~46 ppm Ag and 9.7ppm In. The leach cycle on a single-lift basis would be 120 days, and it would take ~364,000 m^2 of area to be piped and under various stages of leaching or resting. Additional metal dissolution is likely if the ore is leached in multiple lifts but this has other inventory issues as mentioned below.

Conceptual Leach Design

Once the commercial single-lift mass balance is developed, the decision whether to leach a single-lift or in a multiple-lift must be chosen. There are trade-offs to either approach. The negative factor with leaching this ore in a multiple-lift is the high PLS grade, developed to reduce the size of the metal recovery and acid recovery system, has a price to pay in "operating" recovery as the PLS grade for both metals must be retained by the moisture in the heap to support the high PLS grade.

Multiple-lift Leach vs. On/Off "dynamic" Pad

Some added dissolution from the ore in the underlying ore lifts of a multiple-lift leach design can mitigate some of this loss, but until the current testing is completed, it is difficult to estimate what this might be. An on/off pad of 120-day leach duration may be a better economic trade-off under these circumstances. At 120 days, the ore must be taken off the pad to provide room for fresh ore. The PEA Update is based on the On/Off Pad option with no "lock-up" of metal values in the heap.

Future Column Testing and Piloting

While the above assumptions and general treatment of the scale-up criteria are reasonable considering the aggressive nature of the acid-chloride leach, the behavior at the increased ore depth for this chemistry will require full depth column testing to support a definitive feasibility study.

A factorial matrix column test program will be necessary to determine the levels of application rate, acid concentration and solution management schemes. In fact, a two- or three-phased test program could be necessary. The first phase may be shorter columns (two to four meters) testing the optimal solution chemistry and then taking this to a full-depth program in a Phase 2 program which would be designed to confirm the basic assumptions in this initial scale-up. Ultimately, a pilot plant program may be necessary to prove the fully integrated leach, metal recovery, acid recovery and recycle of solutions over a longer period of time.

16.5.2 Crushing, Screening and Stacking

At the process plant location, feed will be reduced in size from run of mine to minus 9 mm in three stages of crushing. Feed will be direct dumped into a primary gyratory crusher. The gyratory product will proceed through a secondary standard cone crusher followed by two tertiary short head cone crushers. Both stages of cone crushers will be preceded by the appropriate primary and secondary screens. The material handling from crusher to pad and pad to residue storage will use a hopper and feeder at the crusher to load 100 ton trucks that will take crushed material to the heap and place the material on the heap. Unloading the leached material from the heap after 120 days and transporting the material to the residue facility will be accomplished using a loader (Cat 994) and 100 ton trucks.

A schematic of the crushing, screening and stacking process flowsheet is shown in Figure 16-6.

16.5.3 Leach Pads

The ore will be leached with a hydrochloric acid, sodium chloride and hypochlorite leach solution. The leach solution percolating from the pad will be recirculated to the top of the leach pad multiple times to build up the silver and indium concentration. Approximately 818 m^3 of the solution volume will be diverted continuously to a metal recovery plant. Following metal recovery, barren solution will be returned back to the heap leach pad.

The heap leach will be designed as an On/Off leach pad to receive 40,000 tpd in a single 8m lift sized for a 120 day leach cycle with sufficient area designated for maintaining the required area under leach (364,000 m^2) plus the loading and unloading area requirements. The conceptual design for this operating requirement is:

The On/Off pad will have 6 cells:

- Cell 1 – Leach 30 days
- Cell 2 – Leach 10 days; Rest 20 days
- Cell 3 – Leach 10 days; Rest 20 days
- Cell 4 – Leach 10 days; Rest 20 days; Last rinse with Barren
- Cell 5 - Off loading
- Cell 6 - Loading

Each cell area is estimated as approximately 90,000 m^2 = (40,000 tpd * 30 days)/(1.65 t/m^3 * 8 m lift height) The total area required for the pad is 540,000 m^2

Construction

The pad will be double lined with a compacted clay sub-base overlain by a 2 mm thick high density polyethylene (HDPE) synthetic membrane. The membrane will be protected by an overlying blanket of crushed feed. A series of ditches and perforated pipes buried in the drain blankets speeds the collection of the pregnant solutions. The leach pad will be constructed on flat ground with down-hill slope of 4 percent or more to minimize earth moving costs of construction. Pregnant, intermediate and barren solutions are collected in solution ponds constructed down slope of the leach pad. Ponds are constructed with a double liner of 1.5 mm thick HDPE synthetic membrane separated by geo-net for a leak detection system.

Operation

The leach pad is loaded in a series of cells each 180 by 500 meters. Stacking begins at the down slope end of a cell progressing upslope. This practice speeds leach solution application to fresh feed and obtains rapid solution return. Leach solution will be applied to the top of the stacked feed by pumping through a series of distribution headers and drip emitter lines. Leach solutions draining from the leach pad collect in a pregnant liquor pond down slope of the pad. The bulk of the leach solution will be continuously re-circulated to the leach pad to build up the silver and indium solution concentrations. The rate of leach solution application to the heap leach is 9 liter/hr/sq meter. At full production the total rate of leach solution application to the heap should roughly average 818 m^3 for a 40,000 tpd operation. A split stream of 818 m^3 of the acid leach solution will be diverted from the PLS Pond to the metal recovery plant.

Following the 120 day leach cycle, the cell will be washed with barren solution to recover the values in the solutions that remain in the heap. The "washed cell will be allowed to drain down to the residual moisture level following which the roughly 1.2 million tons of leached ore will be removed from the cell by loader and trucks to allow fresh ore to be deposited. The leached ore will be deposited in the Residue Facility (Figure 16.8). The Residue Facility will be lined and any solution that migrates downwards through the mass of material will be collected and returned to the intermediate solution pond with the supernatant solution that is recovered.

Additional metal recovery is expected from the residue facility that continues to leach over a longer period. This has not been included in the recoveries used in the financial model

Water Balance

Assuming the run of mine feed will have a moisture content of 3 percent and the stacked feed will have a moisture content of 15 percent under leach, approximately 4,800 tpd or 880 gpm of water on a 24 hr/day basis will be required to initially wet the stacked feed for a 40,000 tpd operation. It is anticipated that this water demand will be sustained throughout the first 120 days of operation as new material is continually being placed under leach. Beyond the first 120 days of operation, cells taken off the leach cycle will drain down to lower moisture content. Assuming cells not under leach will drain down to a 9 percent moisture level, before unloading to the residue facility, the make-up water demand for heap leach construction may be reduced from 880 gpm to as low as 440 gpm for the duration of the operation. These estimates do not include the impact of local precipitation or evaporation and the return of solution from the residue facility.

A schematic of the Leach Facility process flowsheet is shown in Figure 16-7.

16.5.4 Acid Heap Leach Metal Recovery Plant

The highly acidic solution entering the plant near (pH=0) must be partially neutralized to approximately pH = 1.5 to conduct the cementation. Since neutralization of this acid with lime or similar material would consume a tremendous amount of acid from the heap leach operation, this scoping study envisions utilizing a commercial acid recovery system from Eco-Tec Inc. of Pickering, Ontario to return most of this acid to the heap leach prior to neutralization. Pregnant leach solution entering the recovery plant will first pass through the Eco-Tec short bed ion exchange acid purification unit (APU) where an estimated greater than 90 percent of the hydrochloric acid will be recovered on a resin and an estimated 70 percent of the pay metals will pass through the APU with the leach solution to the cementation process.

The cementation of precious metals is accomplished by contact of the acid depleted leachate with scrap iron. It has been assumed for this PEA that the silver/copper/gold "cement" recovered will be processed by South American Silver on-site to a dore' and then shipped to be toll smelted through a suitable smelter/refinery.

The precipitation of indium and gallium is accomplished by raising the pH with sodium hydroxide solution to precipitate the metal hydroxides. This is a crude precipitate and it is necessary to refine further. Section 16.3.7 describes the refining process to produce 99.99 pure indium ingots on site.

The precipitation of lead and zinc can be performed sequentially. The SGS results show that it is possible to obtain a high grade lead concentrate (analysis 75.4% Pb, 0.98% Fe and 120 ppm Zn) followed by a mixed lead-zinc concentrate (analysis 14% Zn, 61.6% Pb, 2.4% Fe).The lead and zinc sulfide precipitates can likely be toll refined as lead and zinc concentrates.

Schematics of the metal recovery process flowsheet are shown Figure 16-8 and Figure 16-9.

Figure 16-6 40,000tpd Crushing Circuit and Material Balance

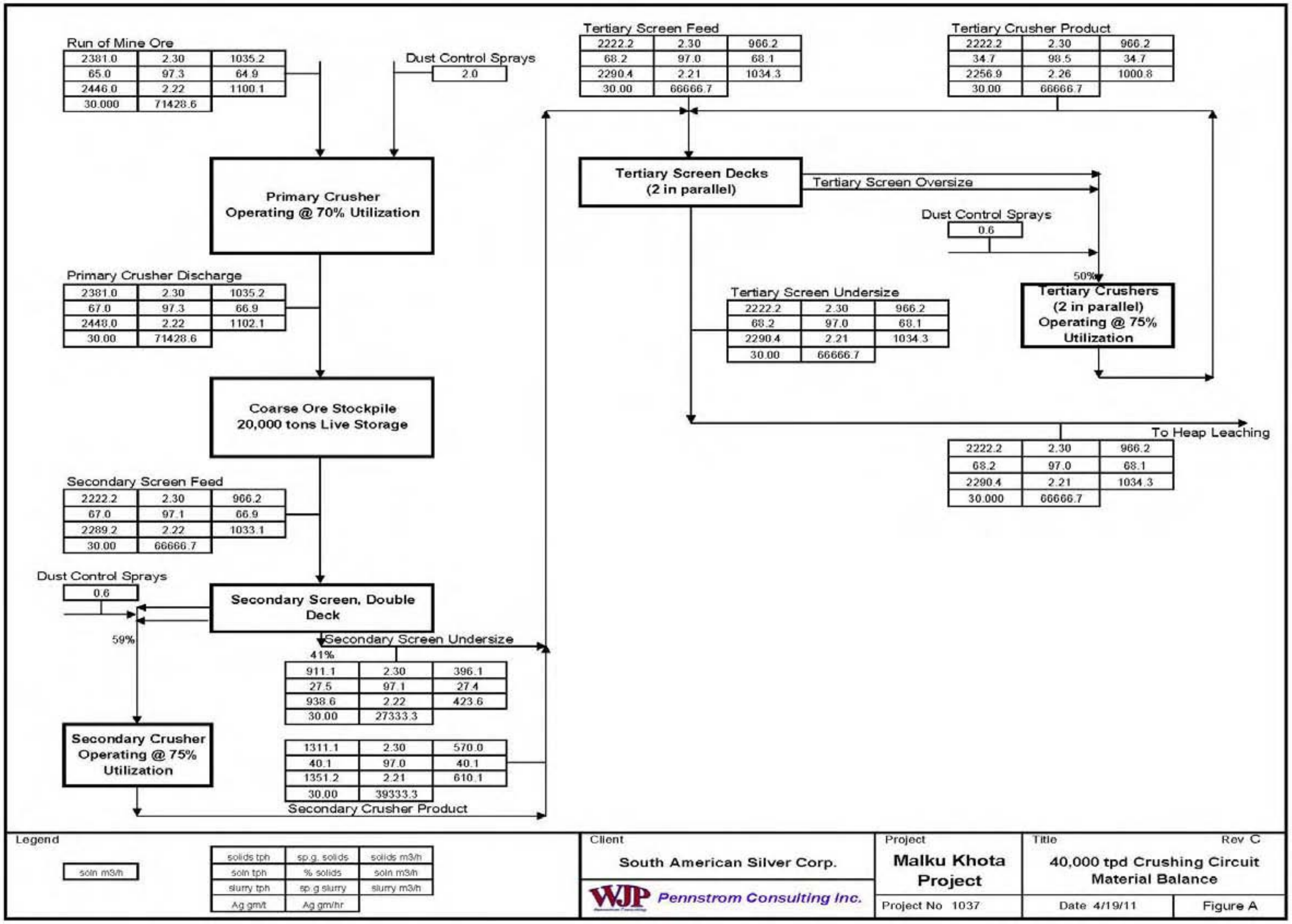


Figure 16-7 40,000tpd Heap Leach Steady State Material Balance

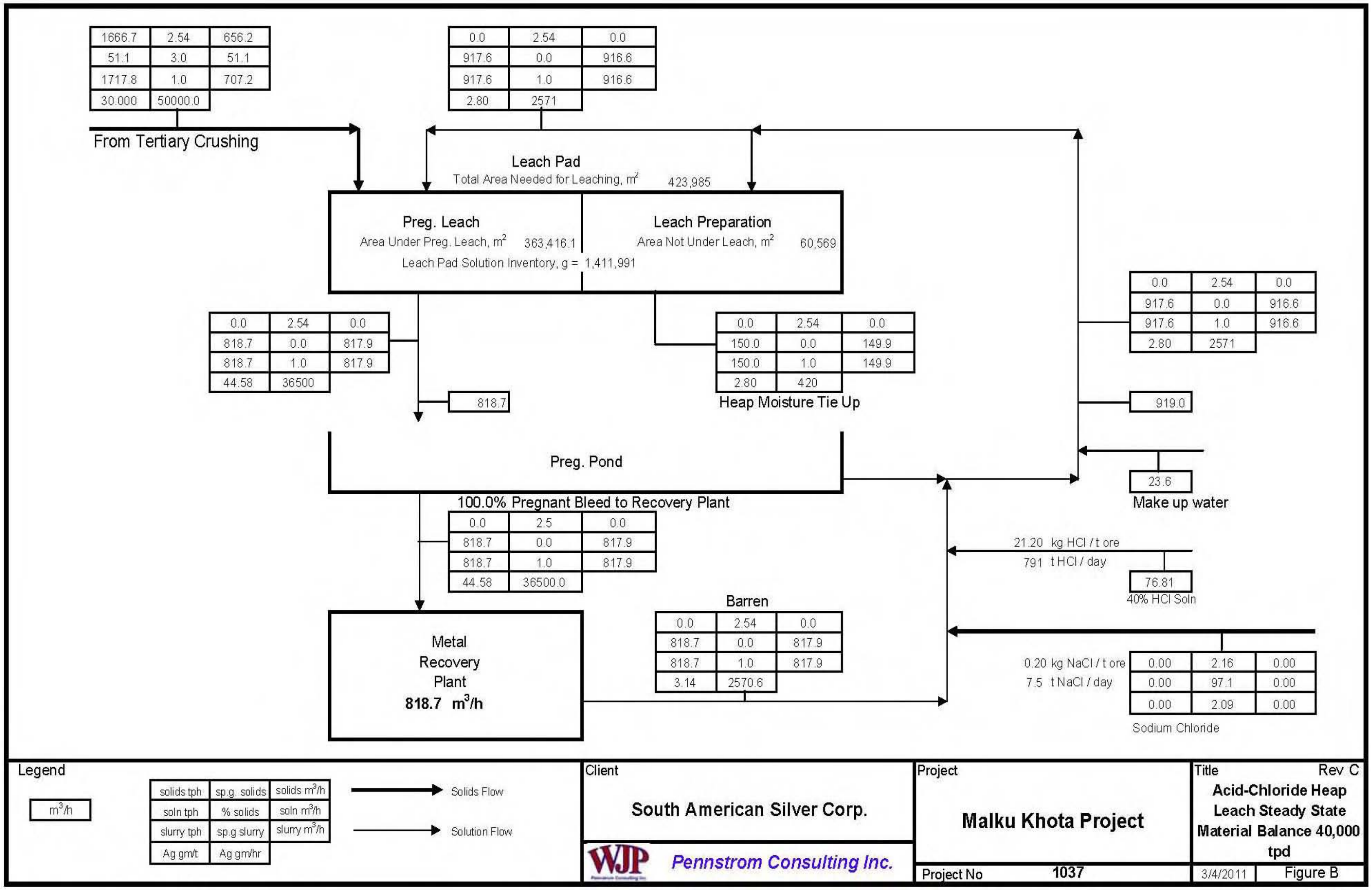


Figure 16-8 40,000tpd Metal Recovery Material Balance

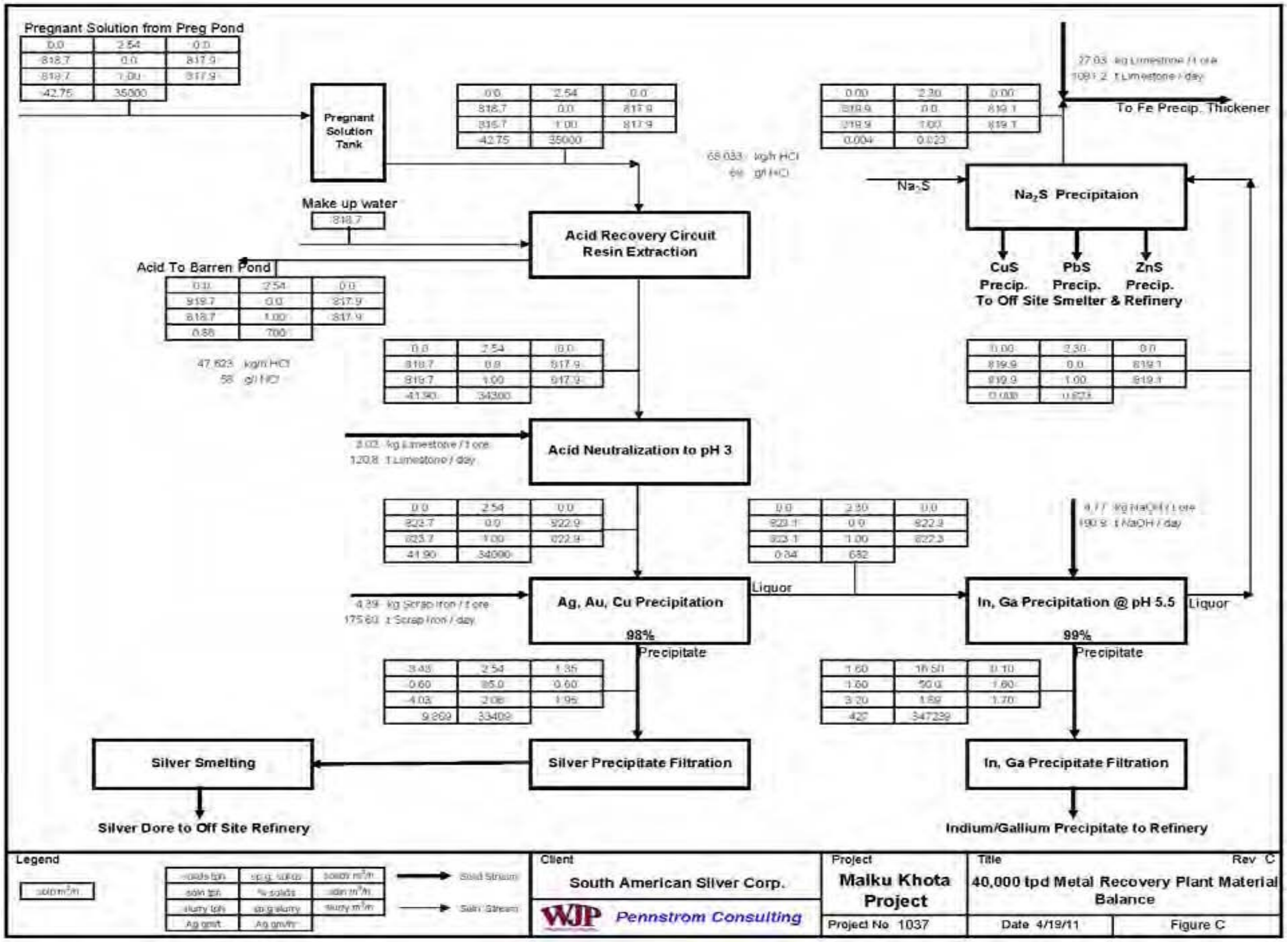


Figure 16-9 40,000tpd Tailings Circuit Material Balance

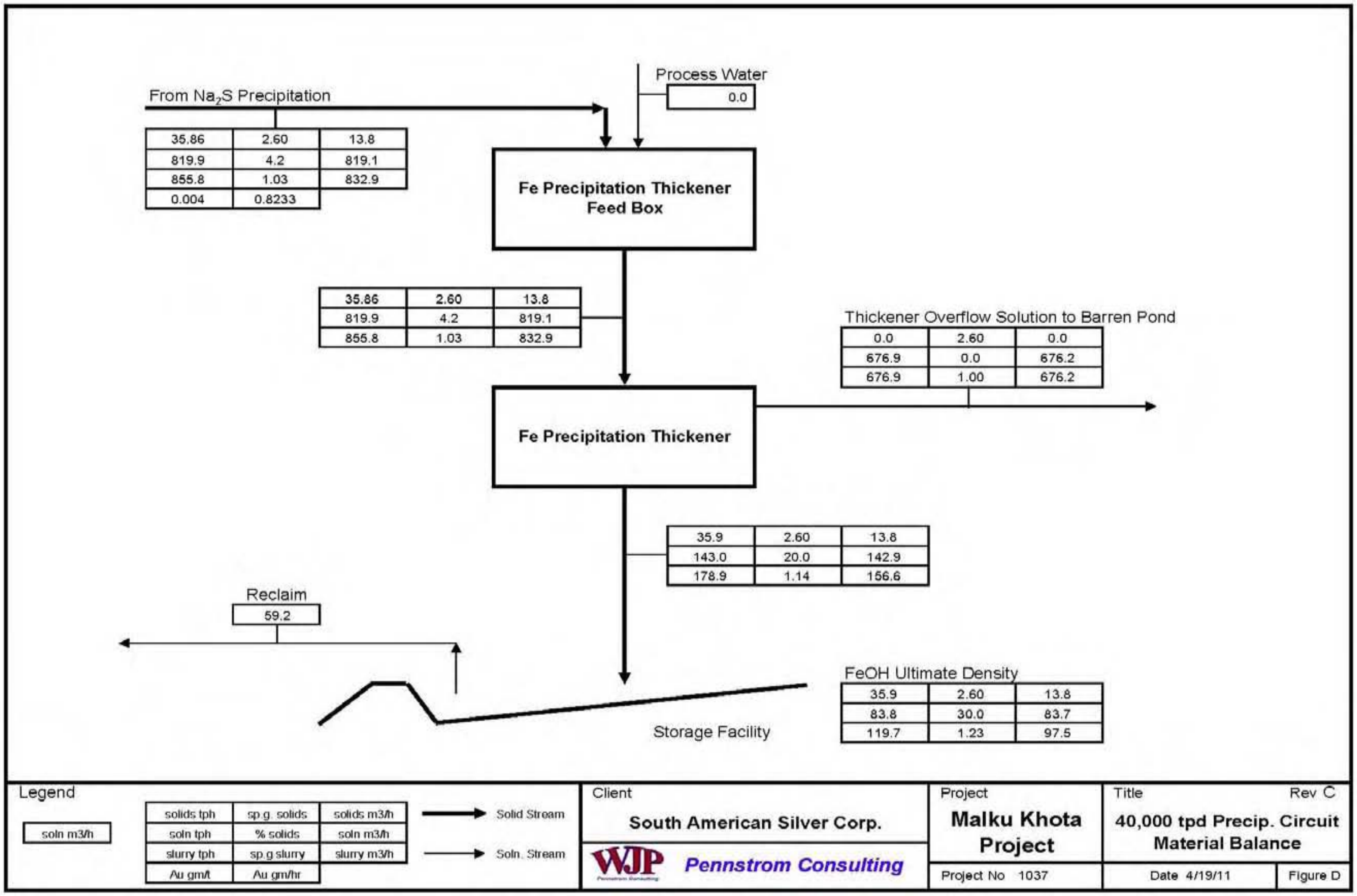
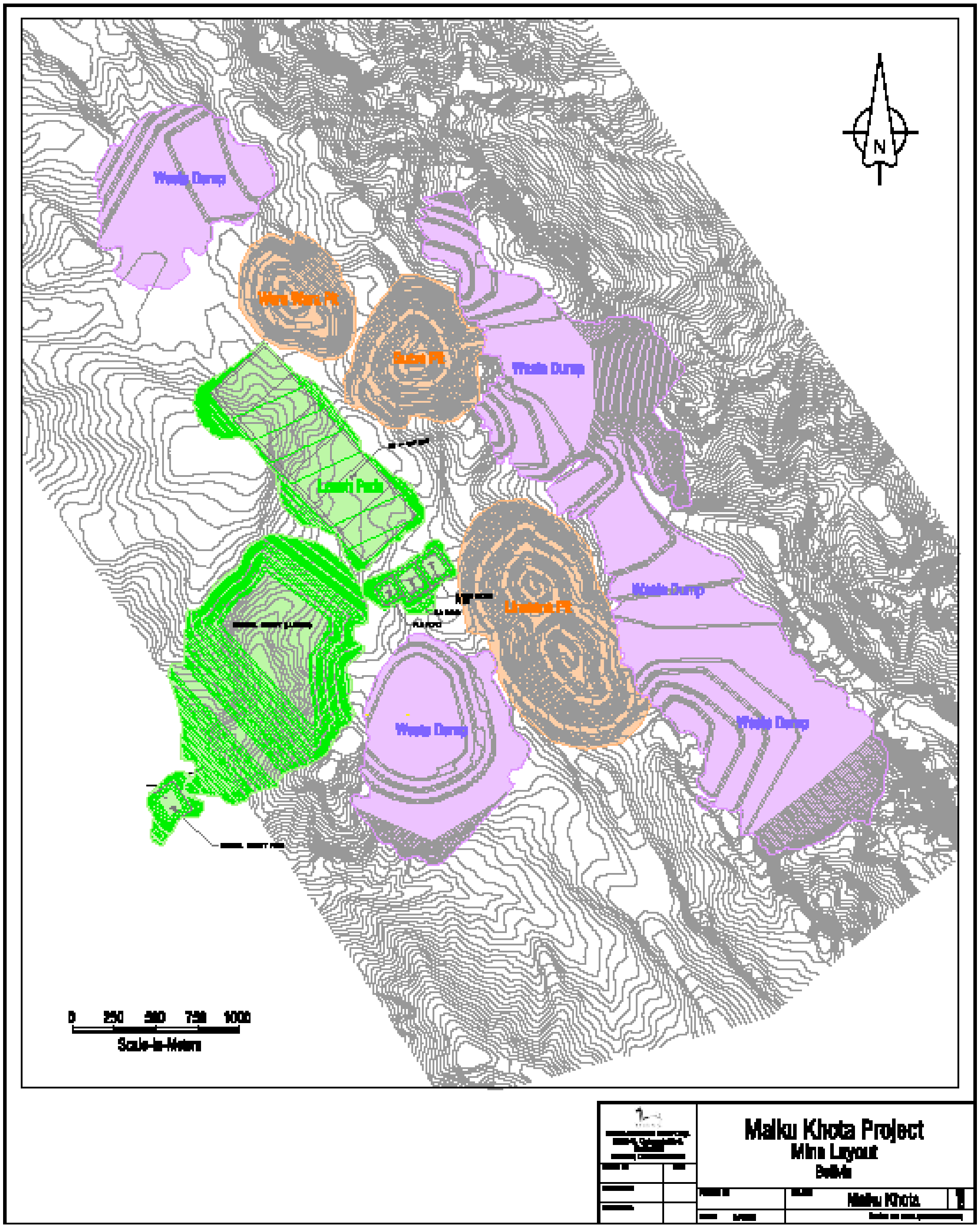


Figure 16-10 Overall Site Layout



17 MINERAL RESOURCE ESTIMATE

This resource estimate is an amendment to a 43-101 resource report commissioned by South American Silver Corp. on its Malku Khota Project in 2008, the results of which were reported on October 1, 2008. SAC reported an Indicated Resource of 144,597,000 oz Silver at a grade of 29.71 g/t and 845,000 kg of Indium at a grade of 5.58 g/t and an Inferred Resource of 177,783,000 oz Silver at a grade of 24.03 g/t and 968,000 kg of Indium at a grade of 4.21 g/t at Malku Khota using a 10 g/t Ag cut-off value. This resource was completed by the consulting group Pincock Alan & Holt ("PAH") and is described in the 2008 Technical Report on the Resource Estimate for the Malku Khota Project, November 14, 2008, by Conner et al., which is filed on SEDAR.

GeoVector Management Inc. (GeoVector) of Ottawa, Ontario and Vancouver, B.C., has been contracted by SASC to provide an updated resource for the Malku Khota project. To complete the updated resource GeoVector assessed the raw database, the available written reports, and the resource modeling data that was available from the 2008 resource report. Based on this review, GeoVector formulated new methodologies and geological models that better reflected the deposit type and the data that is available to generate the resource estimate. As the work progressed and more up to date and/or corrected data became available it was incorporated into GeoVector's studies.

Mineral Resource was estimated by Dr. Allan Armitage, P. Geol, of GeoVector. Dr. Armitage is an independent Qualified Persons as defined by NI 43-101. Practices consistent with CIM (2005) were applied to the generation of the resource estimate. There are no mineral reserves estimated for the Property at this time.

Inverse distances squared interpolation restricted to mineralized domains were used to estimate silver and indium grades (g/t Ag and In) as well as copper, lead, zinc (wt. % Cu, Pb and Zn), and gallium (g/t Ga) into the block models. Measured, Indicated and Inferred Mineral Resources are reported in summary tables in Section 17.9 below, consistent with CIM definitions required by NI 43-101 (CIM, 2005).

17.1 Drill File Preparation

In order for PAH to complete the 2008 resource, digital files containing digital topographic information, drill hole collar information, assay data and lithological logs of the drill hole intercepts were provided by SASC. These files were used to construct the resource model. The database included 75 drill holes for a total of 25,129 metres and 15,208 assays. Drill holes that were completed prior to September 1, 2008 were used in the resource estimation. This drill database is described in the 2008 Technical Report on the Resource Estimate for the Malku Khota Project, November 14, 2008, by Conner et al., which is filed on SEDAR.

Since the 2008 resource, an additional 37 drill holes totalling 14,914 metres were completed on the Malku Khota project. A total of 8,936 assays were collected from these drill holes. In order to complete an updated resource, GeoVector was provided with an updated drill hole database which included the addition of the drill holes completed from September 1, 2008 to present. The updated database included collar locations, down hole survey data, assay data, lithology data and specific gravity data. No resource models were provided to GeoVector from the previous resource.

The updated assay database was checked for errors, including overlaps and gapping in intervals by GeoVector. Typographical errors in assay values, supporting information on source of assay values, and finally a comparison of check assays, duplicates and metallic assays was completed by Pierre Desautels, P. Geo, Principal Resource Geologist for AGP Mining Consultants. Generally the database was in good shape and was accepted by GeoVector as is.

A summary of the complete drill hole database used for the updated resource is presented below (Figure 17-1). The drilling was completed in three separate areas referred to as Limosna, Wara Wara and Sucre (Figure 17-1). Approximately 50% of the drilling on the property was completed in the Limosna area. Due to the lack of drilling, it is uncertain if all three zones will connect into a much larger, continuous mineralized body.

An initial statistical analysis of assays between mineralized areas indicates minor grade variation exists (Table 17-2). Limosna shows higher average grades in silver, indium, lead and zinc than the Wara Wara zone and lower average copper and gallium. Silver grade in Limosna is higher than in Sucre, but most other elements show similar average grades. Grade variation between mineralized zones will be discussed further in section 17.4).

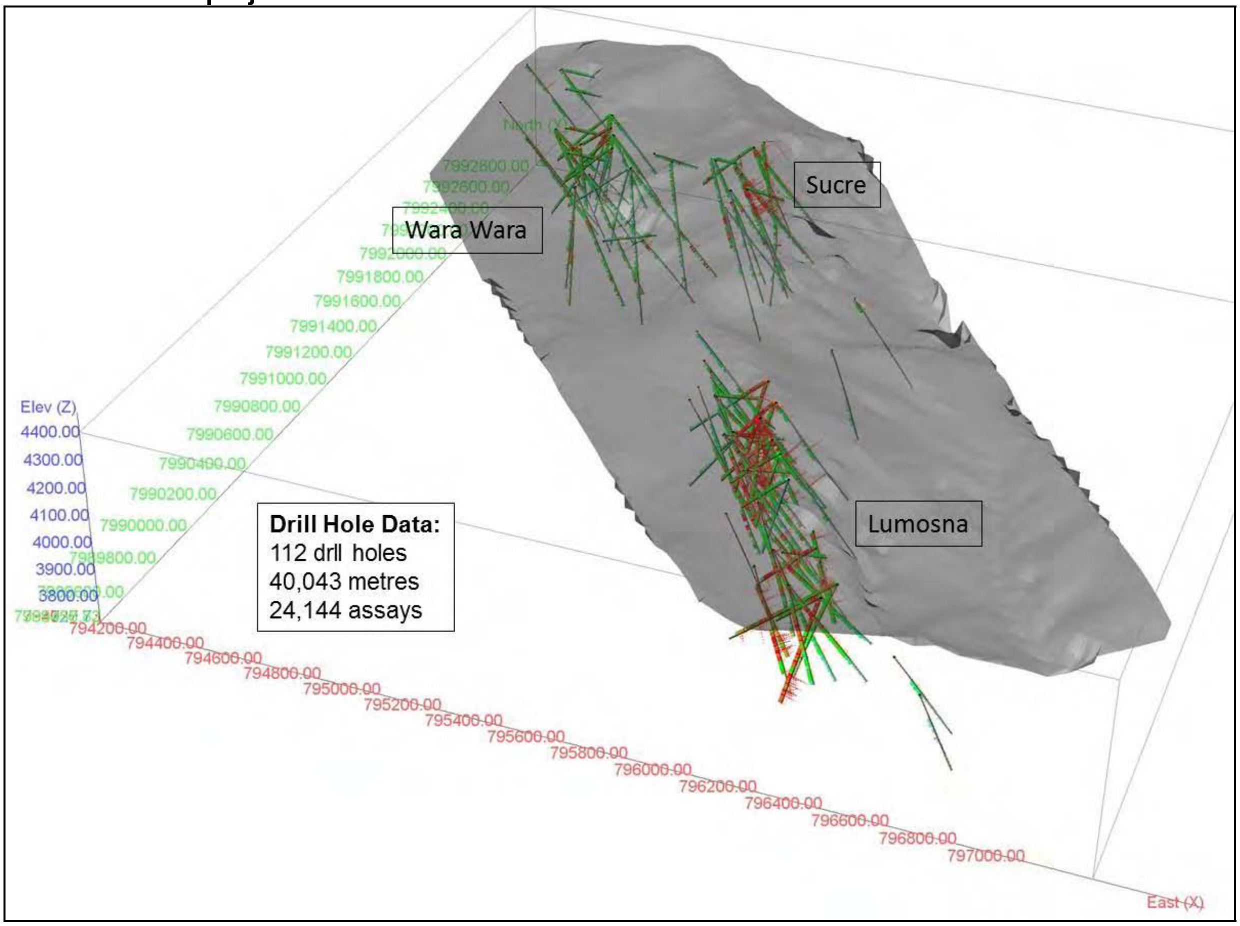
Table 17-1 Summary of the drill hole data used in the resource modeling

Number of drill holes	112
Total meters of drilling	40,043
Total number of assay samples	24,144
Total number of specific gravity samples	17,303

Table 17-2 Summary of all drill hole assay data from the Malku Khota drilling

Variable	AG (g/t)	In (g/t)	CU (%)	PB (%)	ZN (%)	Ga (g/t)
Number of samples	24,144	24,144	24,144	24,144	24,144	24,144
Minimum value	0	0	0	0	0	0
Maximum value	2,370	1,550	4.42	11.10	9.90	54.6
Mean	20.43	4.04	0.02	0.07	0.04	4.36
Median	7.53	1.73	0.00	0.02	0.01	3.80
Variance	3,612.43	189.35	50.34	680.07	438.28	8.90
Standard Deviation	60.10	13.76	0.07	0.26	0.21	2.98
Coefficient of variation	2.94	3.41	4.73	3.59	4.84	0.68
97.5 Percentile	109	22.5	0.08	0.44	0.24	11.35

Figure 17-1 Isometric view looking northwest showing the drill hole distribution on the Malku Khota project.



17.2 Resource Modeling and Wireframing

Mineralization within the Malku Khota project area is predominantly hosted within two main sandstone horizons referred to as the Wara Wara and Malku Khota sandstones (upper Ravelo Formation). Locally mineralization extends into the hanging wall sediments of the Kosmina Formation. A thin shale unit, referred to as the Lutita Edwin Shale forms at the contact of the Malku Khota sandstone unit and the overlying sediments of the Intermedia Unit of the Kosmina Formation. The stratigraphy generally strikes 320 degrees and dips 60-75 degrees to the west. North-South trending normal and thrust faults, broad synclinal and anticlinal folds and late East-West crosscutting faults resulted in local variations in the strike and dip of stratigraphy.

Mineralization within the project area consists of disseminated silver mineralization associated with indium, lead, zinc, copper and gallium together with high-grade silver sulphide and barite veins. A younger stage of gold-silver-bismuth mineralization is present that consists of high-angle and narrow veins that crosscut the earlier silver mineralization and are thought to be associated with a buried intrusion. Three main areas of mineralization have been identified and are referred to as Limosna, Wara Wara and Sucre. Mineralization extends to over 500m depth in areas and ranges in true thickness from 50 to over 400m.

3D resource models were created for the Limosna, Wara Wara and Sucre zones (Figure 17-2). Modelling was completed based on both lithology and distribution of mineralization. Resource model shapes are roughly coincident with the strike and dip of stratigraphy. Model contacts are roughly coincident with a 5 g/t cut off grade (COG) for silver.

Resource model construction involved visually interpreting mineralized zones from cross sections using histograms of silver, indium, zinc and lead values. Polygons of the mineral intersections were made on each cross section and these were wireframed together to create contiguous resource bodies in Gemcom GEMS 6.2.4 software. This modeling exercise provided broad controls of the dominant mineralizing direction. Once the model shape was accepted, a review of grade distribution within each model showed some variation in grade distribution between silver, Indium, copper, lead, zinc and gallium.

The Limosna resource model trends at 325° and dips approximately 75° to the southwest (Figure 17-2). The mineralization is hosted mainly within the Wara Wara and Malku Khota sandstones with very little mineralization extending above the Lutita Edwin Shale and into the overlying sediments. The Wara Wara model trends 325° and dips approximately 70° to the southwest. Wara Wara mineralization is hosted within the Wara Wara and Malku Khota sandstones with a significant component of the mineralization extending above the Lutita Edwin Shale and into the overlying sediments (Figure 17-3). The Sucre model trends 215° and dips 60° to the northwest. Sucre mineralization is primarily hosted within the Wara Wara sandstone.

Extensive low grade mineralization extends into the hanging wall and footwall of all three mineralized zones. A low grade mineralization Halo model was constructed around the main mineralized areas.

Figure 17-2 Isometric view looking northwest showing the Limosna, Wara Wara and Sucre resource models and the Lutita Edwin Shale

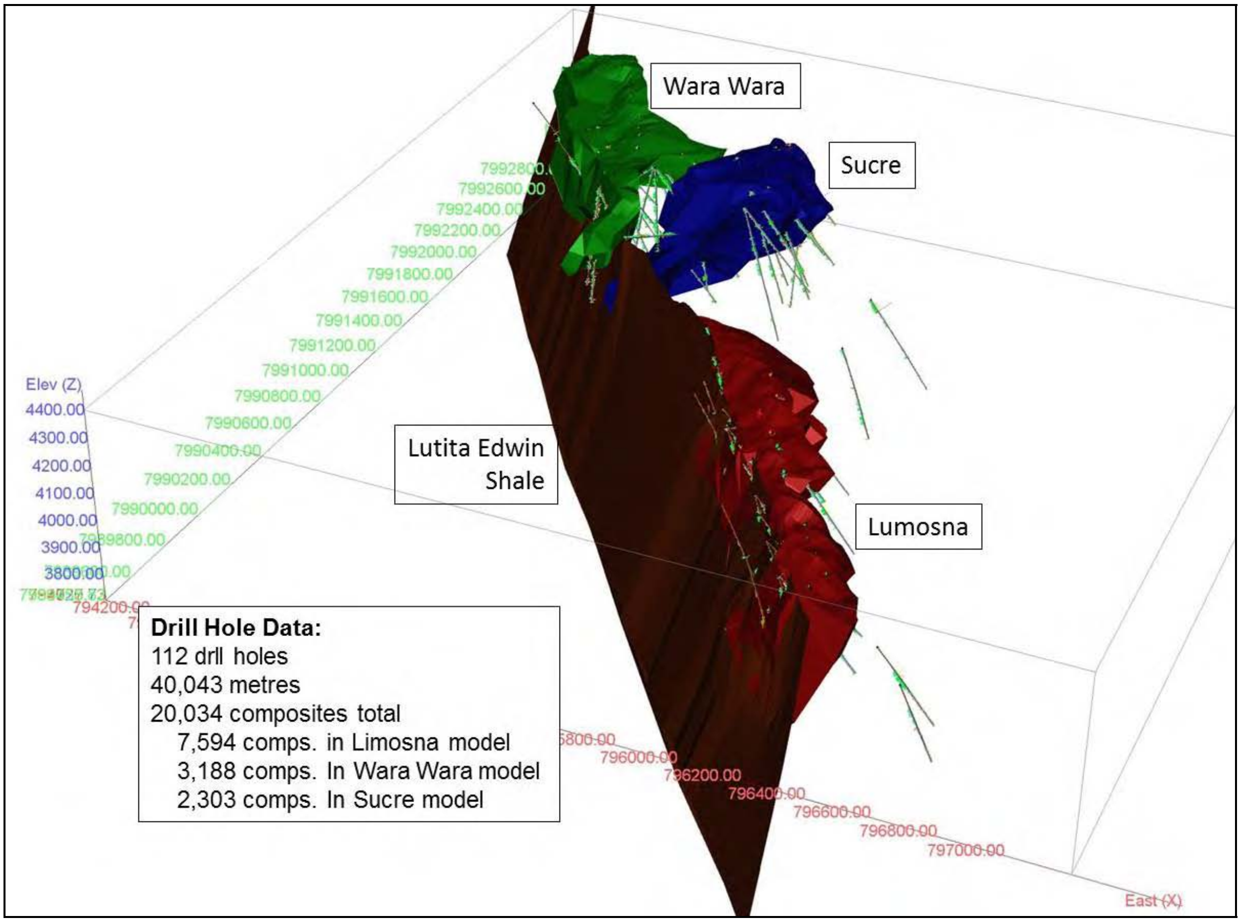
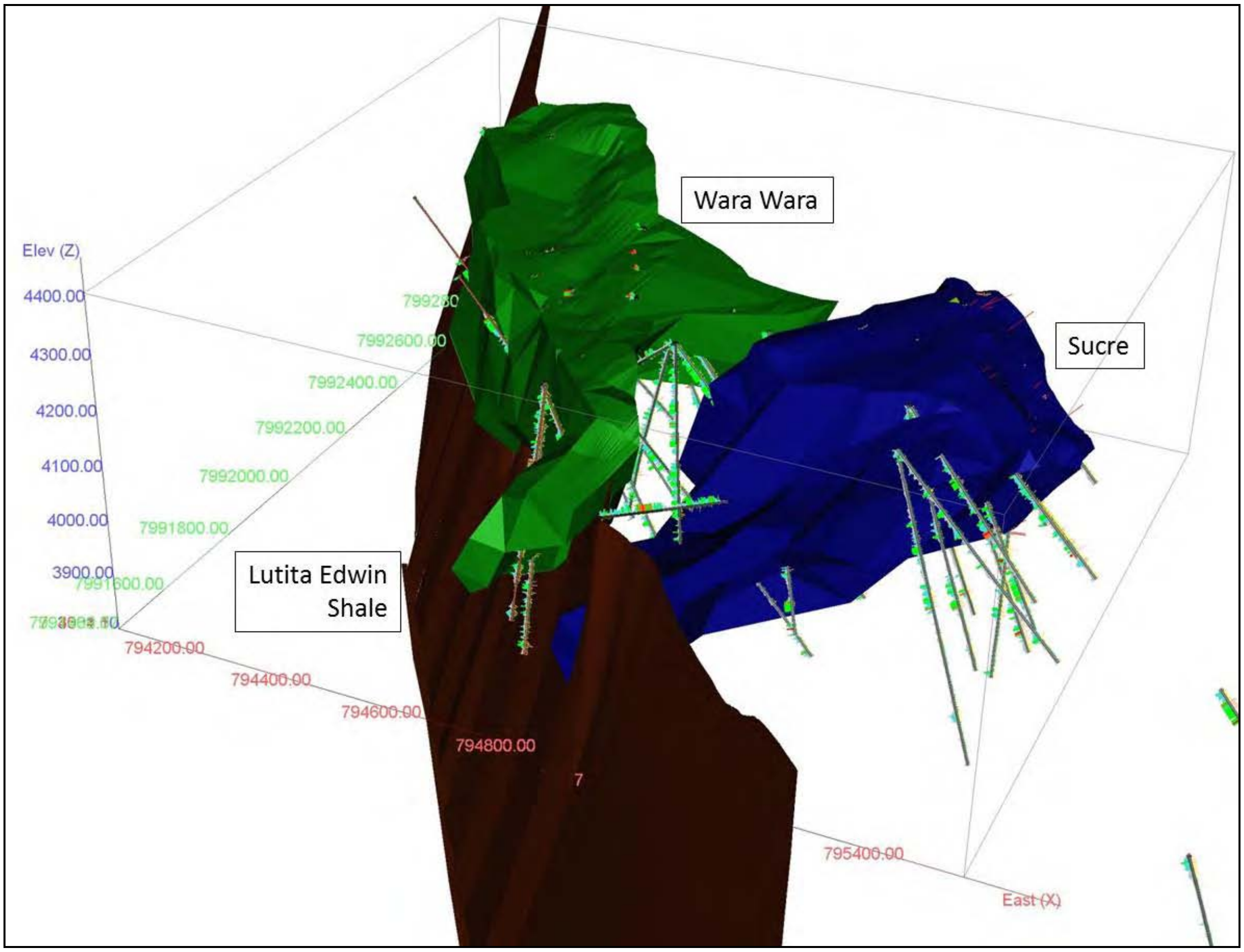


Figure 17-3 Isometric view looking northwest showing the Wara Wara and Sucre resource models and the Lutita Edwin Shale



17.3 Composites

Analysis of the assay sample population used in the previous resource estimation is described in the 2008 Technical Report on the Resource Estimate for the Malku Khota Project, November 14, 2008, by Conner et al., which is filed on SEDAR. The analysis concluded that a composite length of 2m would minimize the dilution due to compositing and be appropriate for a resource model with block dimensions of 10m X 10m X 10m.

The assay sample database available for the updated resource totalled 24,144 samples representing 40,043 metres of core. Average width of the sample intervals was 1.66 meters, within a range of 0.1 meters to 5.1 meters. Of the total assay population ~45% were 2 meter samples and only 248 assays (~1.0%) were greater than 2 meters. For consistencies, two metre composites were used for the updated resource.

Composites were generated starting from the collar of each hole and totalled 20,034 (Table 17-3). For the updated resource, composite populations were generated for each of the three mineralized domains, including Limosna, Wara Wara and Sucre, with each composite population constrained by the samples within those domains. These composite values were used to interpolate grade into their respective resource models. Average grades of silver, Indium, copper, lead, zinc and gallium were marginally lower in the composite population, but this was mostly due to the included "zero" values.

Table 17-3 Summary of the drill hole composite data from the Malku Khota drilling

Variable	AG (g/t)	In (g/t)	CU (%)	PB (%)	ZN (%)	Ga (g/t)
Number of samples	20,034	20,034	20,034	20,034	20,034	20,034
Minimum value	0	0	0	0	0	0
Maximum value	1457	561.52	2.80	6.92	9.06	41.30
Mean	19.63	3.88	0.01	0.07	0.04	4.29
Median	7.96	1.77	0.00	0.03	0.01	3.81
Variance	2,374.27	73.16	22.94	501.76	329.43	6.47
Standard Deviation	48.73	8.55	0.05	0.22	0.18	2.54
Coefficient of variation	2.48	2.20	3.52	3.21	4.38	0.59
97.5 Percentile	106.47	21.10	0.07	0.42	0.24	11

17.4 Grade Capping

An analysis of the 2008 composite database is described in the 2008 Technical Report on the Resource Estimate for the Malku Khota Project, November 14, 2008, by Conner et al., which is filed on SEDAR. PAH concluded based on an analysis of the distribution for Ag and In within the complete composite database indicates that outliers are present, hence, the application of a high grade cut is considered appropriate prior to using the data for any linear grade interpolation. Analysis of separate elements resulted in the application of a high grade cut of 650 g/t for Ag and 200 ppm for In. Zn and Pb did not require the application of a high grade cut.

For the updated resource, the composites were domained into mineralization and waste based on whether they intersected the individual resource models. A total of 13,085 composite sample points occur within the Limosna, Wara Wara or Sucre resource model (Table 17-4). These values were used to interpolate grade into their respective resource blocks. An analysis of the composites from each area indicates there are minor grade variations from each area.

Based on a statistical analysis of the composite database from each resource model, it was decided that no capping was required on the composite populations to limit high values. Descriptive statistics of the composited values for Au, Ag, Cu, Pb and Zn are shown in Table 9. Histograms of the data indicate a relatively log normal distribution of all metals with very few outliers within the database. Analyses of the spatial location of these samples and the sample values proximal to them led GeoVector to believe that the high values were legitimate parts of the population, and that the impact of including these high composite values uncut would be negligible to the overall resource estimate.

Table 17-4 Summary of the drill hole composite data from within the Limosna, Wara Wara and Sucre resource models

Limosna Composites	AG (g/t)	In (g/t)	CU (%)	PB (%)	ZN (%)	Ga (g/t)
Number of samples	7,594	7,594	7,594	7,594	7,594	7,594
Minimum value	0	0	0	0	0	0
Maximum value	1457	151.10	0.49	6.92	9.06	26.53
Mean	32.54	5.92	0.01	0.13	0.06	2.93
Median	16.88	3.13	0.00	0.05	0.02	2.83
Variance	4,549	81	3	1,153	773	2
Standard Deviation	67.44	9.00	0.02	0.34	0.28	1.47
Coefficient of variation	2.07	1.52	2.52	2.62	4.31	0.50
99 Percentile	291.70	43.65	0.06	1.46	0.83	7

Wara Wara Composites	AG (g/t)	In (g/t)	CU (%)	PB (%)	ZN (%)	Ga (g/t)
Number of samples	3,188	3,188	3,188	3,188	3,188	3,188
Minimum value	0.105	0.015	0.00	0.00	0.00	0.29
Maximum value	569	44.55	0.70	0.71	0.22	41.30
Mean	20.42	2.48	0.02	0.03	0.00	5.40
Median	12.13	1.67	0.01	0.02	0.00	4.75
Variance	893	9	9	10	1	8
Standard Deviation	29.88	2.98	0.03	0.03	0.01	2.78
Coefficient of variation	1.46	1.20	1.82	1.10	2.52	0.52
99 Percentile	148.11	14.63	0.12	0.15	0.06	15

Sucre Composites	AG (g/t)	In (g/t)	CU (%)	PB (%)	ZN (%)	Ga (g/t)
Number of samples	2,303	2,303	2,303	2,303	2,303	2,303
Minimum value	0.03	0	0.00	0.00	0.00	0
Maximum value	716	561.52	2.80	1.80	1.35	22.60
Mean	25.52	6.39	0.04	0.04	0.03	5.09
Median	9.93	3.06	0.02	0.03	0.01	4.76
Variance	2,678	303	144	50	39	4
Standard Deviation	51.75	17.41	0.12	0.07	0.06	1.92
Coefficient of variation	2.03	2.72	2.74	1.68	2.30	0.38
99 Percentile	248.57	49.02	0.35	0.24	0.30	11

17.5 Block Modeling

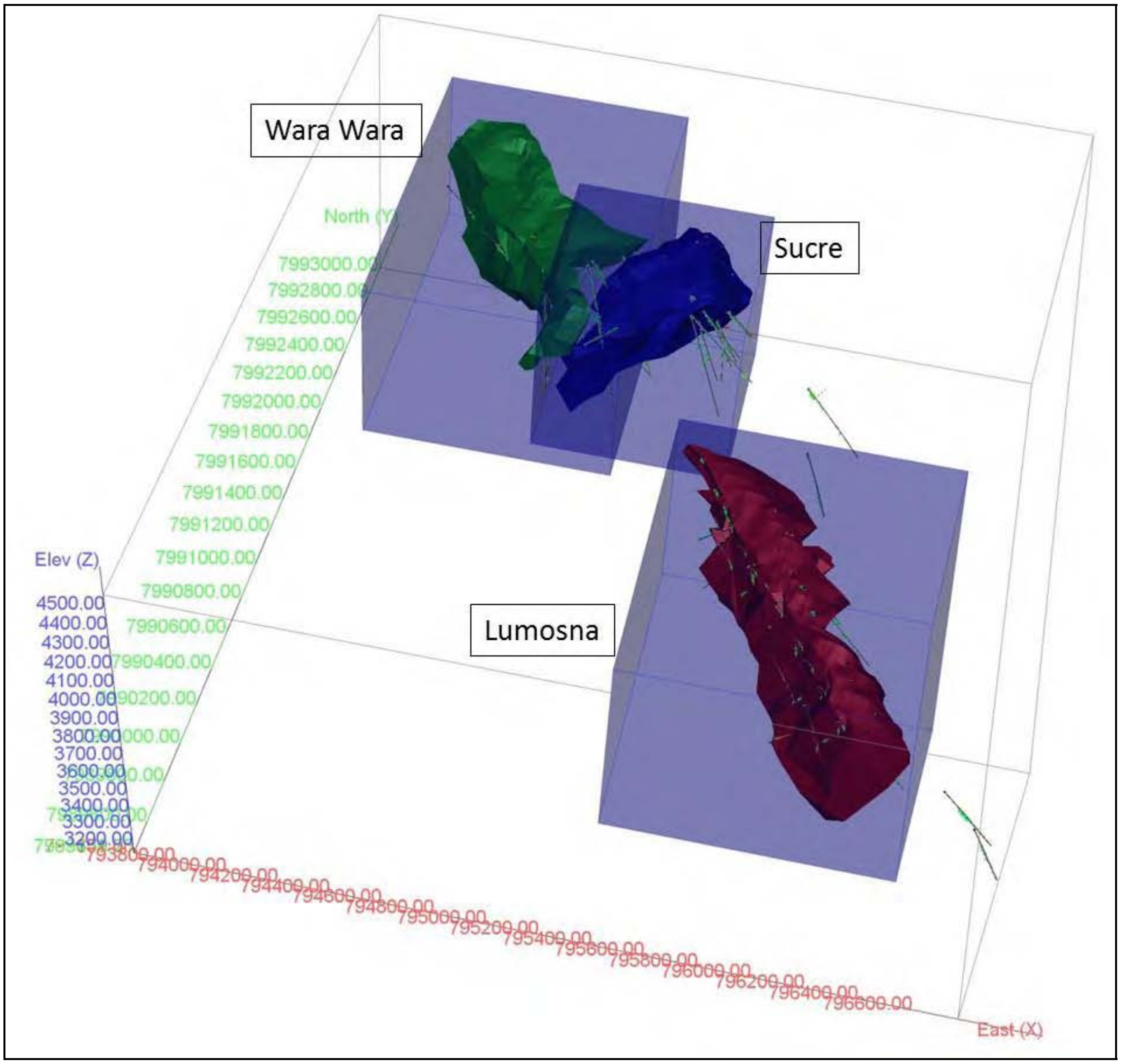
Separate block models were created for each of the three mineralized zones within WGS84 UTM19 space. Block model dimensions are listed in Table 17-5. Block model size was designed to reflect the spatial distribution of the raw data – i.e. the drill hole spacing (Figure 17-4) within each mineralized zone. It was decided to create resource blocks that were 10 x 10 x 10 metre in size in the X, Y and Z directions respectively. At this scale of the deposit this still provides a reasonable block size for discerning grade distribution, while still being large enough not to mislead when looking at higher cut-off grade distribution within the model. The model was intersected with surface topography to exclude blocks, or portions of blocks, that extend above the bedrock surface.

A much larger north-south trending block model, which covers the extents of the Malku Khota deposit area was used to interpolate mineralization in the low grade Halo. Blocks 10 x 10 x 10 metre in size in the X, Y and Z directions were used.

Table 17-5 Block model geometry

Model Name	Limosna			Wara Wara			Sucre		
	X	Y	Z	X	Y	Z	X	Y	Z
Origin (WGMS84, UTM19)	795350	7989700	4400	794000	7991400	4400	794700	7991450	4450
Extent	1100	1300	850	1050	150	750	820	800	750
Block Size	10	10	10	10	10	10	10	10	10
Rotation	0°			0°			0°		

Figure 17-4 Isometric view looking northwest showing the Limosna, Wara Wara and Sucre block resource models.



17.6 Specific Gravity

A review of the specific gravity (SG) data available to September 1, 2008 was completed by PAH (Conner et al., 2008). PAH reviewed 9,915 SG samples, which were taken from throughout the deposit. The mean of these samples was 2.3t/m^3 and this value was used for the previous resource model.

The updated database supplied to GeoVector by SASC included an additional 4,364 SG samples for a total of 14,279 samples (Table 17-6). The mean of the complete data set is 2.3t/m^3 with small variance. The SG data was also analysed based on samples which occur within or outside of the mineralized domains. The average value for Limosna is 2.28t/m^3, for Wara Wara 2.41t/m^3 and for Sucre 2.37t/m^3. Samples which fall outside of the mineralized domains averaged 2.25t/m^3. With increasing grade the SG values of mineralized rocks samples increase.

Based on an analysis of the SG values of samples from within the mineralized domains it was decided that the average SG value of 2.35t/m^3 be used for the updated resource estimate. This is 0.05t/m^3 higher that the value used in the previous resource. An SG value of 2.30t/m^3 was used for the low grade mineralized Halo.

Table 17-6 Summary of the statistics of the SG data from the Malku Khota Project

Variable	Malku Khota	Limosna	Wara Wara	Sucre	Non-mineralized
Number of samples	14,279	6,067	1,993	1,398	4,821
Minimum value	1.42	1.42	1.87	1.8	1.6
Maximum value	3.76	3.01	3.61	3.76	3.2
Mean	2.3	2.29	2.39	2.35	2.26
Median	2.28	2.28	2.41	2.36	2.25
Variance	0.02	0.01	0.02	0.02	0.01
Standard Deviation	0.12	0.09	0.13	0.15	0.12
Coefficient of variation	0.05	0.04	0.06	0.07	0.05
97.5 Percentile	2.57	2.52	2.59	2.63	2.56

17.7 Grade Interpolation

The primary aim of the interpolation was to fill all the blocks within the three resource models with grade. To generate grade within the blocks inverse distance squared (ID2) was used. The size of the search ellipse, in the X, Y, and Z direction, used to interpolate grade into the resource blocks is based on 3D semi-variography analysis of mineralized points within each resource model (Table 17-7). The long axis of all three search ellipses was oriented to reflect the observed preferential long axis (geological trend) of the resource model (Figure 17-5, Figure 17-6 and Figure 17-7). The short Y direction reflects the roughly 1/2 distance of the model in this direction relative to the longer axis. The dip axis of the search ellipse was set to reflect the observed trend of the mineralization down dip. Parameters used to estimate grade into each block of the individual resource models created are listed in Table 17-7.

Table 17-7 Interpolation parameters

Parameter	Limosna			Wara Wara			Sucre		
	Inferred	Indicated	Measured	Inferred	Indicated	Measured	Inferred	Indicated	Measured
Search Type	Ellipsoid			Ellipsoid			Ellipsoid		
Bearing	145°			145°			215°		
Dip	75°			70°			40°		
Anisotropy X	130	65	32.5	190	95	47.5	190	95	47.5
Anisotropy Y	65	32.5	16.25	95	47.5	23.5	95	47.5	23.5
Anisotropy Z	130	65	32.5	190	95	47.5	190	95	47.5
Min. Samples	2	4	8	2	4	8	2	4	8
Max. Samples	20	20	20	20	20	20	20	20	20
Min. Drill Holes	1	1	4	1	1	4	1	1	4
Percentage Blocks Filled	64%	31%	5%	60%	32%	8%	55%	37%	8%

Figure 17-5 Isometric view looking northwest showing the distribution of resource blocks and the search ellipse orientation for the Limosna resource model

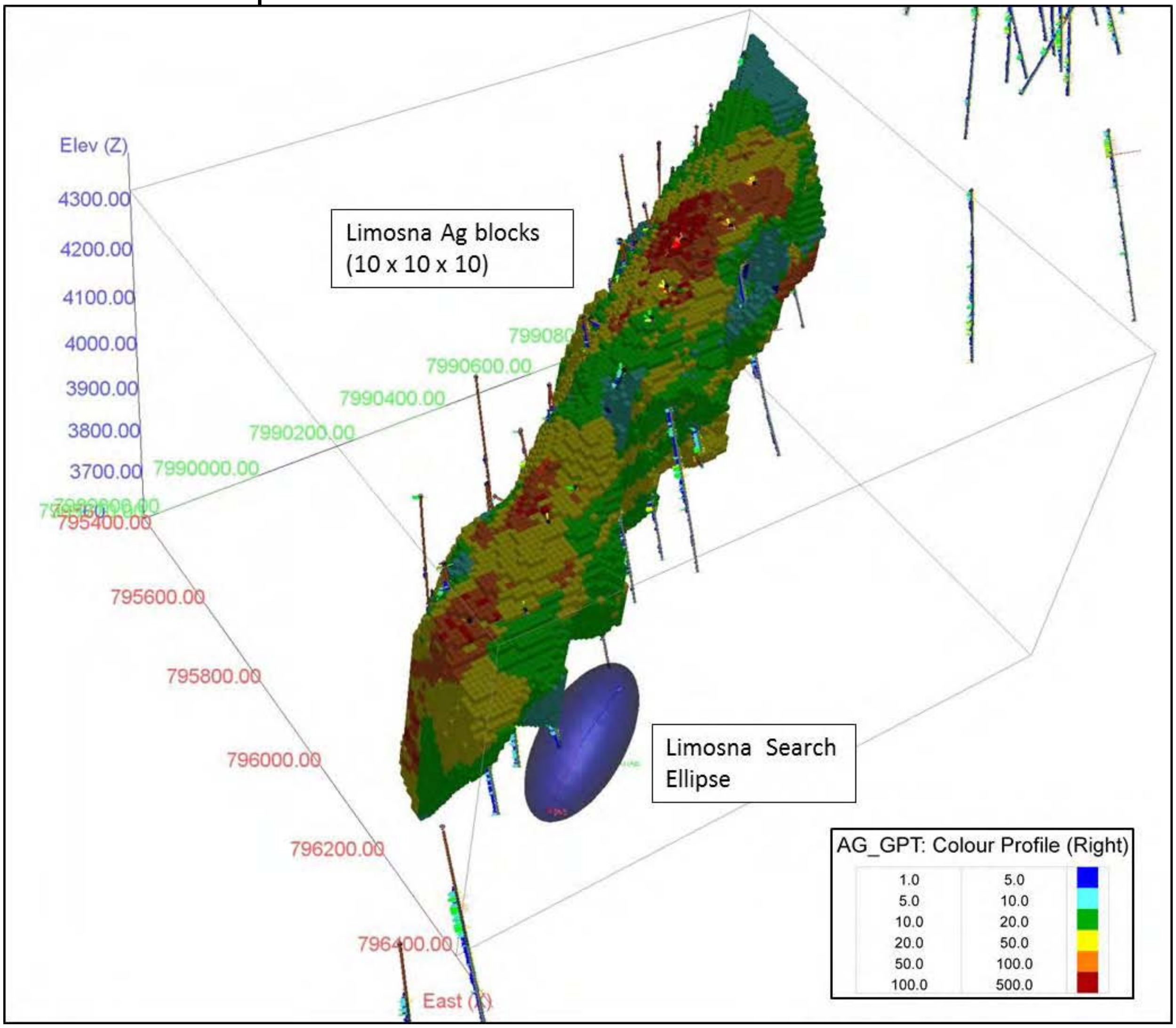


Figure 17-6 Isometric view looking northwest showing the distribution of resource blocks and the search ellipse orientation for the Wara Wara resource model

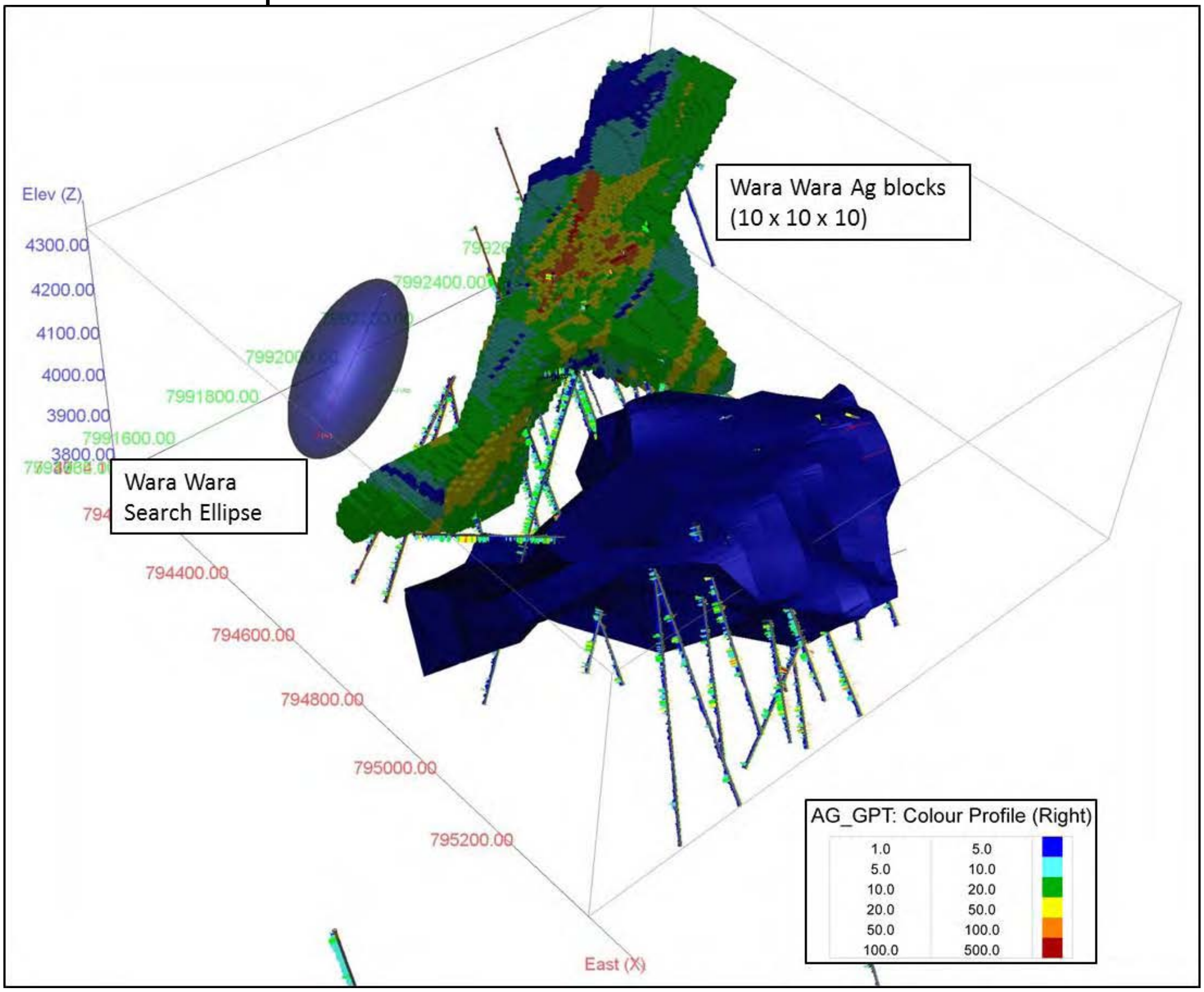
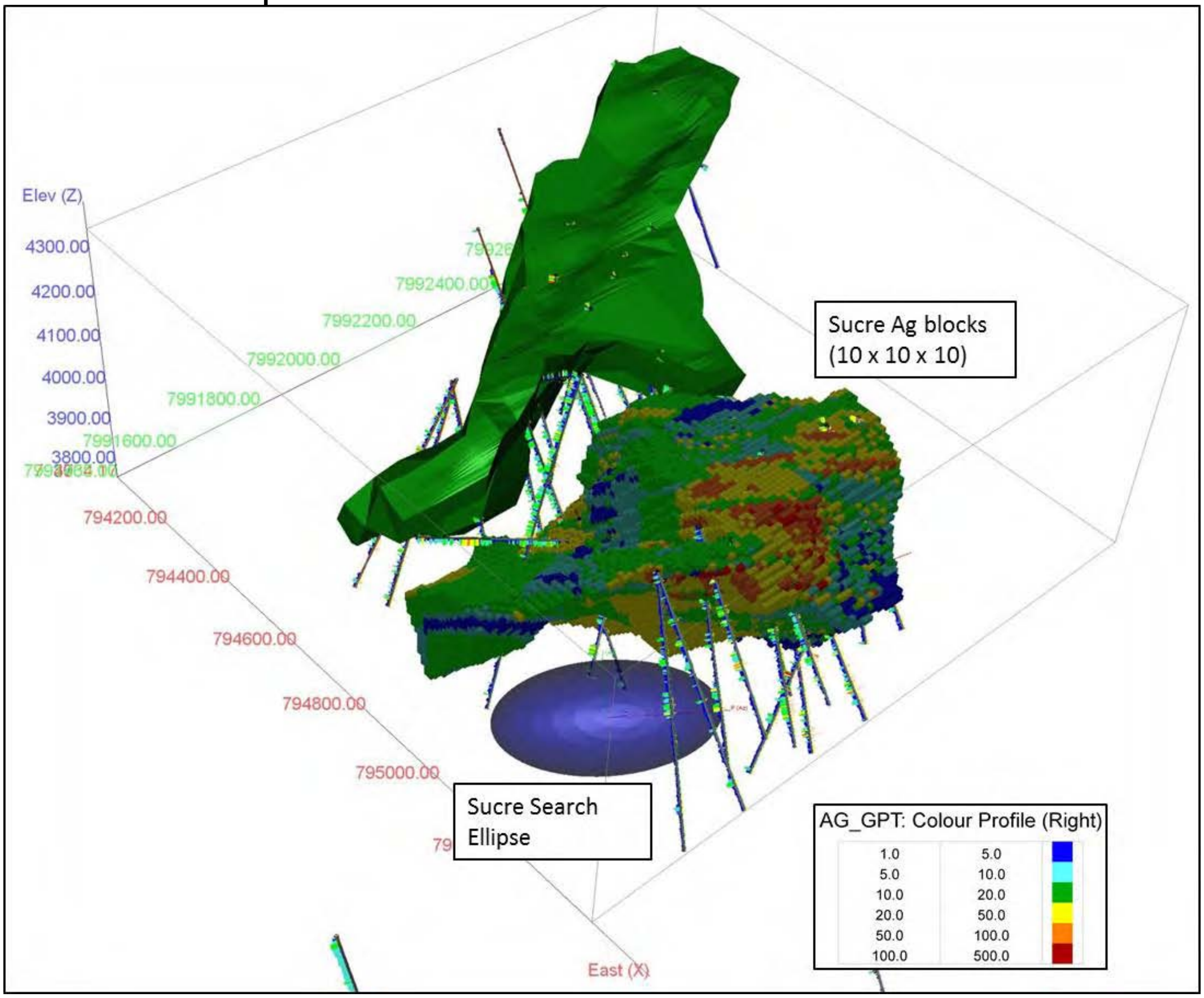


Figure 17-7 Isometric view looking northwest showing the distribution of resource blocks and the search ellipse orientation for the Sucre resource model



17.8 Model Validation

The total volume of the blocks in each resource model, at a 0 cut-off grade value compared to the volume of each wireframe model was essentially identical. The size of the search ellipse and the number of samples used to interpolate grade achieved the desired effect of filling the resource models and very few blocks had zero grade interpolated into them.

Because ID2 interpolation was used the drill hole intersection grades would be expected to show good correlation with the modelled block grades. Visual checks of block grades of silver, Indium, lead and zinc against the composite data on vertical section and in 3D showed excellent correlation between block grades and drill intersections. All three models are considered valid.

17.9 Block Model Classification

The Mineral Resource estimate is classified in accordance with the CIM Definition Standards (2005). The confidence classification is based on an understanding of geological controls of the mineralization, and the drill hole pierce point spacing in the three resource areas. The resource estimate in areas with drill spacing of less than 60 m is classified as Measured and in areas with drill densities between 60 and 100 m is classified as Indicated. The majority of the Measured and Indicated resources are restricted to the upper parts of the resource areas. A significant portion of the total resource is classified as Inferred due to the sparse drill density (> 100 metre) in parts of the resource areas. Mineralization within the low grade mineralized Halo is considered Inferred.

17.10 Resource Reporting

The grade and tonnage estimates contained herein are classified as Measured, Indicated or Inferred Resource given CIM definition Standards for Mineral Resources and Mineral Reserves (2005). As such, it is understood that:

17.10.1 Inferred Mineral Resource:

An ‘Inferred Mineral Resource’ is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

17.10.2 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 can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

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 Preliminary Feasibility Study which can serve as the basis for major development decisions.

17.10.3 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 so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

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 of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

GeoVector has estimated a range of Measured, Indicated and Inferred resources at various silver equivalent (Ag_Eq) cut-off grades for the Limosna, Wara Wara, Sucre Zones and low grade Halo Zones (Table 17-8 to Table 17-11) and the total Malku Khota Project area (Table 17-12). The Ag_Eq grade is based on $16.00/oz for silver and $550/kg for In. The final resource for the Limosna, Wara Wara, Sucre and low grade Halo zones and total Malku Khota Property is reported at a 10 g/t Ag_Eq cut-off grade (Table 17-13). Estimated metal content does not include any consideration of mining, mineral processing, or metallurgical recoveries.

Table 17-8 Measured, Indicated and Inferred Resources for the Limosna Zone

Measured Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	11,721,799	37.2	14,016,622	6.0	70	43.6	16,418,066	0.01	1,906,559	0.12	31,273,398	0.04	10,480,159	2.9	33.4
5.0	11,604,299	37.5	14,007,608	6.0	70	44.0	16,403,190	0.01	1,901,483	0.12	31,067,429	0.04	10,435,446	2.8	33.0
10	**10,781,799**	**40.0**	**13,853,996**	**6.3**	**68**	**46.7**	**16,191,247**	**0.01**	**1,836,252**	**0.13**	**30,150,033**	**0.04**	**9,405,115**	**2.8**	**30.3**
20	8,246,149	48.7	12,920,388	7.1	59	56.4	14,942,717	0.01	1,581,744	0.14	26,222,439	0.04	6,756,150	2.7	22.2
30	6,039,500	60.0	11,649,359	7.5	45	68.0	13,202,187	0.01	1,353,479	0.16	20,731,042	0.04	4,781,948	2.5	15.3
40	4,460,300	71.5	10,258,061	7.6	34	79.7	11,427,785	0.01	1,144,619	0.16	15,558,740	0.04	3,486,850	2.5	11.1
50	3,224,200	85.5	8,865,840	7.1	23	93.1	9,650,122	0.01	970,800	0.17	12,392,113	0.04	2,621,514	2.4	7.6
100	766,100	168.0	4,137,505	4.4	3	172.7	4,254,586	0.03	440,166	0.24	4,055,734	0.04	604,116	2.0	1.5
200	183,300	293.0	1,726,623	3.4	1	296.6	1,748,308	0.04	149,884	0.24	983,486	0.02	94,004	1.9	0.3
500	16,450	567.0	299,899	4.3	0	571.6	302,343	0.05	17,827	0.18	64,645	0.02	7,102	1.6	0.03
Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	123,837,943	26.7	106,501,315	6.4	792	33.6	133,716,640	0.01	14,394,945	0.12	326,224,011	0.07	200,140,491	3.1	378.5
5.0	122,778,093	27.0	106,404,452	6.4	790	33.8	133,568,601	0.01	14,354,547	0.12	325,246,143	0.07	199,533,780	3.1	375.2
10	**110,769,594**	**29.3**	**104,320,806**	**6.9**	**763**	**36.7**	**130,537,169**	**0.01**	**13,416,240**	**0.13**	**314,792,778**	**0.08**	**190,322,407**	**3.1**	**338.0**
20	73,787,646	38.6	91,471,790	8.4	619	47.5	112,763,884	0.01	10,342,733	0.16	254,038,202	0.09	150,149,364	3.0	220.0
30	47,860,097	49.7	76,483,512	9.5	456	59.9	92,161,819	0.01	7,718,266	0.18	193,743,194	0.11	114,842,737	2.8	135.6
40	30,643,998	63.5	62,567,483	10.0	305	74.1	73,059,440	0.01	6,024,888	0.21	143,775,876	0.13	89,839,687	2.7	81.9
50	20,430,899	78.9	51,827,215	9.3	190	88.8	58,359,890	0.01	4,666,977	0.24	107,722,443	0.16	72,410,492	2.6	53.2
100	4,105,450	181.5	23,963,527	4.0	16	185.8	24,524,876	0.02	1,533,045	0.17	15,286,402	0.04	3,948,610	2.5	10.2
200	1,384,150	279.2	12,425,235	3.2	4	282.6	12,576,258	0.02	661,505	0.16	4,889,706	0.03	918,589	2.3	3.2
500	9,400	517.3	156,348	3.7	0.03	521.2	157,527	0.03	7,188	0.25	52,039	0.04	8,010	2.0	0.02
Measured + Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
1.0	135,559,742	27.6	120,517,937	6.4	862	34.4	150,134,706	0.01	16,301,504	0.12	357,497,409	0.07	210,620,651	3.0	411.9
5.0	134,382,392	27.9	120,412,059	6.4	860	34.7	149,971,791	0.01	16,256,030	0.12	356,313,571	0.07	209,969,226	3.0	408.2
10	**121,551,393**	**30.2**	**118,174,802**	**6.8**	**831**	**37.5**	**146,728,417**	**0.01**	**15,252,493**	**0.13**	**344,942,811**	**0.07**	**199,727,522**	**3.0**	**368.4**
20	82,033,795	39.6	104,392,178	8.3	678	48.4	127,706,602	0.01	11,924,477	0.15	280,260,641	0.09	156,905,514	3.0	242.2
30	53,899,597	50.9	88,132,871	9.3	501	60.8	105,364,007	0.01	9,071,745	0.18	214,474,236	0.10	119,624,685	2.8	151.0
40	35,104,298	64.5	72,825,544	9.7	339	74.8	84,487,226	0.01	7,169,507	0.21	159,334,615	0.12	93,326,538	2.6	92.9
50	23,655,099	79.8	60,693,055	9.0	213	89.4	68,010,012	0.01	5,637,777	0.23	120,114,556	0.14	75,032,006	2.6	60.8
100	4,871,550	179.4	28,101,032	4.1	20	183.7	28,779,462	0.02	1,973,210	0.18	19,342,137	0.04	4,552,726	2.4	11.7
200	1,567,450	280.8	14,151,858	3.2	5	284.2	14,324,566	0.02	811,389	0.17	5,873,192	0.03	1,012,593	2.2	3.5
500	25,850	548.9	456,246	4.1	0.1	553.3	459,870	0.04	25,014	0.20	116,684	0.03	15,112	1.7	0.0
Inferred Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	89,422,194	20.4	58,573,468	5.9	529	26.7	76,772,692	0.005	10,022,866	0.11	224,609,701	0.09	181,779,406	3.2	282.0
5.0	89,057,944	20.4	58,552,875	5.9	529	26.8	76,720,493	0.005	10,013,927	0.11	224,029,624	0.09	180,852,266	3.1	280.4
10	**75,728,745**	**23.0**	**55,977,228**	**6.6**	**501**	**30.1**	**73,204,743**	**0.01**	**8,709,675**	**0.13**	**212,159,178**	**0.10**	**168,331,885**	**3.1**	**238.4**
20	45,300,947	31.5	45,832,065	8.4	379	40.4	58,849,468	0.01	6,023,397	0.16	163,778,869	0.13	128,664,071	3.1	142.0
30	28,742,848	38.9	35,996,017	9.8	281	49.4	45,660,864	0.01	4,581,204	0.17	109,942,450	0.15	93,884,665	3.0	87.1
40	16,160,949	49.2	25,562,270	10.7	172	60.6	31,481,558	0.01	3,246,660	0.21	73,096,854	0.19	69,431,240	2.8	44.5
50	8,871,249	64.1	18,273,179	8.6	76	73.3	20,902,381	0.01	2,150,331	0.20	39,182,975	0.27	52,414,915	2.8	24.7
100	1,088,050	141.8	4,960,491	2.8	3	144.8	5,065,204	0.02	503,784	0.07	1,689,592	0.07	1,674,742	3.2	3.4
200	122,200	211.8	832,119	1.3	0.2	213.2	837,594	0.02	45,488	0.05	128,177	0.02	58,625	3.1	0.4
500	0	0.0	0	0.0	0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.0

Table 17-9 Measured, Indicated and Inferred Resources for the Wara Wara Zone

Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Measured Resources															
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	12,083,699	23.5	9,120,239	2.6	32	26.3	10,205,815	0.02	4,383,108	0.03	8,551,994	0.00	1,068,972	5.5	66.9
5.0	11,996,749	23.6	9,112,519	2.6	31	26.4	10,194,373	0.02	4,362,631	0.03	8,520,149	0.00	1,059,179	5.5	66.5
10	**10,551,499**	**26.1**	**8,845,445**	**2.7**	**29**	**29.0**	**9,829,574**	**0.02**	**4,037,605**	**0.03**	**7,928,699**	**0.00**	**899,715**	**5.7**	**59.9**
20	5,898,500	36.7	6,965,125	3.1	18	40.1	7,599,585	0.02	2,689,363	0.04	5,469,677	0.00	448,570	6.2	36.5
30	3,311,150	48.4	5,152,367	3.7	12	52.4	5,577,463	0.02	1,780,543	0.05	3,634,976	0.00	263,227	6.5	21.6
40	1,917,600	60.6	3,737,281	4.4	8	65.3	4,028,660	0.03	1,127,782	0.06	2,439,173	0.00	155,241	6.7	12.9
50	1,212,600	72.2	2,813,387	5.1	6	77.6	3,024,356	0.03	770,798	0.06	1,702,805	0.00	101,825	7.0	8.5
100	166,850	121.0	649,030	7.8	1	129.3	693,725	0.05	170,187	0.09	315,139	0.01	21,285	8.2	1.4
200	9,400	222.1	67,123	11.6	0	234.5	70,881	0.13	27,595	0.05	10,667	0.02	3,466	14.5	0.1
500	0	0.0	0	0.0	0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.00
Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	55,286,096	19.1	33,980,614	2.6	146	21.9	38,999,214	0.02	23,205,507	0.03	31,007,944	0.01	6,952,269	5.4	297.9
5.0	54,449,496	19.4	33,893,260	2.7	145	22.2	38,886,922	0.02	23,023,882	0.03	30,766,063	0.01	6,658,930	5.4	294.1
10	**46,177,497**	**21.7**	**32,251,803**	**2.8**	**131**	**24.8**	**36,764,417**	**0.02**	**20,564,373**	**0.03**	**27,652,170**	**0.01**	**5,515,373**	**5.5**	**256.2**
20	23,669,198	30.8	23,419,817	3.3	79	34.3	26,139,516	0.03	13,422,596	0.03	16,199,117	0.00	2,330,142	6.0	141.9
30	10,673,699	42.5	14,594,879	3.8	40	46.6	15,985,806	0.03	7,727,358	0.03	7,976,635	0.00	1,110,841	6.6	70.4
40	6,157,000	51.3	10,152,582	4.3	26	55.8	11,056,147	0.04	4,765,453	0.03	4,731,031	0.00	580,690	6.4	39.2
50	2,996,250	63.2	6,086,462	4.0	12	67.5	6,501,486	0.03	2,298,070	0.04	2,511,218	0.00	297,772	6.2	18.5
100	61,100	106.7	209,579	8.5	1	115.7	227,353	0.06	82,977	0.05	61,204	0.01	10,783	9.6	0.6
200	0	0.0	0	0.0	0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.00
500	0	0.0	0	0.0	0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.00
Measured + Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
1.0	67,369,795	19.9	43,100,853	2.6	178	22.7	49,205,029	0.02	27,588,615	0.03	39,559,938	0.01	8,021,240	5.4	364.8
5.0	66,446,245	20.1	43,005,779	2.7	177	23.0	49,081,295	0.02	27,386,513	0.03	39,286,212	0.01	7,718,109	5.4	360.6
10	**56,728,996**	**22.5**	**41,097,247**	**2.8**	**160**	**25.5**	**46,593,990**	**0.02**	**24,601,978**	**0.03**	**35,580,870**	**0.01**	**6,415,088**	**5.6**	**316.1**
20	29,567,698	32.0	30,384,942	3.3	98	35.5	33,739,101	0.02	16,111,959	0.03	21,668,794	0.00	2,778,712	6.0	178.4
30	13,984,849	43.9	19,747,246	3.8	53	48.0	21,563,268	0.03	9,507,900	0.04	11,611,611	0.00	1,374,069	6.6	92.1
40	8,074,599	53.5	13,889,862	4.3	35	58.1	15,084,807	0.03	5,893,235	0.04	7,170,204	0.00	735,931	6.5	52.1
50	4,208,850	65.8	8,899,849	4.3	18	70.4	9,525,842	0.03	3,068,869	0.05	4,214,023	0.00	399,597	6.4	26.9
100	227,950	117.1	858,609	8.0	2	125.7	921,077	0.05	253,164	0.07	376,343	0.01	32,068	8.6	2.0
200	9,400	222.1	67,123	11.6	0	234.5	70,881	0.13	27,595	0.05	10,667	0.02	3,466	14.5	0.1
500	0	0.0	0	0.0	0.0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.0
Inferred Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	84,679,895	15.4	41,987,448	2.4	207	18.0	49,092,170	0.017	31,067,237	0.02	34,902,867	0.01	12,929,329	4.9	414.8
5.0	82,651,845	15.7	41,791,216	2.5	204	18.4	48,818,743	0.017	30,134,676	0.02	34,541,071	0.01	11,694,484	4.9	401.5
10	**69,705,696**	**17.6**	**39,367,381**	**2.6**	**179**	**20.3**	**45,520,235**	**0.02**	**26,733,840**	**0.02**	**29,855,831**	**0.00**	**7,154,863**	**4.8**	**332.8**
20	19,747,049	30.4	19,276,974	2.8	56	33.4	21,197,640	0.03	11,359,471	0.03	12,003,206	0.00	1,963,865	4.9	97.1
30	8,065,200	44.2	11,453,406	2.9	23	47.3	12,259,182	0.04	6,485,744	0.02	3,148,935	0.01	994,275	4.4	35.6
40	4,507,300	54.1	7,844,588	3.4	15	57.8	8,375,331	0.04	3,842,230	0.01	1,158,431	0.01	587,411	4.4	19.9
50	2,730,700	63.3	5,561,612	2.7	7	66.2	5,814,632	0.04	2,252,248	0.01	604,317	0.00	293,571	4.3	11.7
100	9,400	95.4	28,827	8.7	0	104.7	31,644	0.07	13,536	0.04	8,647	0.01	1,463	8.7	0.1
200	0	0.0	0	0.0	0.0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.0
500	0	0.0	0	0.0	0.0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.0

Table 17-10 Measured, Indicated and Inferred Resources for the Sucre Zone

Measured Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	11,364,599	29.8	10,907,719	8.4	95	38.8	14,189,949	0.03	8,763,741	0.05	11,630,933	0.029	7,204,515	5.0	56.7
5.0	11,204,799	30.2	10,895,024	8.5	95	39.3	14,168,490	0.04	8,680,056	0.05	11,532,436	0.029	7,135,896	5.0	55.9
10	**9,656,149**	**34.2**	**10,620,046**	**9.5**	**92**	**44.3**	**13,769,229**	**0.04**	**8,073,966**	**0.05**	**10,586,634**	**0.028**	**5,934,260**	**5.0**	**48.3**
20	5,938,450	49.4	9,438,433	12.5	75	62.8	11,999,923	0.05	6,128,037	0.06	7,482,645	0.027	3,552,234	5.0	29.8
30	4,044,350	64.5	8,386,459	15.3	62	80.9	10,514,211	0.05	4,696,809	0.06	5,709,927	0.026	2,304,045	5.0	20.4
40	3,188,950	74.6	7,647,936	17.5	56	93.3	9,565,386	0.05	3,829,112	0.07	4,849,314	0.025	1,730,280	5.1	16.2
50	2,624,950	82.8	6,988,782	19.6	52	103.8	8,759,957	0.06	3,236,284	0.07	4,253,513	0.024	1,380,064	5.1	13.5
100	1,005,800	121.9	3,942,743	33.4	34	157.6	5,097,070	0.06	1,434,076	0.10	2,322,983	0.02	404,176	5.1	5.2
200	162,150	195.1	1,016,985	88.1	14	289.3	1,508,233	0.10	352,304	0.19	695,976	0.02	56,276	5.9	1.0
500	7,050	387.4	87,828	116.6	0.8	512.1	116,079	0.16	24,384	0.20	31,820	0.02	3,508	5.9	0.04
Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	76,950,745	25.1	62,045,978	5.5	422	30.9	76,538,702	0.05	76,993,297	0.04	68,919,947	0.02	39,759,504	5.2	401.4
5.0	76,403,196	25.2	62,005,539	5.5	421	31.1	76,474,886	0.05	76,717,355	0.04	68,687,009	0.02	39,513,154	5.2	399.0
10	**67,054,896**	**28.0**	**60,387,989**	**6.0**	**399**	**34.4**	**74,102,527**	**0.05**	**72,386,267**	**0.04**	**62,204,138**	**0.02**	**34,735,943**	**5.2**	**349.2**
20	37,630,548	41.2	49,826,516	8.1	305	49.9	60,320,598	0.06	51,473,907	0.04	36,253,783	0.03	22,305,639	5.0	189.0
30	22,364,949	56.2	40,442,133	10.5	235	67.5	48,511,855	0.07	36,681,461	0.05	23,699,773	0.03	14,938,992	5.0	111.1
40	15,495,899	68.7	34,207,480	12.3	191	81.8	40,764,317	0.09	29,313,020	0.05	18,083,864	0.03	11,058,925	5.0	77.6
50	12,295,199	76.9	30,418,417	13.6	167	91.5	36,171,065	0.09	24,841,120	0.06	15,216,365	0.03	9,061,495	5.0	61.0
100	3,113,750	120.5	12,065,029	28.1	88	150.6	15,075,071	0.14	9,865,184	0.08	5,580,729	0.05	3,187,236	5.0	15.7
200	531,100	180.8	3,087,320	79.8	42	266.1	4,544,524	0.14	1,656,998	0.16	1,906,834	0.02	254,153	5.8	3.1
500	0	0.0	0	0.0	0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.0
Measured + Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
1.0	88,315,345	25.7	72,953,697	5.9	517	31.9	90,728,651	0.04	85,757,038	0.04	80,550,881	0.02	46,964,019	5.2	458.1
5.0	87,607,995	25.9	72,900,563	5.9	516	32.2	90,643,376	0.04	85,397,410	0.04	80,219,445	0.02	46,649,049	5.2	454.9
10	**76,711,045**	**28.8**	**71,008,035**	**6.4**	**491**	**35.6**	**87,871,756**	**0.05**	**80,460,233**	**0.04**	**72,790,772**	**0.02**	**40,670,202**	**5.2**	**397.6**
20	43,568,997	42.3	59,264,949	8.7	380	51.6	72,320,521	0.06	57,601,944	0.05	43,736,428	0.03	25,857,873	5.0	218.8
30	26,409,298	57.5	48,828,591	11.2	297	69.5	59,026,066	0.07	41,378,270	0.05	29,409,700	0.03	17,243,037	5.0	131.5
40	18,684,849	69.7	41,855,417	13.2	247	83.8	50,329,702	0.08	33,142,131	0.06	22,933,178	0.03	12,789,205	5.0	93.8
50	14,920,149	78.0	37,407,199	14.7	219	93.7	44,931,021	0.09	28,077,405	0.06	19,469,878	0.03	10,441,559	5.0	74.5
100	4,119,550	120.8	16,007,772	29.4	121	152.3	20,172,142	0.12	11,299,260	0.09	7,903,712	0.04	3,591,412	5.1	20.9
200	693,250	184.1	4,104,306	81.8	57	271.5	6,052,757	0.13	2,009,302	0.17	2,602,809	0.02	310,429	5.9	4.1
500	7,050	387.4	87,828	116.6	0.8	512.1	116,079	0.16	24,384	0.20	31,820	0.02	3,508	5.9	0.04
Inferred Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	43,148,347	19.5	27,033,561	2.8	122	22.5	31,238,304	0.048	45,535,555	0.03	29,464,709	0.01	12,567,454	5.2	225.0
5.0	42,558,497	19.7	26,966,452	2.9	122	22.8	31,150,700	0.048	45,427,253	0.03	29,234,805	0.01	12,549,153	5.2	221.8
10	**34,429,848**	**22.9**	**25,370,596**	**3.2**	**110**	**26.3**	**29,134,597**	**0.05**	**41,117,709**	**0.03**	**25,140,170**	**0.01**	**10,933,524**	**5.2**	**180.3**
20	14,443,099	37.6	17,476,414	4.6	67	42.6	19,775,763	0.08	25,555,838	0.03	11,027,459	0.02	7,770,407	4.6	66.9
30	8,170,949	50.9	13,382,051	5.2	43	56.5	14,850,086	0.10	18,318,516	0.04	6,610,123	0.03	5,046,264	4.6	37.9
40	5,832,700	59.6	11,173,177	5.5	32	65.4	12,272,878	0.12	14,956,746	0.04	5,030,223	0.03	3,821,772	4.6	26.8
50	4,500,250	65.6	9,498,522	5.6	25	71.6	10,358,159	0.12	11,563,281	0.04	3,918,540	0.03	2,932,077	4.5	20.4
100	380,700	113.1	1,384,984	6.5	2.5	120.1	1,470,529	0.23	1,953,315	0.03	283,388	0.10	811,048	4.4	1.7
200	7,050	230.5	52,252	6.3	0.04	237.2	53,781	0.53	82,324	0.04	6,901	0.01	1,654	5.8	0.04
500	0	0.0	0	0.0	0	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.0

Table 17-11 Total Measured, Indicated and Inferred Resources for the Malku Khota Project

Measured Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	35,170,097	30.1	34,044,579	5.6	197	36.1	40,813,830	0.02	15,053,409	0.07	51,456,325	0.02	18,753,645	4.5	157.1
5.0	34,805,847	30.4	34,015,151	5.6	196	36.4	40,766,053	0.02	14,944,170	0.07	51,120,014	0.02	18,630,521	4.5	155.4
10	**30,989,448**	**33.4**	**33,319,487**	**6.1**	**188**	**39.9**	**39,790,049**	**0.02**	**13,947,823**	**0.07**	**48,665,367**	**0.02**	**16,239,090**	**4.5**	**138.6**
20	20,083,098	45.4	29,323,946	7.6	152	53.5	34,542,225	0.02	10,399,144	0.09	39,174,761	0.02	10,756,954	4.4	88.5
30	13,394,999	58.5	25,188,185	8.9	119	68.0	29,293,861	0.03	7,830,830	0.10	30,075,944	0.02	7,349,221	4.3	57.4
40	9,566,849	70.4	21,643,279	10.3	98	81.3	25,021,830	0.03	6,101,513	0.11	22,847,227	0.03	5,372,371	4.2	40.2
50	7,061,749	82.2	18,668,009	11.4	80	94.4	21,434,435	0.03	4,977,883	0.12	18,348,431	0.03	4,103,403	4.2	29.5
100	1,938,750	140.0	8,729,278	19.7	38	161.1	10,045,381	0.05	2,044,429	0.16	6,693,856	0.02	1,029,577	4.2	8.1
200	354,850	246.3	2,810,732	42.4	15	291.6	3,327,422	0.07	529,783	0.22	1,690,129	0.02	153,746	4.1	1.4
500	23,500	513.1	387,727	38.0	0.9	553.7	418,422	0.08	42,210	0.19	96,466	0.02	10,610	2.9	0.1
Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	256,074,785	24.6	202,527,907	5.3	1,359	30.3	249,254,556	0.02	114,593,749	0.08	426,151,903	0.04	246,852,265	4.2	1,077.8
5.0	253,630,785	24.8	202,303,250	5.3	1,357	30.5	248,930,409	0.02	114,095,783	0.08	424,699,214	0.04	245,705,863	4.2	1,068.3
10	**224,001,987**	**27.3**	**196,960,598**	**5.8**	**1,293**	**33.5**	**241,404,113**	**0.02**	**106,366,881**	**0.08**	**404,649,086**	**0.05**	**230,573,723**	**4.2**	**943.5**
20	135,087,392	37.9	164,718,123	7.4	1,004	45.9	199,223,998	0.03	75,239,236	0.10	306,491,102	0.06	174,785,145	4.1	550.9
30	80,898,745	50.6	131,520,524	9.0	731	60.2	156,659,480	0.03	52,127,085	0.13	225,419,602	0.07	130,892,571	3.9	317.1
40	52,296,897	63.6	106,927,545	10.0	522	74.3	124,879,904	0.03	40,103,361	0.14	166,590,770	0.09	101,479,303	3.8	198.7
50	35,722,348	76.9	88,332,093	10.3	369	88.0	101,032,441	0.04	31,806,168	0.16	125,450,026	0.10	81,769,759	3.7	132.7
100	7,280,300	154.8	36,238,134	14.3	104	170.1	39,827,300	0.07	11,481,205	0.13	20,928,336	0.04	7,146,629	3.6	26.5
200	1,915,250	251.9	15,512,555	24.4	47	278.0	17,120,782	0.05	2,318,503	0.16	6,796,539	0.03	1,172,743	3.3	6.3
500	9,400	517.3	156,348	3.7	0.03	521.2	157,527	0.03	7,188	0.25	52,039	0.04	8,010	2.0	0.02
Measured + Indicated Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
1.0	291,244,882	25.3	236,572,486	5.3	1,556	31.0	290,068,386	0.02	129,647,158	0.07	477,608,228	0.04	265,605,910	4.2	1,234.8
5.0	288,436,632	25.5	236,318,401	5.4	1,553	31.2	289,696,462	0.02	129,039,953	0.07	475,819,228	0.04	264,336,384	4.2	1,223.7
10	**254,991,434**	**28.1**	**230,280,085**	**5.8**	**1,481**	**34.3**	**281,194,163**	**0.02**	**120,314,704**	**0.08**	**453,314,453**	**0.04**	**246,812,812**	**4.2**	**1,082.0**
20	155,170,490	38.9	194,042,069	7.4	1,156	46.9	233,766,224	0.03	85,638,380	0.10	345,665,863	0.05	185,542,099	4.1	639.4
30	94,293,744	51.7	156,708,709	9.0	851	61.3	185,953,341	0.03	59,957,915	0.12	255,495,547	0.07	138,241,791	4.0	374.5
40	61,863,746	64.6	128,570,823	10.0	621	75.4	149,901,735	0.03	46,204,874	0.14	189,437,997	0.08	106,851,674	3.9	238.8
50	42,784,097	77.8	107,000,102	10.5	450	89.0	122,466,875	0.04	36,784,050	0.15	143,798,457	0.09	85,873,162	3.8	162.3
100	9,219,049	151.7	44,967,412	15.5	143	168.2	49,872,681	0.07	13,525,633	0.14	27,622,192	0.04	8,176,206	3.7	34.5
200	2,270,100	251.0	18,323,287	27.2	62	280.1	20,448,204	0.06	2,848,286	0.17	8,486,669	0.03	1,326,489	3.4	7.7
500	32,900	514.3	544,074	28.2	0.9	544.4	575,949	0.07	49,398	0.20	148,505	0.03	18,620	2.6	0.1
Inferred Resources															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	948,307,233	6.7	202,889,390	2.2	2,046	9.0	273,201,726	0.01	251,656,093	0.04	796,069,528	0.04	808,750,473	4.5	4,304.0
5.0	496,707,052	11.5	182,903,941	2.9	1,425	14.5	231,874,050	0.02	177,997,208	0.05	565,579,851	0.04	427,735,435	4.4	2,198.5
10	**230,013,794**	**18.9**	**140,027,216**	**4.1**	**935**	**23.3**	**172,181,594**	**0.02**	**102,081,329**	**0.07**	**362,157,607**	**0.05**	**246,150,339**	**4.4**	**1,001.1**
20	86,622,802	31.8	88,433,724	6.2	534	38.3	106,801,550	0.03	49,297,443	0.11	204,097,385	0.08	153,269,890	3.9	336.2
30	48,215,170	41.5	64,372,619	7.4	358	49.5	76,675,385	0.03	33,153,008	0.12	123,767,688	0.10	101,717,778	3.6	174.5
40	26,930,177	52.2	45,215,137	8.2	222	61.0	52,833,686	0.04	22,935,116	0.13	79,695,329	0.13	74,677,158	3.5	93.1
50	16,251,909	64.3	33,626,176	6.7	110	71.6	37,391,110	0.05	16,449,502	0.12	43,827,095	0.16	55,810,006	3.5	57.5
100	1,478,150	134.1	6,374,302	3.8	6	138.2	6,567,377	0.08	2,470,635	0.06	1,981,627	0.08	2,487,252	3.5	5.2
200	129,250	212.8	884,371	1.6	0.2	214.5	891,375	0.04	127,812	0.05	135,078	0.02	60,279	3.2	0.4
500	0	0.0	0	0.0	0.00	0.0	0	0.00	0	0.00	0	0.00	0	0.0	0.00

Table 17-12 Inferred Resource for the low grade mineralized halo.

Inferred Resource															
Ag_Eq (g/t)		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
Cut-off	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
1.0	731,056,797	3.2	75,294,912	1.6	1,187	4.9	116,098,559	0.01	165,030,435	0.03	507,092,252	0.04	601,474,284	4.6	3,382.2
5.0	282,438,766	6.1	55,593,397	2.0	570	8.3	75,184,114	0.01	92,421,352	0.04	277,774,351	0.04	222,639,532	4.6	1,294.7
10	**50,149,506**	**12.0**	**19,312,012**	**2.9**	**146**	**15.1**	**24,322,018**	**0.02**	**25,520,105**	**0.09**	**95,002,428**	**0.05**	**59,730,067**	**5.0**	**249.6**
20	7,131,708	25.5	5,848,271	4.6	33	30.4	6,978,680	0.04	6,358,737	0.11	17,287,852	0.09	14,871,547	4.2	30.1
30	3,236,173	34.0	3,541,145	3.3	11	37.5	3,905,254	0.05	3,767,545	0.06	4,066,180	0.03	1,792,574	4.3	13.9
40	429,228	46.0	635,102	4.7	2.0	51.0	703,919	0.09	889,478	0.04	409,821	0.09	836,735	4.4	1.9
50	149,710	60.8	292,863	4.5	0.7	65.6	315,938	0.15	483,642	0.04	121,263	0.05	169,443	4.2	0.6

Table 17-13 Measured, Indicated and Inferred Resource for the Limosna, Wara Wara and Sucre Zones, low grade mineralized halo, and the total Malku Khota Project at a 10 g/t Ag_Eq grade.

Resource		Ag (g/t)		In (g/t)		Ag_Eq (g/t)		CU (%)		Pb (%)		Zn (%)		Ga (g/t)	
	Tonnes	Grade	Ozs	Grade	Tonnes	Grade	Ozs	Grade	Lbs	Grade	Lbs	Grade	Lbs	Grade	Tonnes
Limosna Zone															
Measured	10,781,799	40.0	13,853,996	6.3	68	46.7	16,191,247	0.01	1,836,252	0.13	30,150,033	0.04	9,405,115	2.8	30
Indicated	110,769,594	29.3	104,320,806	6.9	763	36.7	130,537,169	0.01	13,416,240	0.13	314,792,778	0.08	190,322,407	3.1	338
Measured + Indicated	121,551,393	30.2	118,174,802	6.8	831	37.5	146,728,417	0.01	15,252,493	0.13	344,942,811	0.07	199,727,522	3.0	368
Inferred	75,728,745	23.0	55,977,228	6.6	501	30.1	73,204,743	0.01	8,709,675	0.13	212,159,178	0.10	168,331,885	3.1	238
Wara Wara Zone															
Measured	10,551,499	26.1	8,845,445	2.7	29	29.0	9,829,574	0.02	4,037,605	0.03	7,928,699	0.004	899,715	5.7	60
Indicated	46,177,497	21.7	32,251,803	2.8	131	24.8	36,764,417	0.02	20,564,373	0.03	27,652,170	0.005	5,515,373	5.5	256
Measured + Indicated	56,728,996	22.5	41,097,247	2.8	160	25.5	46,593,990	0.02	24,601,978	0.03	35,580,870	0.005	6,415,088	5.6	316
Inferred	69,705,696	17.6	39,367,381	2.6	179	20.3	45,520,235	0.02	26,733,840	0.02	29,855,831	0.00	7,154,863	4.8	333
Sucre Zone															
Measured	9,656,149	34.2	10,620,046	9.5	92	44.3	13,769,229	0.04	8,073,966	0.05	10,586,634	0.03	5,934,260	5.0	48.3
Indicated	67,054,896	28.0	60,387,989	6.0	399	34.4	74,102,527	0.05	72,386,267	0.04	62,204,138	0.02	34,735,943	5.2	349.2
Measured + Indicated	76,711,045	28.8	71,008,035	6.4	491	35.6	87,871,756	0.05	80,460,233	0.04	72,790,772	0.02	40,670,202	5.2	397.6
Inferred	34,429,848	22.9	25,370,596	3.2	110	26.3	29,134,597	0.05	41,117,709	0.03	25,140,170	0.01	10,933,524	5.2	180.3
Low Grad Mineralized Halo															
Inferred	50,149,506	12.0	19,312,012	2.9	146	15.1	24,322,018	0.02	25,520,105	0.09	95,002,428	0.05	59,730,067	5.0	249.6
Total Malku Khota Project															
Measured	30,989,448	33.4	33,319,487	6.1	188	39.9	39,790,049	0.02	13,947,823	0.07	48,665,367	0.02	16,239,090	4.5	139
Indicated	224,001,987	27.3	196,960,598	5.8	1,293	33.5	241,404,113	0.02	106,366,881	0.07	404,649,086	0.05	230,573,723	4.3	943
Measured + Indicated	254,991,434	28.1	230,280,085	5.8	1,481	34.3	281,194,163	0.02	120,314,704	0.07	453,314,453	0.04	246,812,812	4.3	1,082
Inferred	230,013,794	18.9	140,027,216	4.1	935	23.3	172,181,594	0.02	102,081,329	0.07	362,157,607	0.05	246,150,339	4.3	1,001

17.11 Disclosure

GeoVector does not know of any environmental, permitting, legal, title, taxation, socio-economic, marketing or political issue that could materially affect the Mineral Resource Estimate. In addition GeoVector does not know of any mining, metallurgical, infrastructural or other relevant factors that could materially affect the Mineral Resource estimate.

18 OTHER RELEVANT DATA AND INFORMATION

18.1 Introduction

This section discusses a number of subjects that are relevant to the economic analysis, including:

- Environmental, socio-economic and permitting status.
- Physical characteristics of the project including hydrology, hydro-geology and geotechnical.
- Infrastructure including road and rail access, power, water, and gas supplies and construction capability.
- Mine development and design aspects.
- Capital costs, operating costs and manning requirements.
- Economic analysis.
- Project development plan.

18.2 Environmental, Socio-Economic and Permitting

The environmental and social review process will be carried out in parallel with the feasibility study stages, essentially in three phases:

- Environmental and social baseline data collection to provide at least 12 months of data to be used in the subsequent steps. Baselining work will be undertaken in parallel with the PFS. Social baselining work has already been committed and environmental baselining is planned to start in 2011.
- Preparation of the Environmental and Social Impact Assessment (ESIA) and the related Management Plans (MPs) or Action Plans (APs) that describe the plans to mitigate the impacts. Some preliminary work on the ESIA may start during the latter stage of the PFS but most will be undertaken in parallel with the DFS.
- Project Permit applications and approvals come after completion of the ESIA and Management Plans and no work will be undertaken during the PFS.

For projects that will be debt-funded there are effectively two parallel environmental and review processes that need to be merged:

- One that meets the requirements of the Bolivian government; and
- One that meets the standards applied by the banking community. These can be “IFC” or the “Equator Principles” standards, which are generally more stringent than government requirements.

In order to meet both the Bolivian government and the international banking community requirements, the greater bulk of the baselining work and the work associated with the ESIA and the EMP will be undertaken by a La Paz-base, Bolivian consultant, working under the guidance and direction of an internationally-recognized consultant. Both consulting firms have worked with SASC through the PEA and are fully briefed on the approach and requirements of subsequent phases.

18.3 Physical Characteristics

18.3.1 Hydrology / Hydrogeology

In June 2010, South American Silver Corp (SASC) contracted Artois Consulting Ltd. (Artois) to undertake the hydrological and hydrogeological assessment of the Malku Khota mine project consisting of the following tasks:

- To carry out surface water and groundwater monitoring, and train SASC personnel in hydro(geo)logical field measurement techniques.
- To summarize the meteorological data (precipitation, evaporation and temperatures) from nearby stations and estimate storm recurrence intervals, peak discharge and flood volumes.
- To develop an initial conceptual model and water balance to quantify preliminary inflows, discharges and resource availability for the catchment.
- To provide advice with regard to the most likely drainage and water control scenarios for the operational areas.
- To present recommendations with regard to the continuous monitoring of groundwater levels, river flows and springs. The results of this basic assessment are presented in this document.

A preliminary report presents the findings based on a 10-day field visit and the analysis of existing meteorological and environmental data. The following observations are made in the report:

- The mine project area is located at a high altitude (3800-4600 mamsl) within the Andean Cordillera where most of the rain occurs during the summer wet season (November-March). At this altitude, the mean annual precipitation is estimated to be in the order of 922 mm/year.
- The proposed mining area is located in the upper most part of the watershed, from which 6 small sub-catchments originate. Based on preliminary estimates, monthly river flows in each of the sub-catchments range between 11 and 1801 L/s for the dry and wet season respectively.
- The groundwater flow in the region is controlled by the karstic limestones and, to a lesser extent, by the porous sandstones. The low permeability moraine deposits and clayey silt valley infill sediments separate the remnant glacial lagoons from the underlying groundwater table. The groundwater level is expected to occur at an elevation of approximately 4100 mamsl (or 100 m below surface) near the proposed open pit areas. These natural conditions have the following implications for mining:
- The mine's position in the upper part of the watershed, and the occurrence of a prolonged dry season (April-October), simplify the site water management as relatively small volumes are generated and require diversion. In addition, conventional pit drainage methods can be implemented progressively as the pit deepens below the groundwater level. The dewatering system will probably consist of in-pit sumps, groundwater pumping wells and local sub-horizontal drain holes.
- During the annual wet season (November-March), retention dams within the U-shaped valleys can provide storage of excess run-off and intercept potential mine contact water. This could form the main water supply for the mine operation if the appropriate environmental and legal permits are obtained.
- After closure, it is likely that pit lakes will form in the upper part of the catchment. The water retention dams can be used for community development projects (such as future

irrigation projects or small-scale hydro-electric power supply). On-going hydrological and hydrogeological monitoring is required to improve the water balance estimate.

Hydrogeological implications for mining and conclusions for the PEA

Based on this preliminary assessment, the implications for mining are four-fold:

- Dewatering and drainage: Once the pit floor deepens below 4100 mamsl, active dewatering will be required using pumping wells and in-pit sumps. Although long-term pumping rates may vary between 8 and 40 L/s, short-term peaks may exceed this rate temporarily. In addition, the low permeability shales and silicified layers interbedded within the sandstone units may prevent the gradual drainage and maintain elevated pore pressures behind the pit wall. The drilling of sub-horizontal drain holes may be required to relieve pressures.
- Environmental impacts of contact water infiltration: The infiltration rate of contact water (e.g. below the waste rock dumps) may be relatively low as the proposed sites are partially underlain by low permeability moraines and clayey silts. Nevertheless, when placed directly in contact with the fractured bedrock, karstic limestone or faults, significant infiltration may occur. This will need to be managed using interception wells, liners or low permeability barriers.
- Water supply: In the immediate vicinity of the project area (5 km radius), groundwater flow would, theoretically, average around 275 L/s. Under current natural conditions, the groundwater discharges as spring flow and baseflow to the river system. As a result, any future groundwater extraction, whether for drainage or water supply purposes, will likely reduce the surface water flows in the watershed during the dry season. This will need to be managed appropriately.
- Mine closure: Following closure, the groundwater levels will likely rebound and pit lakes will form in the abandoned excavations. Further work is required to determine the filling time and the lake water chemistry. Progressive abandonment may be practical if the hydraulic connectivity between the pits is low.

18.3.2 Geotechnical Characteristics

The open pit scenarios examined for this assessment used a constant pit wall slope of 43 degrees; however, a steeper pit wall, such as 50 degrees, may improve the economics of the project. The justification for steepening pit walls will need to be developed through detailed geotechnical analyses. South American Silver Corporation has begun the process with detailed core logging, RQD measurement and core photography but has not yet initiated formal geotechnical studies.

18.4 Infrastructure

18.4.1 Road Access

The Malku Khota Project site is a relatively remote area of the Altiplano in Bolivia. Currently road access is via an improved gravel road from Oruro, via Bolivar, Sacaca, and Chiro Khasa. The road is a busy bus and truck route and provides reliable access for regular commercial truck service for people and supplies along this route. It is nominally 10 m wide and with drainage ditches and is maintained in good condition. The distance from Oruro to Malku Khota is 190 km. A survey is needed to confirm that the entire route to site can be traversed by rigs with containers, but it appears that this would be the case. Some road widening and realignment will be required before plant operations begin.

Figure 18-1 Typical view on the road from Oruru

A second access route is via improved dirt roads from Cochabamba, via Sakani. The route from Cochabamba is 130 km. It is on good surfaces to a point near the town of Capinota. Thereafter, for approximately 40 km to the site, the road is un-surfaced and in poor condition with many hairpin bends that would be impassable for large vehicles without major realignments. Surface runoff in the rainy season causes washaways.

The main road from Oruro traverses the SASC property to the west of the exposed mineralized zones. The town of Sakani is located approximately 12 km to the northeast of the project area, and is the location of the CMMK field office and the core storage and cutting facilities with electric power and phone service. The travel time from Oruro to the Malku Khota project area is approximately four hours during the dry season and five hours during the rainy season (December to March). The travel time between the SASC property and Cochabamba is approximately four hours during the dry season.

Oruro lies on the Pan Pacific Highway linking Argentina, Bolivia, Peru and the countries to the North. The closest port to Oruro by road is at Mollendo in Peru. The distance from Mollendo to Oruro is 752 km via La Paz.

18.4.2 Rail Access

The closest rail terminal to the site is at Oruro. Oruro is connected by rail to the Chilean port of Antofagasta via Uyuni. This would be the principal transport route for shipping concentrates. Rail links to Brazil and Argentina are from Santa Cruz. Figure 18-2 below shows the major Bolivian road and rail network.

Figure 18-2 Road and Rail Network

18.4.3 Power

In 1994, Bolivia passed a new electricity law requiring state-owned utility company ENDE (Empresa Nacional de Electricidad) to unbundle power generation, transmission and distribution assets. The law also established the Superintendency of Electricity as the regulatory body for the Bolivian electricity sector. Its mandate included protecting consumer rights; granting and amending concessions and licenses; approving international interconnections and stipulating the quantity of electricity exports and imports; supervising the activities of the National Committee of Load Dispatch, the body responsible for the coordination and administration of transactions on the national grid, the SIN (Sistema Interconectado Nacional); and approving and setting prices and tariffs for the electricity industry.

An estimated 83% of the country's installed generating capacity is connected to Bolivia's national grid, the SIN, while the remaining 17% is classified as "Aislados", or independent of the grid. Most of Bolivia's installed generating capacity is from oil and natural gas.

In 2003, there were nine power-generating companies in Bolivia, which supplied the SIN with electricity. Three companies - EGSA (Empresa de Generación Guaracachi), COBEE (Compañía Boliviana de Energia Eléctrica), and EVH (Empresa Eléctrica Valle Hermoso) - accounted for 58% of electricity generated. Some regions of Bolivia are not connected to the national grid and rely on local generators, as well as on cooperatives and self-producers (mainly mining and manufacturing operations) for electricity. The total capacity of the companies supplying power to the national grid is 1200MW. Consumption in all of Bolivia is near capacity at 1100MW.

For the PEA Update a preliminary study for the power supply to Malku Khota was commissioned from a La Paz based consultant. The range of supply to be investigated was 10MW to 44 MW to cover the potential requirements of the project. The study was completed in February 2011 and has provided information on the following topics:

- Identified the location of power sources close to the project
- Analyzed different alternatives for the supply of power to the project
- Given budget costs to supply power to the Malku Khota site based on current construction costs from a recently completed project. Detailed the legal framework and regulations governing the supply of power in Bolivia and provided information on the regulation of contracts within these laws
- Identified the companies that may be contracted to supply the project with electrical power
- Provided a cost for the power supply based on current contracts

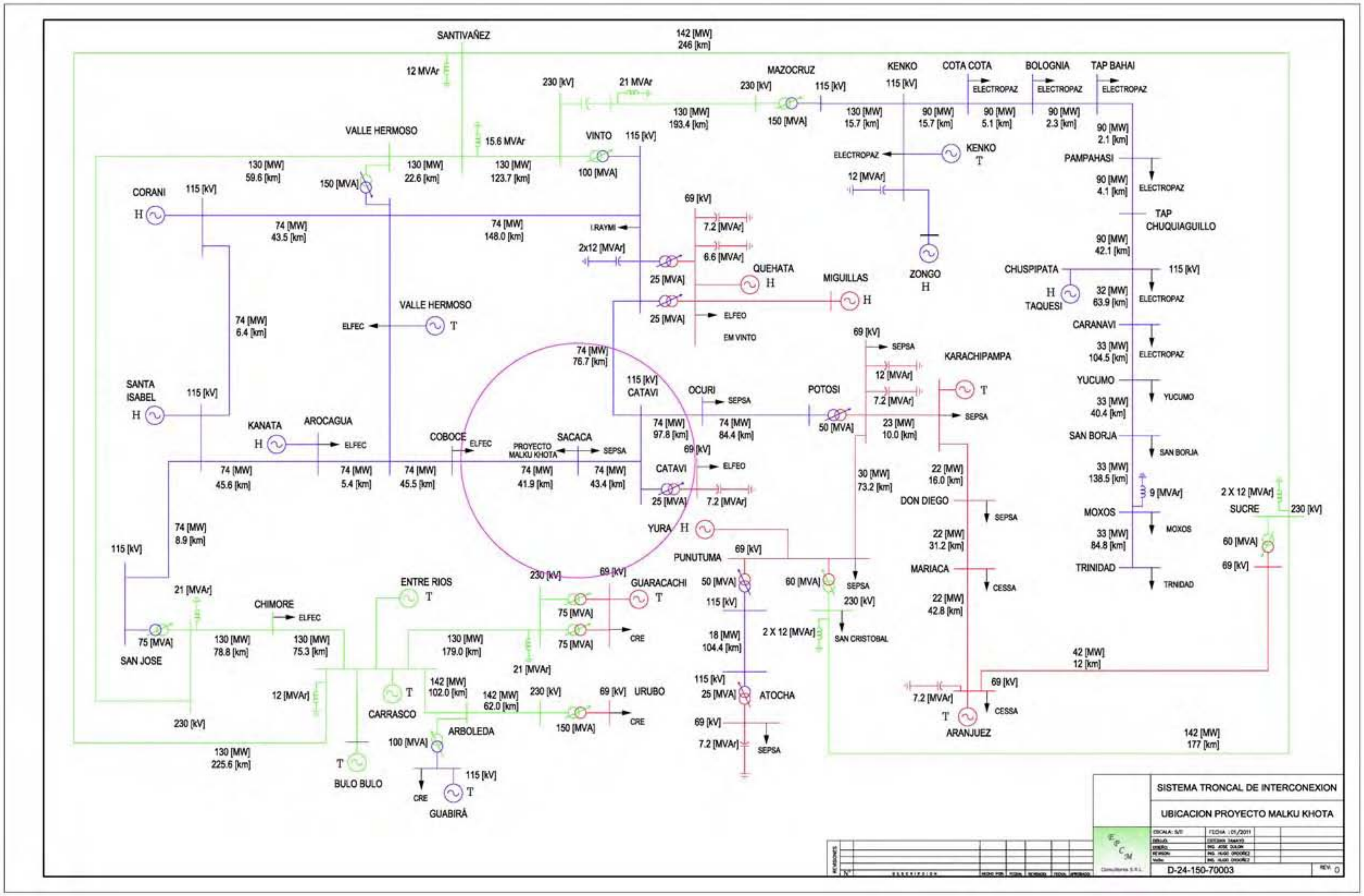
The preliminary study identified the best option for supplying power to the project. This is the construction of a 115Kv line from the substation at the town of Sacaca. The following budget cost has been estimated for a conceptual design of the Power Supply to Malku Khota to meet a maximum demand of 40MW:

- Construction of a new substation at Sacaca
- Construction of 42 km, 115 kv power line from Sacaca to Malku Khota
- Construction of the substation at Malku Khota

The total investment is estimated at $5,8M. The current average energy cost to similar mineral operations in Bolivia is $0,042 /kWh

Figure 18-3 shows the location and route of the power supply to the project.

Figure 18-3 Power Supply Route from Grid to Project

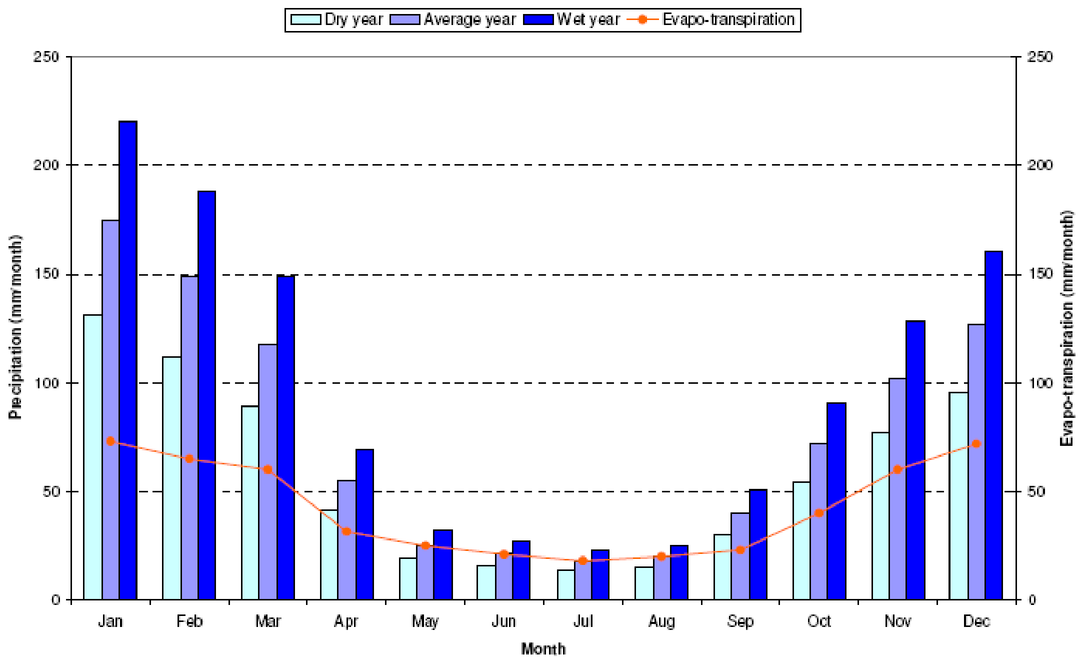


18.4.4 Water

Detailed analyses and studies of current water rights status, availability of water, and the potential impacts to Laguna Wara Wara and Laguna Malku Khota (the high elevation natural lakes) will be required for subsequent planning studies.

Considering options for the plant water supply, the hydrological study conducted by Artois Consulting provided preliminary information about the water regime in the region of the project. The following Figure 18-4 is extracted from the study report:

Figure 18-4 Monthly Precipitation and Evapo-transpiration Estimates



Source: LocClim, 2001 and Senamhi, 2009.

The results of the water balance indicate that there will be a net surplus of water throughout most of the operation, regardless of the climate condition. A more detailed water balance will be done when the final siting of the heap leach pad and residue facility has been decided, using a more comprehensive climate database.

The precipitation and surface run-off from the leach pad, residue facility and undisturbed areas will be combined in a water management plan that will take account of the water demand of the project and the water available from mine de-watering. Any excess water will need to be stored in a storm water pond or treated and released. An allowance for capturing storm water is included in the estimate.

18.4.5 Gas

Bolivia has supplies of very cheap natural gas and the site is within 60 km of an existing gas pipeline. Power generation on site using natural gas is thought to be an attractive option but has not been investigated sufficiently at this stage to include in this PEA.

18.4.6 Infrastructure to Service Project Implementation

Visits were made to vendors and contractors in Bolivia, Chile and Peru - 26th October 2010 through 12 November 2010. The purpose of the visits was threefold:

- Identify local and regional companies that can provide goods and services to the project in the execution phase. The criteria for evaluation is that any such provision of goods or services should be comparable or competitive by cost, schedule and quality with the same product or service supplied by foreign companies;
- Visits were arranged to mining operations that had recently completed their projects. The purpose was to get feedback on their experiences with local companies.
- Initiate preliminary investigations of the supply of utilities to the project

The result of the visit was assessed as very positive in that it provided the following:

- Most (if not all) all of the construction services can be sourced locally from contractors based in Bolivia (mostly located in Santa Cruz) and at significantly lower cost than by construction companies from Peru or other regional countries. There is a strong capability of construction companies that have developed locally to support the gas industry. With the cut back in exploration and development within that industry, there is capacity for the mining projects in development.
- Fabrication of structural steelwork and tanks will be done at vendors workshops in Bolivia. The quality of the products is first class.
- The experience of companies with current operations (San Cristobal; Minera San Bartolome; Empresa Minera Paititi) has provided comparative implementation strategies and outcomes that will be used to devise SASC's implementation strategy that has the benefit of lessons learned.
- Equipment supply was universally reported as being from countries outside Bolivia. There are vendor's agents for many international companies located in the country (Siemens, CAT, Komatsu, Panalpina are all represented).
- A very well developed road transport industry exists in Bolivia.
- Customs clearance is available at the delivery point inside Bolivia for bonded goods. Importation of equipment will not be a problem.

Contractors and vendors visited were prepared to provide budget pricing for rates based contracts that will be used in the PFS. Enquiry documents will be produced at an early stage in the PFS.

18.5 Mine Development

18.5.1 Geologic Model Importation

The geologic model for the Wara Wara, Sucre and Limosna areas was updated with the latest drilling and interpretation as noted in Section 17 of this report. The geologic model was created using the Gemcom© software package. For mine design work MineSight© software with its bundled Lerch-Grossman routine was used but required a transfer of the model information. The bounds for the geologic model are shown in Table 18-1.

Table 18-1 Model Boundaries

Model Type	X	Y	Z
Geologic Model			
Minimum	794,005	7,989,705	2,955
Maximum	796,455	7,992,905	4,455
Mining Model			
Minimum	794,005	7,989,405	2,955
Maximum	796,705	7,992,905	4,455

The mining model boundaries were expanded by 300 m on the south side and 250 m on the east side to accommodate potential open pit shells. An ASCII file export of the geologic model was created in Gemcom and imported in MineSight. The file contained:

- X, Y, Z coordinates for each block center
- resource classification (1 = Measured, 2 = Indicated, 3 = Inferred)
- specific gravity
- block percent
- Silver (g/t)
- Indium (g/t)
- Silver Equivalent (g/t)
- Copper (ppm)
- Lead (ppm)
- Zinc (ppm)
- Gallium (g/t)
- Gold (g/t)
- Antimony (ppm)

Prior to importation of the ASCII file, the mining model was coded with a default specific gravity to populate the blocks that would be outside of the geologic model. This assumes that no additional mineralization was present in the expanded area or is waste for this study. With the information in the model, several calculations were completed for ease of use later in the mine design process. The copper, lead and zinc were converted to percentages. Also, the three main mineralization areas as well as the low grade shell were coded with a numerical value to allow their tonnes and grade to be reported separately should that be required.

18.5.2 Mine Deposit Evaluation

Using the mining model, a series of 27 Lerch Grossman pit shells were generated to assist in understanding the deposit and the impact various parameters had on the available resource tonnage for processing. The variables included:

- Net Metal Prices
- Process Recoveries
- Mine Operating Cost
- Process Operating Cost
- General and Administrative Cost
- Wall Slopes

Initially the parameters from the PAH Preliminary Economic Assessment were used to determine the changes the latest drilling and geologic interpretation had overall. In all instances, a production rate of 40,000 tonnes per day of leach pad feed material was considered.

That initial analysis coupled with updated input costs for mining, processing and G&A were applied to create the ultimate pit shell. The parameters for the development of this ultimate pit shell and used in the pit design process were as shown in Table 18-2.

Table 18-2 Pit Shell Parameters

Input Parameter	Units	Value
Metal Prices (net after deductions)		
Silver	$/oz	14.70
Indium	$/kg	408.00
Lead	$/lb	0.74
Zinc	$/lb	0.74
Copper	$/lb	2.45
Gallium	$/kg	408.00
Recoveries		
Silver	%	70
Indium	%	77
Lead	%	49
Zinc	%	59
Copper	%	81
Gallium	%	26
Operating Cost		
Mining	$/t total material	1.21
Processing	$/t feed material	6.43
General and Administrative	$/t feed material	0.80
Wall Slope Angle	Overall degrees	43

The mining cost used to develop the pit shell was based on estimates arising from earlier pit design work using approximated average haulage profiles. Processing and G&A costs used the values developed for the 40,000tpd acid-chloride heap leach case as described in Sections 18.7.2 and 18.7.3 respectively.

18.5.3 Pit Phase Design

To design the various pits, proper phasing needed to be determined that would allow for early higher grade material release with minimal stripping requirements. To determine these higher value areas, the Base Case metal prices (refer Section 18.9.2) were factored down, representing lower metal price regimes. All the metal prices were factored equally rather than simply adjusting one metal such as silver. This was considered to be a reasonable approach as silver represented the majority of the revenue stream but still received contribution from the other metals in the final revenue calculation.

A series of 14 additional pit shells were developed with metal prices ranging from -10% of the Base Case metal prices to -75% of the Base Case prices. Each area (Wara Wara, Sucre, Limosna) was then examined in detail to determine phases which provided sufficient working width and reasonable phase mine life. The result was three phases for each area. The Lerch Grossman pit shells were used for guidance in the detailed design.

No geotechnical study was completed for the PEA study so certain assumptions were made for the wall slopes. These were considered reasonable at this stage of study after observing the current slopes present on the site. The assumptions were:

- Inter-ramp slope angle = 47.5 degrees

Comprised of
  - Bench face angle = 65 degrees
  - Bench height between berms = 20 meters
  - Berm width = 9 meters

- Overall Slope angle = 43.0 degrees
  Determined by
  - Inter-ramp angle of = 47.5 degrees
  - Haul road ramps = 20.4 meters wide
  - Haul road ramp spacing = 125 meters vertically

Using these assumptions, the phases were designed in detail to better approximate the tonnes and grade expected to be mined in each area and phase.

The determination of the ultimate wall using the Lerch-Grossman routine is based on the mining cutoff calculation on a block by block basis. The total revenue considering net metal prices, recoveries and operating costs is determined for each block. If that block supports the mining of waste material above it at the appropriate slope, then it is mined forming the pit slope.

To calculate the tonnes and grade for each phase, a milling cutoff was used as the lower limit. The calculation of the milling cutoff occurred within each block as part of the Lerch-Grossman evaluation as well and includes all the parameters except the mining cost. This is based on the logic that within the ultimate pit, all the material has been calculated to be moved either as waste or feed material for processing. Haulage of the material to the pit crest and dump is considered and is a "sunk" cost. If that material can still generate a positive return by processing it rather than placement in the waste pile then this is considered above the milling cutoff. This has been used to report out the tonnes and grade for the various pit phases.

A simple cutoff calculation for the pits using silver only and the previously mentioned operating costs and recoveries yielded the following for the mining and the milling cutoffs:

1) Mining Cutoff = 25.5 g/t Silver
2) Milling Cutoff = 21.9 g/t Silver

When the tonnes and grade from the phases were reported, they were split into two different bins to assist in maximizing the net present value of the mine schedule. The boundaries were:

1) Low Grade
   = Milling cutoff to Milling Cutoff + 15 g/t Silver
   = 21.9 g/t to 36.9 g/t
2) High Grade
   >Milling cutoff + 15 g/t Silver
   > 36.9 g/t

The ranges were based on the silver grade only. In all cases the material in the high and low grade met or exceeded the milling cutoff regardless of the silver bin grade used to show the tonnes and grade.

In total nine separate phases were designed (3 for each pit area). The insitu leach tonnage and grade have been shown in Table 18-3.

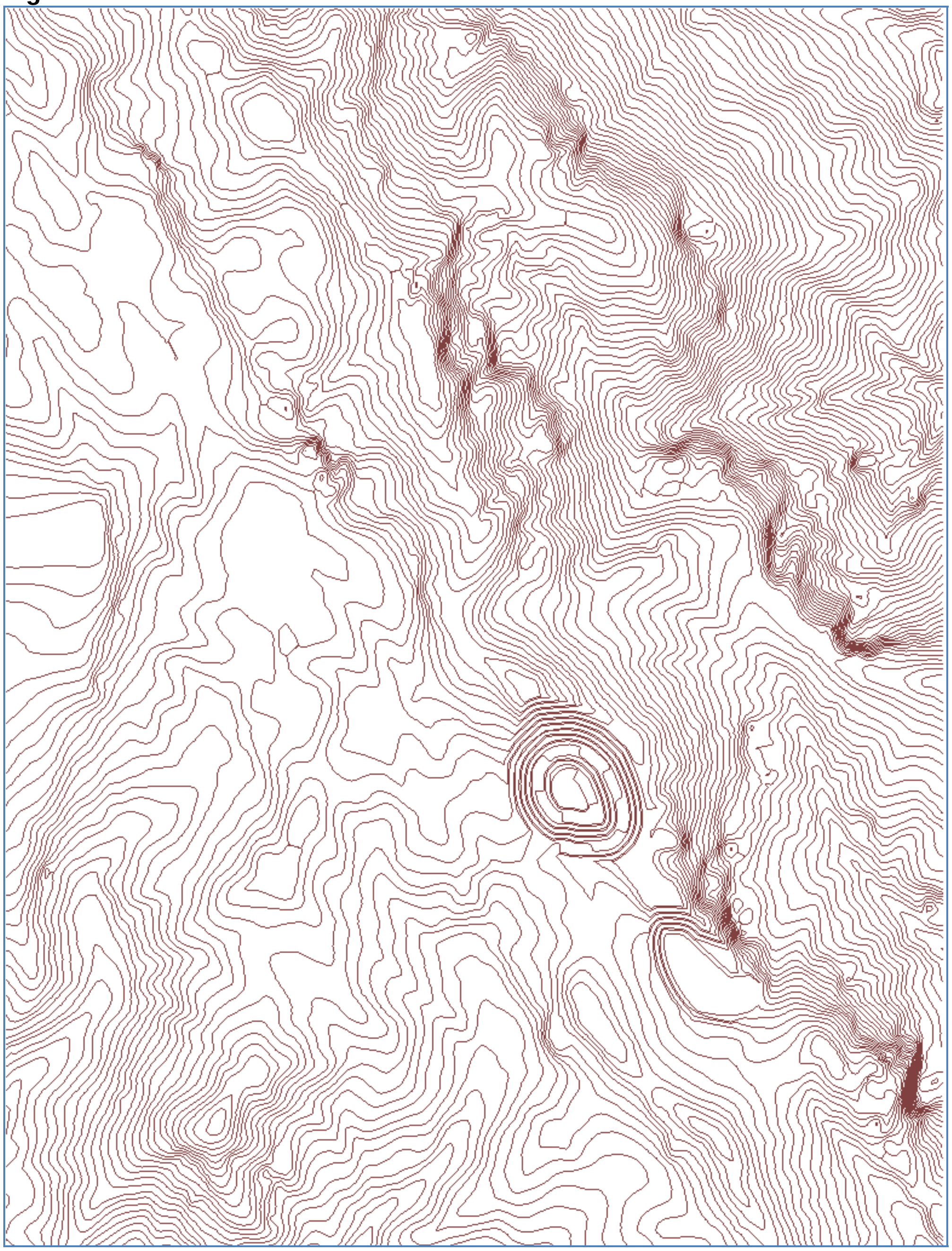
Table 18-3 Pit Phase Insitu Tonnages and Grades

Phase	Leach Feed (Mt)	Ag (g/t)	In (g/t)	Pb (%)	Zn (%)	Cu (%)	Ga (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
Limosna										
Phase 1	28.6	49.2	6.84	0.14	0.04	0.01	2.87	20.4	49.0	0.71
Phase 2	15.7	28.3	6.87	0.11	0.04	0.00	2.27	53.1	68.8	3.39
Phase 3	66.2	28.0	9.76	0.12	0.10	0.00	3.31	214.3	280.5	3.24
Wara Wara										
Phase 1	11.3	37.6	4.04	0.04	0.00	0.02	6.38	8.4	19.7	0.74
Phase 2	8.6	29.1	3.18	0.03	0.01	0.03	6.19	15.5	24.1	1.80
Phase 3	4.3	32.6	2.85	0.02	0.01	0.03	5.69	19.0	23.3	4.47
Sucre										
Phase 1	20.9	44.8	11.25	0.05	0.02	0.04	5.11	17.3	38.2	0.83
Phase 2	23.2	28.3	5.62	0.05	0.02	0.05	5.34	32.1	55.3	1.39
Phase 3	20.0	35.3	4.51	0.04	0.03	0.08	4.53	63.6	83.6	3.18
All Phases	198.8	34.29	7.50	0.09	0.05	0.02	4.06	443.8	642.6	2.23

Starter Pits - Phase 1 (All Areas)

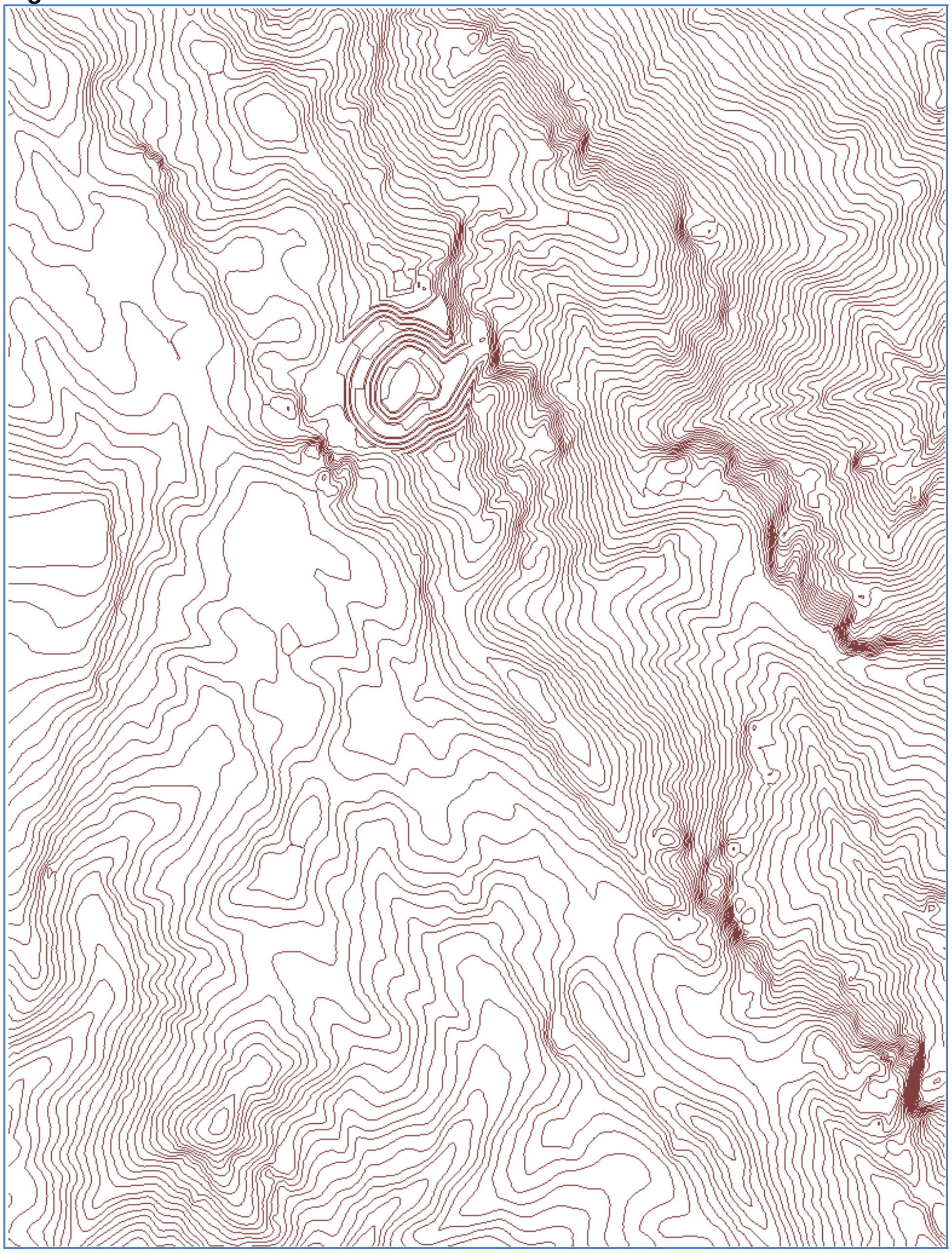
The starter phases for each area targeted the ridge tops where higher grade and lower strip ratio was prevalent. In the case of the Limosna pit, the starter pit is actually two separate pits along the length of the ridge. The southernmost starter pit daylights to the valley on the east side for easy access and waste disposal and requires no ramp system. The northern pit uses the ridge for access and develops a small ramp system below the level of the pit crest. The ridge between the two pits is leveled at the same time as mining to ensure that easy access is maintained for later phases.

Figure 18-5 Limosna Starter Pit



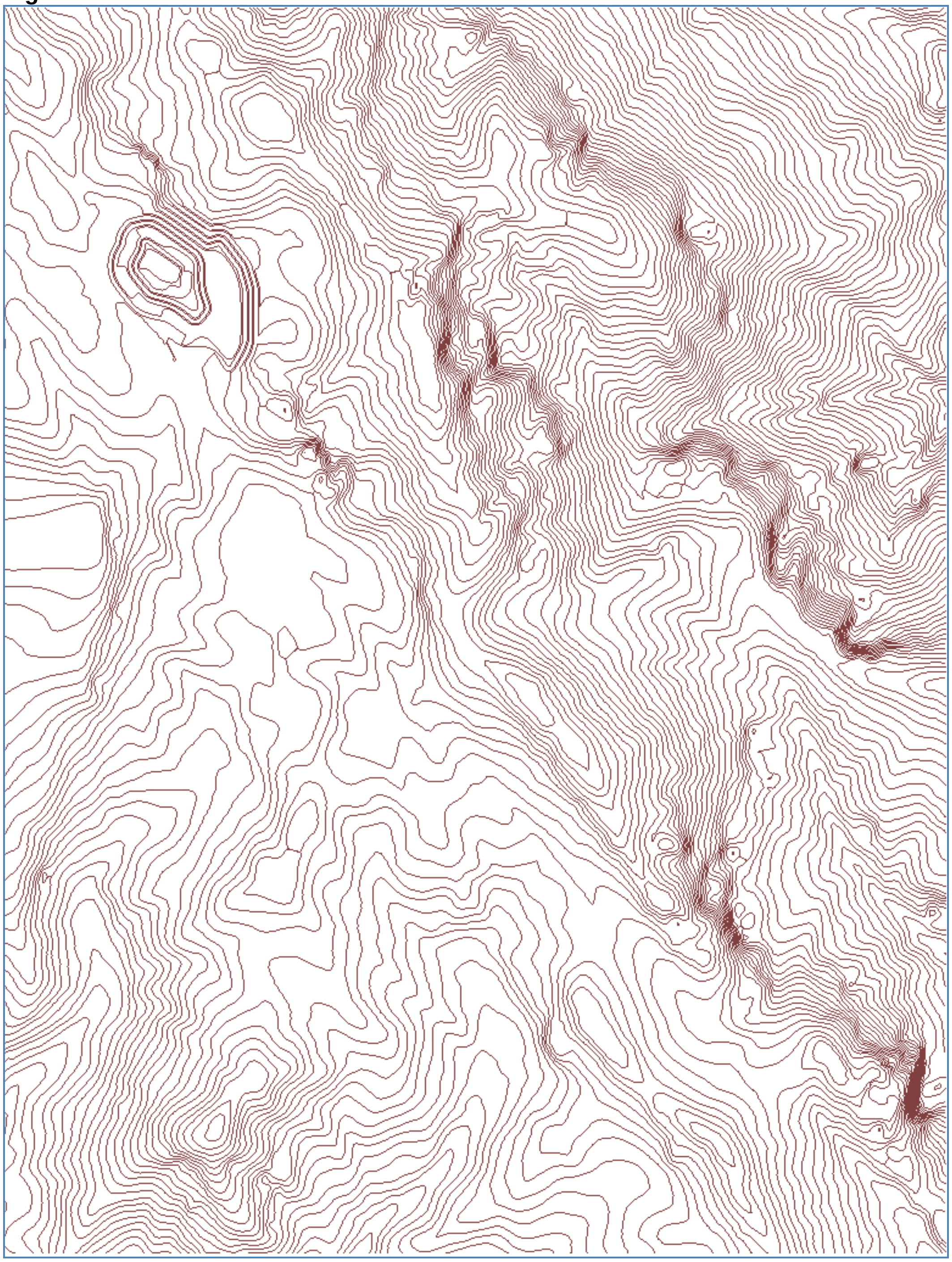
The Sucre starter pit to the north of Limosna is a single pit also starting at the top of the ridge and using the side slopes for access. Waste disposal is to the east into the same valley as the Limosna pit, while access to the west and south is maintained for leach feed haulage. A ramp system is developed for waste and leach feed access.

Figure 18-6 Sucre Starter Pit



The Wara Wara starter pit is located to the west of the Sucre pit and does not have the access to the eastern valley for waste disposal. Waste is hauled to the northwest and a ramp system accessing higher grade material is put in place. Leach feed access is to the southeast in the pit.

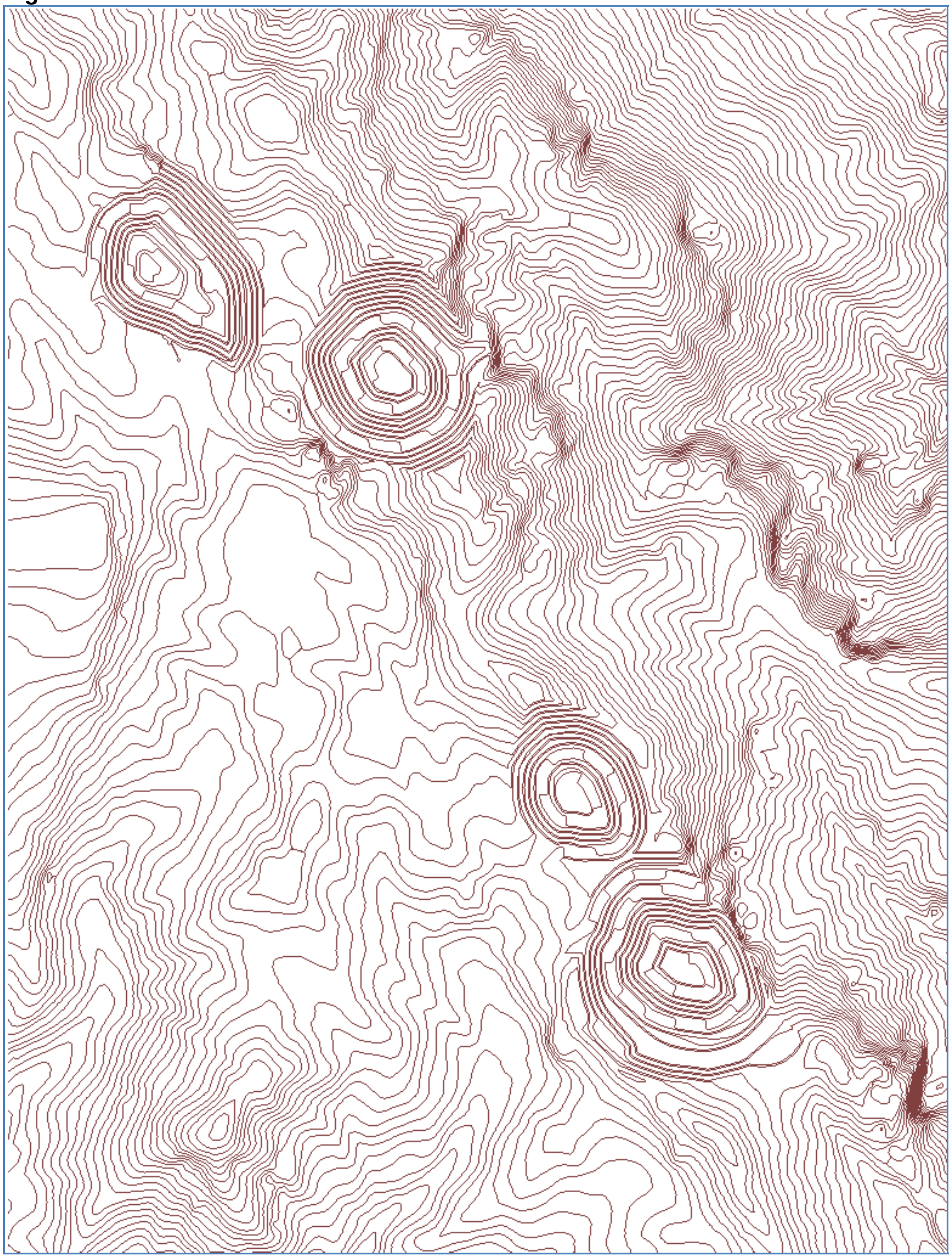
Figure 18-7 Wara Wara Starter Pit



Second Phase (All Areas)

The second phases in each area push the pits along the ridge top and deeper. In the case of Sucre and Limosna, access to the eastern valley is always maintained for waste storage requirements. The second phase of Limosna deepens the southernmost pit only for this phase development. Sucre pushes the wall further back and deepens the pit from the starter pit. In the case of Wara Wara, the pit is elongated to the southeast and also deepens the overall pit.

Figure 18-8 Phase 2 Pits - All Areas



Final Phases (All Areas)

In all cases the pits are deepened with well-established ramp systems. The Limosna pit at all times maintains waste access to the east, although waste material is placed to the southwest of the pit entrance. Limosna ends in two separate areas with a central ridge maintained. The two pits have their own access ramps at the end of the mine life.

The Sucre pit maintains waste dump access on the east side for the majority of the final phase, but then must haul waste to the Wara Wara waste dump. A small shoot of leach feed is mined near the pit entrance.

The Wara Wara final phase pushes the pit deeper. Waste from the pit continues to be placed to the northwest along the ridge.

18.5.4 Open Pit Mine Schedule

The leach pad has been designed to process at a rate of 40,000 tonnes per day in an on/off configuration. The mine provides feed material to the crusher, then hauls and places the crushed material on the pad. When the leaching cycle is complete, the residue will be rehandled and placed in a residue dump to the southwest of the leaching facility.

The concept presented for this PEA study is that initial pre-stripping of the pit areas would be accomplished with the smaller truck size (91t) and loaders which allow for easier access to the ridge top. Once access had been established and the leaching facility was ready for material, the larger 136 t trucks would be matched with hydraulic shovels to provide the feed. The smaller trucks would then be responsible for the crushed material movement and rehandle of the residue. The smaller trucks were less likely to compact the material on the leach pads which may result in blinding of the solutions. They also have comparable production rates and unit costs in the configuration proposed.

The mining schedule is based on the three mining areas and their respective phases. To determine the mining sequence that provided the best NPV to the project, each of the phases was ranked according to its net value where net value is equal to the sum of all metal revenues less operating costs (excluding capital). The net value per phase are shown in Figure 18-9.

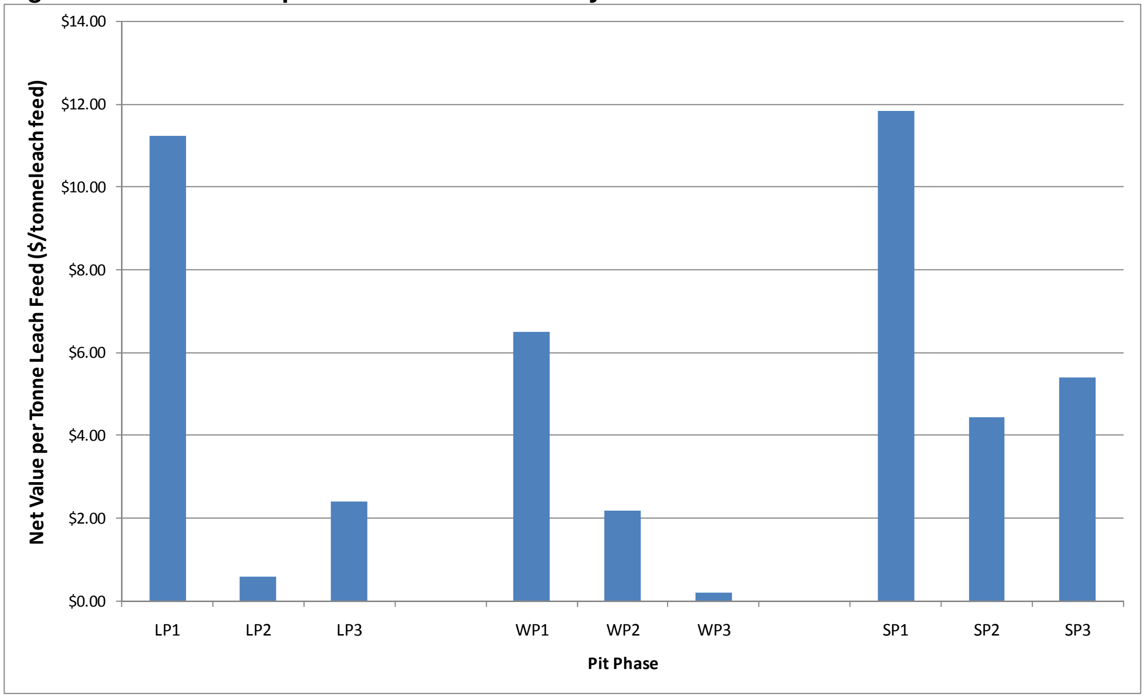
Using this as a guideline, Sucre Phase1 and Limosna Phase1 were the initial pits followed by Wara Wara Phase 1. After those were complete, a focus on production was made in the Sucre area and Limosna pits. The Limosna pit, Phase 2 while of lower value was required to be stripped to open the higher value Phase 3. Limosna Phase 3 also has significant tonnage that the other higher value phases do not contain. So for that reason, the Limosna Phase 2 was accelerated.

The mine schedule was completed using the insitu tonnes and grade from the model. To that, a 2% dilution factor was added to the leach feed. Ore losses were assumed to equal the dilution gains and thus only a drop in the feed grade was carried in the mining schedule for this study.

Three different 40,000 tonne per day schedules were completed and evaluated. The first case was mining material from the milling cutoff with no stockpiling of higher grade feed. The second case was leaving the low grade material behind and mining only the high grade (Milling cutoff +15g/t). The third case examined was mining high grade material initially, stockpiling the lower grade feed and reclaiming that material at the end of the mine life.

The mine life for the first or base case was 14.5 years. The second case was reduced to 13.5 years with the final or third case remaining at 14.5 years.

Figure 18-9 Net Value per Tonne Leach Feed by Phase



The NPVs for the three cases were very similar. This was partially due to the fact that any benefits in the higher initial grades to the leach pad were offset by increased capital to mine at a faster rate to maintain the 40,000 tonnes per day. As the project advances in the next stages, this should be re-examined. For this study, the base case was advanced for use in the final cashflow.

The resulting mine schedule delivered 198.8 Mt of feed to the leach pads at average grades of 33.6 g/t silver, 7.35 g/t indium, 0.09% lead, 0.05% zinc, 0.02% copper and 3.98 g/t gallium. A small stockpile was developed during the pre-stripping of the initial phases and it is incorporated in the feed once the leach plant starts. Waste material totaling 443.8 Mt was placed in storage around the pit areas. The overall strip ratio was 2.23:1.

The leach pad feed by year and grade bin has been shown in Figure 18-10. Feed grades by year have been shown in Figure 18-11. Waste material by phase has been shown in Figure 18-12. As shown in the waste figure, the waste tonnage moved increases dramatically in Year 6 and continues for a period of 6 years at which time it tails off to the end of the mine life. This waste is increase is primarily due to the stripping of Limosna Phase 2 and 3.

Figure 18-10 Leach Pad Feed by Year

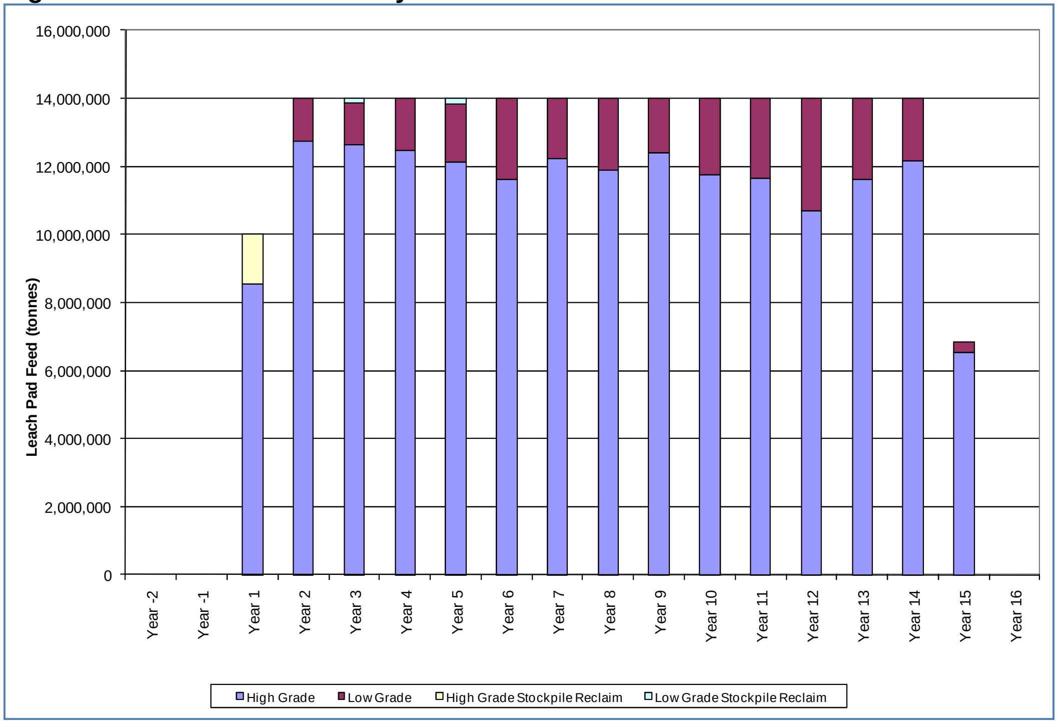


Figure 18-11 Feed Grades by Year

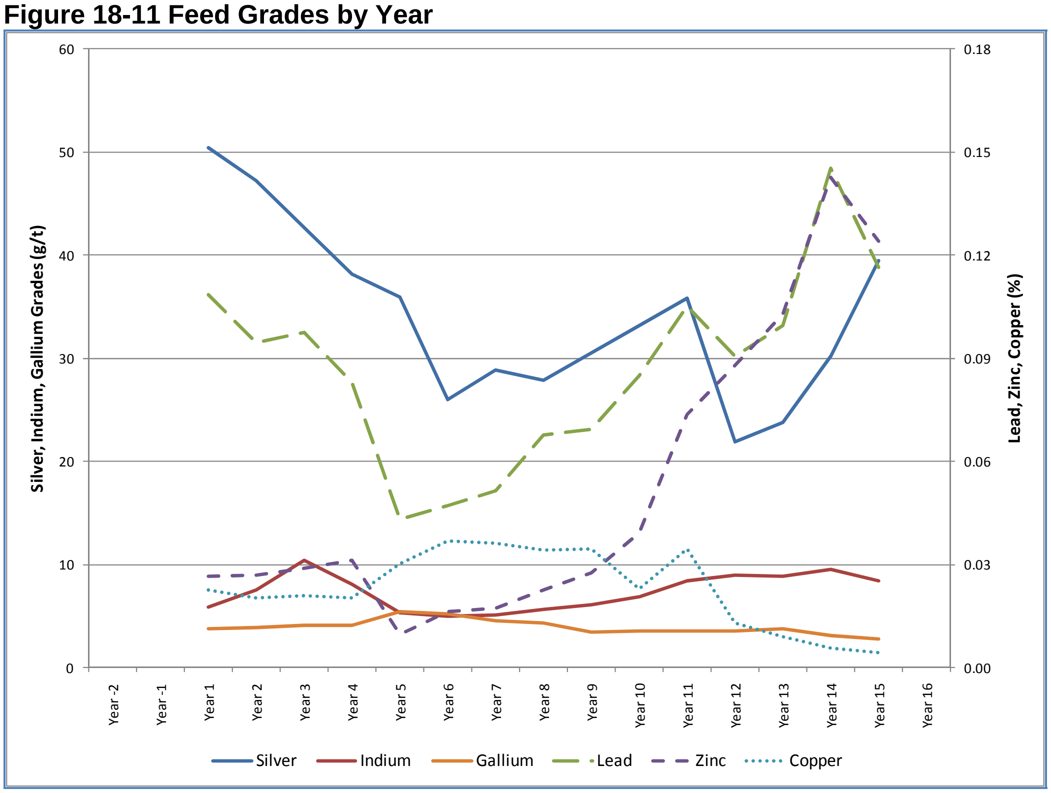
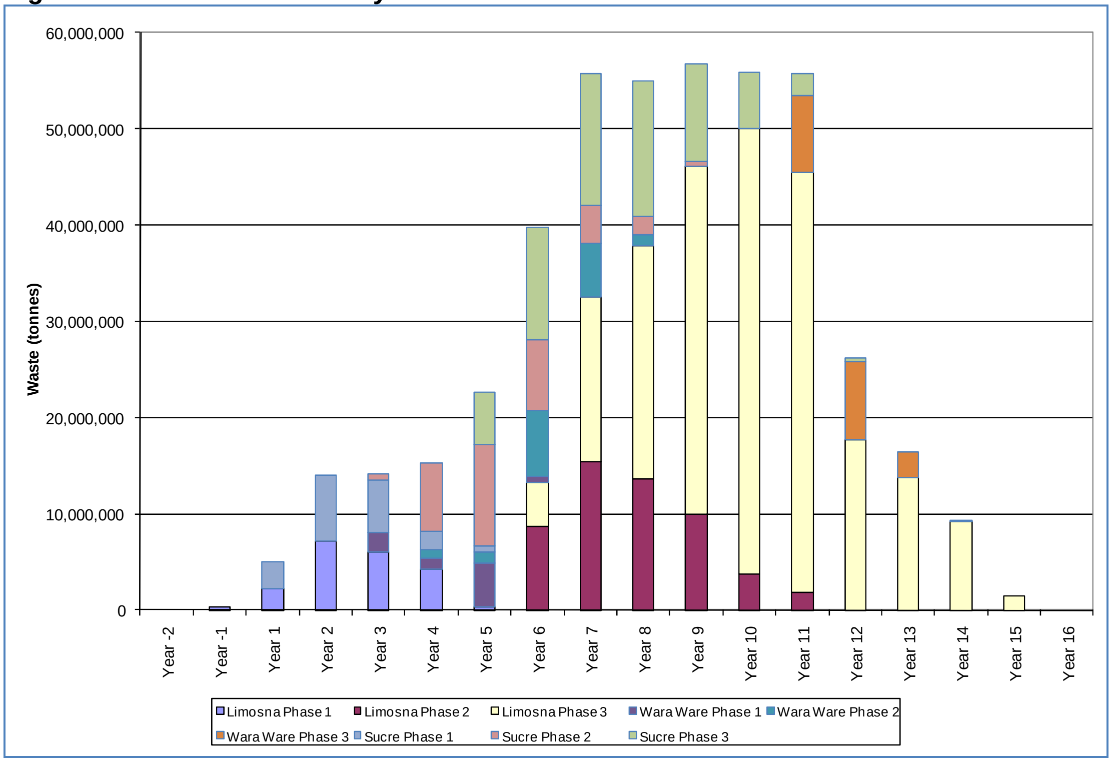


Figure 18-12 Waste Tonnes by Year and Phase



18.5.5 Waste Dump Design

Waste material from all the pit areas has been assumed to be non-acid generating for the purposes of this study. Further study on the exact waste rock properties is recommended as the project advances.

Waste from the three areas is stored on the periphery of the pits to help in the control of operating costs. Limosna will utilize two storage areas:

- East valley
- Southwest by pit entrance

Using the east valley for waste storage greatly assists the haulage costs of the Limosna pit by reducing the vertical lift required to remove the waste from the pit. The pit is able to daylight and provide flat waste hauls. As the initial phases are mined, wrap around dumps are utilized to maximize the storage space for later phases by maintaining the waste material higher in the valley.

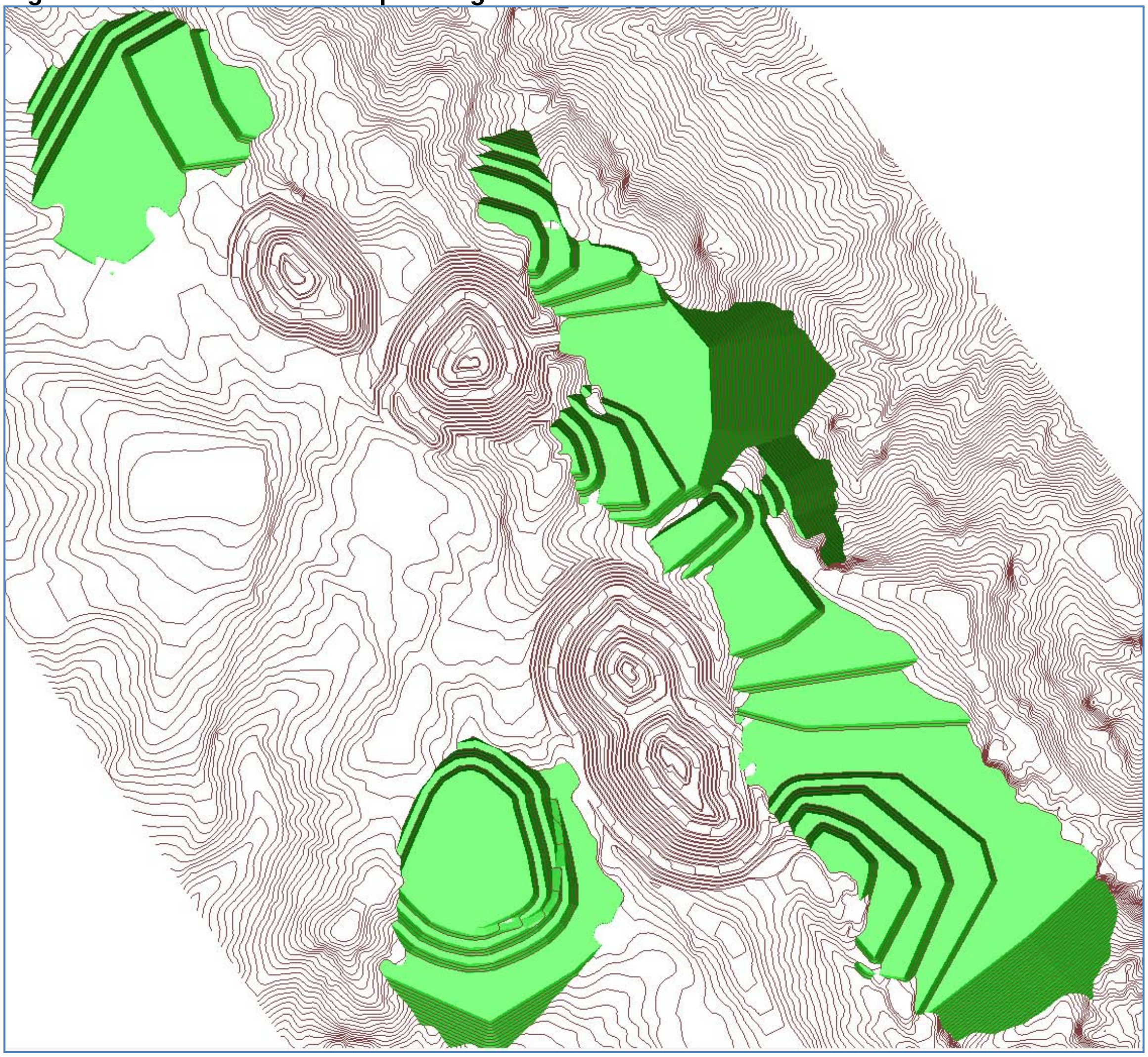
The south west dump for Limosna allows shorter haul material to be stored near the pit entrance rather than a circuitous haul to the east valley.

Sucre uses the east valley in a similar manner as Limosna with wrap around dumps initially then a larger pad for lower levels. A portion of the waste is stored using the Wara Wara dumps.

Wara Wara, not having the benefit of the eastern valley utilizes a side hill dump to the northwest of the pit area, following the ridge. For the upper benches of mining, this means a flat haul is maintained while lower levels will have to lift only a small amount once outside the pit crest.

The final waste dump configuration has been shown in Figure 18-13.

Figure 18-13 Final Waste Dump Configuration



18.6 Capital Costs

18.6.1 Open Pit Mine Capital Requirements

The operational plan described in this study is preliminary in nature and includes Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the operational LOM plans can be realized.

The capital costs for the open pit mine are based around the development of the Wara Wara, Sucre and Limosna open pits with conventional mining equipment. Diesel powered rotary drills with 200 mm bits will be used for production drilling. Haulage trucks with a capacity of 91 and 136 tonnes were matched to 21 cubic meter diesel hydraulic shovels. Support equipment includes track dozers, graders, rubber tired dozers and additional ancillary equipment. Table 18-4 provides detail on the major open pit capital costs by unit and Table 18-5 is by period.

Table 18-4 Major Equipment - Open Pit Capital

Equipment	Capacity	Unit Cost ($)	Operating Life (hr)	Life of Mine Fleet Cost ($)
Production Drill	200 mm	1,400,000	25,000	15,400,000
Front-end Loader	17 m3	4,656,000	35,000	23,280,000
Hydraulic Shovel	21 m3	6,000,000	60,000	24,000,000
Breaker Loader	6.5 m3	903,000	20,000	2,709,000
Haulage Truck	91 tonne	1,868,000	60,000	29,888,000
Haulage Truck	136 tonne	2,792,000	60,000	89,344,000
Tracked Dozer	433 kW	1,267,000	35,000	11,403,000
Tracked Dozer	231 kW	608,000	35,000	1,216,000
Grader	233 kW	791,000	20,000	6,328,000
Rubber-Tired Dozer	350 kW	1,026,000	30,000	4,104,000
Support Equipment	Variable	-	-	14,417,000
Total Mine Capital				**222,089,000**

Table 18-5 Open Pit Capital by Period

Equipment	Total Capital ($)	Pre-Production Capital Year -2 to Year -1 ($)	Production Capital Year 1 ($)	Sustaining Capital Year 2+ ($)
Production Drill	15,400,000	2,800,000	-	12,600,00
Front-end Loader	23,280,000	9,312,000	-	23,280,000
Hydraulic Shovel	24,000,000	-	12,000,000	12,000,000
Breaker Loader	2,709,000	903,000	-	1,806,000
Haulage Truck - 91t	29,888,000	14,944,000	-	14,944,000
Haulage Truck - 136t	89,344,000	-	19,544,000	69,800,000
Tracked Dozer	11,403,000	3,801,000	-	7,602,000
Tracked Dozer	1,216,000	608,000	-	608,000
Grader	6,328,000	1,582,000	-	4,746,000
Rubber-Tired Dozer	4,104,000	1,026,000	-	3,078,000
Support Equipment	14,417,000	5,336,000	392,000	9,081,000
Total Mine Capital	**222,089,000**	**40,312,000**	**31,936,000**	**149,841,000**

A provision for indirect costs was included at 10% of direct costs. An allowance provision of 5% was included to cover estimate omissions. Initial capital (Year -2, Year -1) represents 32% of the life of mine capital requirements.

18.6.2 Processing Plant and Infrastructure

<u>Process Plant and Facilities</u>

Capital cost estimates for processing facilities listed below were prepared for SASC by a third-party estimating consultant.

- Crushing and Screening
- Leaching
- Metals Recovery
- Power Supply

- Site Development
- Air, Water, Fire & Fuel
- Office, Laboratory & Workshops
- Reagents/ Solids Handling

The fundamental building block of the process plant estimate is a sized equipment list for major process equipment, which list is based on process flowsheets developed specifically for the project based on laboratory testwork. Process plant capital costs for crushing, leaching and metal recovery facilities were developed using the following approach.

Equipment pricing is based on the consultant's database and mine and mill equipment costs published by Western Mine Engineering. Equipment installation factors based on a similar project were applied to generate costs for bulk materials and installation labor. Escalation factors were applied to adjust prices to current market conditions.

- A budget estimate for the Acid Recovery Plant was obtained from the company that conducted acid recovery tests during the course of the PEA.
- A scoping-level study and cost estimate was received for an Indium refinery from a consultant that has designed a similar unit built recently in Peru.

Direct Field Costs (DFCs) are included in equipment factors for site earthworks, concrete, structural steel, piping, insulation, painting and coatings, electrical, and instrumentation and controls. Buildings are based on escalated building costs for other similar projects. Costs for supplying power to the site were based on an estimate prepared for SASC by a La Paz based consultant who worked for the electrical power authority in Bolivia.

Costs for administration buildings, including management offices, warehouse, workshops, change rooms and medical facilities have been factored on similar operations.

The estimate includes a provision to selectively upgrade road alignments between Oruru and the site. The scope and extent of the work required will be studied in the PFS.

Labor rates and some Bolivian construction costs were provided by SASC on the basis of an in-country survey undertaken in Bolivia in October-November 2010.

Contractor indirect costs are included in an all-inclusive wage rate applied to the direct labor hours. The "all-in" rate is determined by taking the sum of the burdened direct craft hourly wages plus all the indirect labor, material, and rental/subcontract costs and dividing by the number of direct craft hours. The following accounts are included in the all-in rate:

- Construction service labor
- Temporary construction facilities
- Construction small tools and consumables
- Craft premium pay
- Construction equipment rental
- Construction contractor mobilization/demobilization
- Field staff and clerical
- Contractor Overhead & Profit

Leach Pads, Ponds and Tailings Residues Facilities

Capital estimates for non-process plant earthworks facilities listed below were prepared for SASC by specialist third-party engineering consultant.

- Leach Pads

- Process Component Monitoring System (PCMS)
- ILS, PLS and Stormwater Ponds
- Residue Facility
- Residue Facility Pond

Construction quantities for earthworks, pipework, geosynthetics, and other items considered in the design were estimated commensurate with a conceptual level design.

Quantities were estimated for the mass earthworks required to form the On/Off pad surface, including excavation of top soil and unsuitable materials to reach a competent foundation. For this study, thickness of the topsoil and unsuitable materials to be removed was assumed as 500 mm. A 2 km hauling distance was assumed for stockpiling the topsoil and unsuitable materials. Excavated material were considered either common cut, rippable and solid rock (requiring drill and blast operations) and assumed distribution of each of these materials are listed below:

- Common Cut - 20 percent
- Rock – 80 percent
- Rippable rock - 60 percent
- Drill and Blast Rock – 40 percent

The On/Off pad was laid out such that the excavation volumes of suitable fill materials will provide sufficient quantities to meet the required fill volume. A potential borrow source for the prepared subgrade was assumed 2 km from the On/Off pad limit. It was assumed that protective layer (PL),drainage layer (DL) and drainage aggregate will be obtained from crushing and/or screening operations.

Cost estimates were generated based on the concept of an owner-operated civil construction fleet, which will assume the responsibility of bulk earthworks associated with the project. The balance of items for the cost estimate was assumed to be completed by specialist contractors. The unit rates associated with mass earthworks by construction fleet were provided by SASC. The unit rates for work items by specialist contractors were based on a database that the consultant has established over years of construction of pads on similar projects. In general, the following earthwork related items were considered the responsibility of the construction fleet:

- Mass excavation and /or hauling of common cut, rippable rock, and rock from drill and blast operations
- Placement and compaction of fill

Other Project Costs

In addition to the direct costs described above, the estimate includes the following allowances and provisions:

- Camp and catering costs are included at $15 per direct manhour for the process plant and 7.5% of Total Field Costs for leach pad and residue facilities.
- Construction management costs are included at $11 per direct manhour.
- Overseas freight, inland freight, taxes, and import duties are included at 30% of equipment cost.
- Vendor assistance during construction is included at 2% of equipment cost.
- Spares parts are included at 10% of process equipment cost.
- First fills and commissioning/ pre-commissioning are included at 1% of DFCs and associated Indirects.

- Preliminary engineering, detailed engineering and construction support costs are included as a percentage of DFCs.
- Owner's Costs are included as a percentage of DFCs.
- All other pre-engineering costs including prefeasibility and feasibility studies and environmental review and permitting are treated as sunk costs and are excluded.
- This is a preliminary, scoping level estimate with an accuracy range of ±35-50%. The estimate is based on Q1 2011 costs and is expressed in US dollars. Estimate accuracy allowance, risk contingency and forward escalation are excluded.

Initial Capital Cost Summary

A capital cost estimate summary is shown in Table 18-6.

Table 18-6 Initial Capital Cost Estimate Summary

Cost Center	Estimated Cost (US$K)
Process Plant	
Crushing & Screening	$32,591
Leaching	$9,066
Metal Recovery	$80,353
Power Supply	$5,800
Site Development	$7,529
Air, Water, Fire & Fuel	$3,950
Office, Lab & Workshops	$5,190
Reagents/ Solids Handling	$5,446
Process Plant Direct Costs	$149,924
Leach Pads & Residues	
Leach Pad & PCMS	$39,626
ILS, PLS & Stormwater Ponds	$19,352
Residue Facility	$12,681
Residue Facility Pond	$4,293
Residue Facility Liners, Pipework etc.	$5,375
Leach Pads Direct Costs	$81,329
Total Direct Costs	$231,252
Indirect Costs	
Camp & Catering	$14,125
Field CM, Office, QA/QC	$10,439
Engineering (EPM)	$19,950
Freight, Taxes & Duties	$11,468
Vendor Reps & Commissioning	$3,397
Spares & First Fills	$2,098
Owner's Costs	$31,826
Indirect Costs	$93,304
Total Process Plant & Infrastructure	$324,556

18.6.3 Process Facilities – Sustaining Capital

Sustaining capital for the leach pads and residue dams and ponds were estimated as part of the process of estimating initial capital for the facilities.

- Leach pad overliner replacement costs including associated labor costs average $384,350 per year, starting in Year 1.
- Earthworks, liner systems, groundwater pipework and solution collection pipework costs average $3,108,040 per year starting in Year 1.
- Life-of-Mine Sustaining Capital in the processing facilities amounts to $40.2 million.

18.7 Operating Costs

18.7.1 Open Pit Operating Cost

Mine operating costs were developed from base principles using hourly rates provided by vendors present in the area and a database of costs from other operations worldwide. These hourly rates were based on an owner-operated scenario with the vendor providing direct technical support in maintenance and training.

Key inputs into the mine operating cost estimate fuel, and labor. The diesel fuel cost for this study was estimated at $0.51/L delivered to the site. All the mine equipment was diesel powered for this analysis.

Labor costs were developed from research at other Bolivian mining operations. Burdens were applied at 25% for key Staff and 35% for Hourly personnel. Mine shifts were assumed to be using a 12-hour shift schedule. Mine operations labor requirements for the 40,000 t/d mine have been shown graphically in Figure 18-14. Maintenance staff numbers are reduced because the majority of the work is completed by the equipment vendor.

Open pit mining at Malku Khota utilizes proven technology and equipment. Rock drilling is accomplished with the use of 200 mm rotary blasthole drills. These drills are diesel powered to provide for greater mobility within the pit. Material haulage is handled using trucks in the 91 and 136 tonne class. The smaller trucks are used to initiate mining along the ridge then transition into heap leach trucks for material placement and rehandle. The larger trucks are used when the pit development advances to a point sufficient to accommodate the slightly larger units.

Selective mining is possible with the use of diesel hydraulic shovels but they are also of sufficient size to lower unit operating costs. Track dozers, graders, and rubber-tired dozers round out the major equipment list. Support equipment includes water trucks, a small backhoe with rock hammer, utility loaders, blasting loaders, pickup trucks, small submersible pumps, and light plants. Table 18-7 shows the major mine equipment requirements.

The mine equipment requirements remain constant throughout the mine life balancing increased waste movement with shorter waste haul distances to keep haulage truck numbers lower. The waste hauls remain shorter than comparable mines due to the proximity of deeply incised valleys adjacent to the open pits. The ore haulage is to a central crushing location where the material is then placed on the leach pads for a time, then re-handled and placed in the residue dump. Some of the support equipment is required on the heap leach pad as well as the residue dump and has been accounted for in the mine operating costs.

The large front-end loaders will assist in pioneering material in the initial years with the smaller trucks and act as backup loading unit during the mine life. As the mine matures, the loaders are responsible for 25% of the mill feed and 25% of the waste. The shovels will move the remainder.

Table 18-7 Major Mine Equipment Requirements

Equipment	Capacity	Year -2	Year -1	Year 1 - 15
Production Drill	200 mm	2	2	4
Front-end Loader	17 m3	1	2	2 - 3
Hydraulic Shovel	21 m3	-	-	2 - 4
Breaker Loader	6.5 m3	-	-	1
Haulage Truck	91 tonne	5	3	8
Haulage Truck	136 tonne	-	-	7 - 32
Tracked Dozer	433 kW	3	3	5
Tracked Dozer	231 kW	1	1	1
Grader	233 kW	2	2	3
Rubber-Tired Dozer	350 kW	1	1	3

The small front end loader would be used at the primary crusher tramming material from temporary piles to ensure the primary crusher is properly charged. Additional jobs would include general work around the crusher and the heap leach pad as required. The primary crusher discharges into a hopper where the 91 tonne trucks can be loaded directly. Cleanup around this hopper would also be handled by the small loader.

Mine engineering and general operating costs are included. This covers the mine operations department, both supervision and staff, mine engineering and geology cost functions.
Drilling for the open pits is completed using diesel drills with a 200 mm diameter bit in a rotary configuration. The pattern size used varies for mill feed and waste and is shown in Table 18-8.

Table 18-8 Drill Pattern Specification

Specification	Unit	Mill Feed	Waste
Bench Height	m	10	10
Sub-drill	m	1.9	2.0
Blasthole Diameter	mm	200	200
Pattern Spacing - Staggered	m	7.4	7.7
Pattern Burden - Staggered	m	6.4	6.7
Hole Depth	m	11.9	12.0

The wider pattern spacing for waste was considered to keep the rock coarser for the waste dump saving on explosives costs. The greater sub-drill was included to allow for caving of weaker zones without having to re-drill the hole. Table 18-9 outlines the parameters used for estimating drill productivity.

Table 18-9 Drill Productivity Criteria

Drill Activity	Unit	Mill Feed	Waste
Pure Penetration Rate	m/min	0.54	0.54
Hole Depth	m	11.9	12.0
Drill Time	min	22.04	22.22
Move, Spot and Collar Blasthole	min	3.00	3.00
Level Drill	min	0.25	0.25
Add Steel	min	-	-
Pull Drill Rods	min	0.50	0.50
Total Setup/Breakdown Time	min	3.75	3.75

Total Drill Time per Hole	min	25.80	26.00
Drill Productivity	m/h	27.7	27.7

Explosives costs were determined from a quotation from local vendors and other projects in the Americas. A heavy ANFO product was considered in the costing of the explosives. The powder factors used in the explosive calculation are shown in Table 18-10.

Table 18-10 Design Powder Factors

	Unit	Mill Feed	Waste
Powder Factor	kg/m3	0.61	0.56
Powder Factor	kg/t	0.23	0.22

Loading costs were estimated using the hydraulic shovels as the primary material movers. The front-end loader in the first two years (pre-stripping) would be responsible for phase initiation and the bulk of material mined in that period. The first shovels will start in Year 1. With both shovels functional, the loader will mine 25% of mill feed and 25% of waste on an annual basis. The loading percentage averages and other loading information are shown in Table 18-11 for the 136t trucks.

Table 18-11 Loading Parameters

	Units	Front-End Loader	Hydraulic Shovel
Waste Tonnage Loaded	%	25	75
Mill Feed Tonnage Mined	%	25	75
Bucket Fill Factor	%	90	95
Cycle Time	Sec	35	30
Trucks Present at the Loading Unit	%	80	80
Loading Time	min	3.62	3.20

The trucks present at the loading unit refers to the percentage of time that a truck is available to be loaded. To maximize truck productivity and reduce operating cost, it is more efficient to slightly under-truck the shovel. The single largest operating cost item is the haulage and minimizing this cost by maximizing truck productivity is crucial to lower operating costs. The value of 80% comes from typical standby a shovel encounters due to a lack of trucks.

Haulage profiles were determined for each pit phase for crusher or waste dump. From these profiles, Caterpillar's FPC software was used to determine haulage cycle times. These cycle times were applied to the appropriate yearly tonnage by destination and phase to estimate the haulage costs.

Support equipment costs were determined using either a percentage applied to the truck hours or the loading hours. As indicated earlier, these percentages resulted in the need for six track dozers, three graders and three rubber-tired dozers. Their tasks include cleanup of the shovel face, roads, dumps, and blast patterns as well as activities around the heap leach and its associated residue dump. The graders will maintain the crusher feed and waste haul routes.

The equipment rates applied, less the operating labor, are shown in Table 18-12. All rates include consumables such as fuel, tires, drill steel, bits, as well as associated maintenance costs from the vendor. Fuel consumption is estimated from base principles using the FPC software as a check. Operating labor is calculated separately.

Table 18-12 Major Equipment Hourly Rates

Equipment	Hourly Rate ($/h)
Production Drill	269
Front-end Loader	293
Hydraulic Shovel	516
Breaker Loader	103
Haulage Truck - 91t	120
Haulage Truck - 136t	158
Tracked Dozer	129
Grader	62
Rubber-Tired Dozer	81

The mining cost is calculated by year to take into account changing haulage routes which helps in determining equipment requirements. To the mine operating cost was added the sampling cost. Every blast hole was assumed sampled to help identify heap leach feed boundaries. A sample cost of $40 per sample was applied.

The LOM average cost is shown in Table 18-13. This cost is for the total material moved.

Table 18-13 Open Pit Mine Operating Unit Costs

Open Pit Operating Category	Life of Mine Cost ($/t total material)
General Mine and Engineering	0.04
Drilling	0.10
Blasting	0.15
Loading	0.23
Hauling	0.40
Support	0.14
Sampling	0.02
Total	**1.08**

18.7.2 Processing Costs

Processing costs were estimated as described below and summarized in Table 18-14.

- Power unit costs at $0.042/ kWh are based on a budget estimate provided to SASC by their Bolivian power consultant.
- Reagent consumptions are based on the metallurgical testwork carried out to date.
- Unit costs for reagents and consumables are typically based on costs sourced in Bolivia from existing operations or from Bolivian suppliers.
- Hydrochloric acid will be delivered as a 33% concentrated solution; sodium chloride as crude salt.
- Lime can be utilized in the process as pebble quick lime or limestone. For the purpose of the PEA, it has been assumed that limestone would be locally mined.

- "On-off" loading and unloading of the heap leach pads and progressive construction of the residue dam walls after Year 2 of operation are treated as Sustaining Capital and not included in the processing costs.

Table 18-14 Operating Cost Schedule

Item	Quantity		Rate		Cost/ tonne	
Crushing & Grinding						
Power	3.10	kWh/ t	$0.042	/ kWh	$0.13	
Liners & grinding media	1	lot	$0.17	/ tonne	$0.17	
Conveyor & screen parts	1	lot	$0.50	/ tonne	$0.50	
Total - Crushing & Grinding					$0.80	
Leaching & Refining						
Power	1.00	kWh/ t	$0.042	/ kWh	$0.04	
Maintenance	1	lot	$0.15	/ lot	$0.15	
Acid (HCl)	21.2	kg/ tonne	$70.00	/ tonne	$1.48	
NaCl	0.2	kg/ tonne	$100.00	/ tonne	$0.02	
NaOCl	0.5	kg/ tonne	$240.00	/ tonne	$0.12	
Scrap Iron (Fe)	4.39	kg/ tonne	$330.00	/ tonne	$1.45	
NaOH	0.95	kg/ tonne	$45.00	/ tonne	$0.04	
Limestone (CaCO3)	38	kg/ tonne	$10.00	/ tonne	$0.38	
Na2S	0.81	kg/ tonne	$800.00	/ tonne	$0.65	
Pad piping & materials	1	lot	$0.05	/ tonne	$0.05	
Metal Recovery miscellaneous	1	lot	$0.25	/ tonne	$0.25	
Total - Leaching & Refining					$4.64	
Other Processing Costs						
Solution/ residue handling	1	lot	$0.40	/ tonne	$0.40	
Heap leach handling	1	lot	$0.43	/ tonne	$0.43	
Labor	1	lot	$0.14	/ tonne	$0.14	
Total - Other Processing					$0.97	
Total - Processing Costs					$6.41	/ tonne

Material Handling Costs

Crusher to heap = $0.15 per tonne (includes a hopper at $0.05 per tonne and transport on trucks to heap at $0.10) and heap to residue = $0.46 per tonne (includes loader at $0.20 per ton and trucks to residue facility at $0.26) for a total cost of $0.61 per tonne to be added to the heap process cost for material handling.

18.7.3 General & Administration (G&A)

For the Base Case Acid-chloride Heap Leach operation, G&A costs are included at $0.80 per tonne of leach feed, which, at 14Mt of leach feed per annum, equates to at $11.2 million per annum.

18.8 Manpower

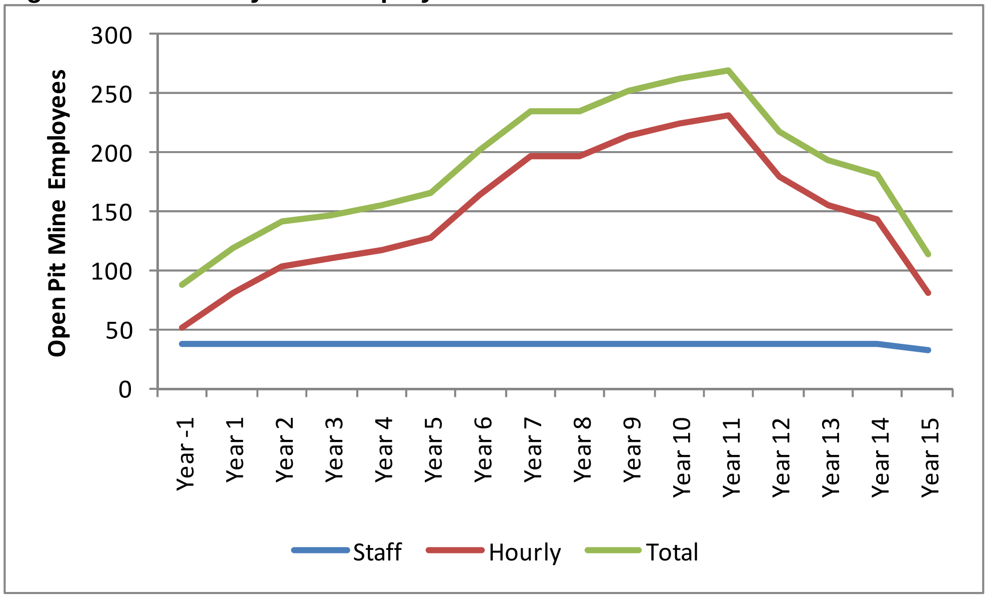
18.8.1 Open Pit Personnel

The open pit responsibilities include mining within the open pits as well as placement and rehandling of the leach pad material. A smaller fleet of equipment with its associated support has been included in the mine operating labor calculation.

The staff employees remain steady at 38 throughout the mine life. This includes mine operations, engineering and geology. A small contingent of employees is included in the maintenance area to monitor the vendor maintenance program and ensure that value is maintained with that program.

The hourly employees vary by year as the waste stripping and haulage requirements change. They vary from a low of 51 in Year -1 to a peak of 270 in Year 11. The year by year employee levels have been shown in Figure 18-14 below.

Figure 18-14 Year-by-Year Employee Numbers



18.8.2 Process Plant Personnel

The plant operating personnel includes the staff and hourly personnel required to manage and operate the acid-chloride heap leach, metal recovery, solution handling and process plant supporting functions. The supporting functions include the laboratory, and management and technical personnel including both operating and maintenance personnel. Plant personnel for the processing facilities at Malku Khota were developed from first principles and operations labor were developed by looking at the equipment to be operated and maintained as well as the area to be covered logistically by one person. Operations staffing is based on the technical aspects of the processing facility for technical staff and the number of persons to be managed for the managerial staff. The following table, Table 18-15, provides the resulting personnel requirements for the Malku Khota on-site process facilities.

Table 18-15 Plant Personnel Requirements

Position	Number
Staff Positions	
Process Manager - Expatriate	1
Chief Metallurgist - Expatriate	1
Process General Foreman	1
Maintenance General Foreman - Expatriate	1
Metallurgists	2
Chemists	1
Shift Foreman	8

Position	Number
Maintenance/Electrical Foreman	4
Chief Assayer	1
Refiner	4
Assayer	16
Sub-total Staff	40
Hourly Positions	
Crushing Operator	12
Metal Recovery Plant Operator	12
Cementation Operator	8
Heap Leach Operator	16
Product Packaging / Shipping	12
Residue Storage Operator	8
Reagent Handling / Helper	16
Maintenance Artisans	32
Sub-total Hourly	116
Total Process Operating Personnel	**156**

18.9 Project Economics

A discounted cash flow model has been developed for the project, the main parameters for which are summarized below.

18.9.1 Mine Production Rates and Grades

Table 18-16 is a summarized extract from the year-by-year mine production plan for the 40,000tpd acid-chloride heap leach Base Case.

Table 18-16 Acid-chloride heap leach case – Mined tonnages and grades

Year	Ore Tonnes	Waste Tonnes	Strip Ratio	Ag (g/t)	In (g/t)	Pb%	Zn%	Cu%	Ga (g/t)
1	10,000,000	4,982,700	0.56	50.46	5.92	0.109	0.027	0.023	3.74
2	14,000,000	13,977,100	1.00	47.27	7.51	0.094	0.027	0.020	3.89
3	14,000,000	14,167,200	1.02	42.65	10.42	0.098	0.029	0.021	4.08
4	14,000,000	15,318,700	1.09	38.10	8.10	0.083	0.031	0.020	4.09
5	14,000,000	22,619,600	1.64	35.94	5.33	0.043	0.010	0.030	5.43
6	14,000,000	39,795,600	2.84	25.97	5.04	0.047	0.016	0.037	5.24
7	14,000,000	55,808,300	3.99	28.86	5.15	0.051	0.017	0.036	4.53
8	14,000,000	55,046,400	3.93	27.81	5.66	0.068	0.022	0.034	4.33
9	14,000,000	56,725,300	4.05	30.49	6.13	0.069	0.027	0.034	3.46
10	14,000,000	55,876,900	3.99	33.16	6.87	0.085	0.039	0.023	3.58
11	14,000,000	55,803,400	3.99	35.82	8.45	0.105	0.074	0.035	3.53
12	14,000,000	26,159,100	1.87	21.94	9.00	0.091	0.088	0.013	3.54
13	14,000,000	16,477,000	1.18	23.73	8.82	0.100	0.103	0.009	3.75
14	14,000,000	9,262,400	0.66	30.16	9.57	0.145	0.143	0.006	3.06
15	6,831,800	1,490,100	0.22	39.45	8.42	0.116	0.124	0.004	2.81

18.9.2 Metal Prices

Three metal price scenarios were used to show the project's sensitivity to varying metal prices using the same pit model (shown in Table 18-17). (At higher metal prices, updated mine models would likely increase total metal production.)

- a Base Case scenario using the approximate 3 year trailing average price for metals;
- a Middle-Case with approximate 1 year trailing average metal prices; and
- a Recent Case which reflects recent monthly average metals prices.

Table 18-17 Metal Price Scenarios

Metal	Base Case	Middle Case	Recent Case
Silver (US$/ ounce)	$18.00	$25.00	$35.00
Indium (US$/ kg)	$500.00	$570.00	$650.00
Lead (US$/ lb)	$0.90	$1.00	$1.20
Zinc (US$/ lb)	$0.90	$1.00	$1.10
Copper (US$/ lb)	$3.00	$3.70	$4.30
Gallium (US$/ kg)	$500.00	$570.00	$730.00

18.9.3 Recoveries

Recoveries for pay metals have been derived for the different ore zones on a weighted average basis, based on mineral resource estimates described in Section 17 and metallurgical testwork results described in Section 16 of this report.

Table 18-18 Recoveries by Metal

Metal	Acid-chloride Heap Leach Recovery
Silver	73.6%
Indium	81.0%
Lead	51.1%
Zinc	62.0%
Copper	84.8%
Gallium	26.9%

18.9.4 Metal Pay Factors, Treatment Charges, Refining Charges (TC/RCs) and Transport

At this early stage in the project there are no offtake agreements in place for the final products. The following pay factors and allowances for TC/RCs and transport have been used in the financial model.

Table 18-19 Pay Factors, TC/RCs & Transport

Metal	Pay Factor	TC/RCs and Transport
Silver	97%	$0.36/ oz
Indium	100%	$0.00/ kg
Lead	90%	$0.14/ lb
Zinc	85%	$0.14/ lb
Copper	95%	$0.24/ lb
Gallium	100%	$0.00/ kg

18.9.5 Gold Credits

Gold credits are included $0.72/ tonne as described in Section 18.

18.9.6 Operating Costs

Operating Costs are based on estimates described in Section 18.7 and are included in the financial model on a per tonne basis using the rates shown below.

Operating Cost Summary

Mining	$1.08	per tonne mined
Processing	$6.41	per tonne of leach material
G&A	$0.80	per tonne of leach material

18.9.7 Initial Capital

Initial Capital Costs were estimated as described in Section 18.6 and are summarized in the financial model as shown below.

Capital Cost Summary

Mine pre-production	$3.8 M
Mining Fleet	$83.1 M
Process Plant & Infrastructure	$149,9 M
Leach Pads and Residue Facilities	$81.3 M
Indirect Costs	$93.3 M
Total Initial Capital	$411.4 M

18.9.8 Working Capital

The financial model bases Working Capital requirements on 90 days between the date of sale (production) and receipt of payment. The balance in the account at the end of mine life is zero.

18.9.9 Sustaining Capital and Closure

Deferred and sustaining capital requirements for the production phase have been estimated on an year-by-year basis for mining and on a rate per tonne basis for processing. Average costs per tonne equate to the following:

Sustaining Capital	
Mining (US$/ total tonne)	$0.27
Process Plant (US$/ ore tonne)	$0.25

Allowances for reclamation costs and salvage value (positive cash flow) at closure are as follows.

Closure	
Salvage Value (US$ million)	$2.0
Reclamation Cost (US$ million)	$10.0

18.9.10 Estimate Accuracy and Contingency

The capital cost estimates have a scoping study accuracy range of ±35-50%. For the purposes of the financial modeling no contingency is applied to costs as the estimates reflect the approximate mid-point on the expected final cost curve. The impact of changes to capital costs on financial returns is covered under Sensitivities.

18.9.11 Key Financial Indicators

The financial indicators in Table 18-20 are outputs of the financial model for the 40,000 tpd acid-chloride Base Case.

Table 18-20 Key Financial Indicators - Acid-chloride Case

Measure	Base Case	Mid-Case	Recent
Metal Prices			
Silver (US$/ounce)	$18.00	$25.00	$35.00
Indium (US$/kg)	$500.00	$570.00	$650.00
Lead (US$/lb)	$0.90	$1.00	$1.20
Zinc (US$/lb)	$0.90	$1.00	$1.10
Copper (US$/lb)	$3.00	$3.70	$4.30
Gallium (US$/kg)	$500.00	$570.00	$730.00
Average Operating Cash Flow			
1st 5 years (per year)	$185 M	$287 M	$430 M
Life-of-Mine (per year)	$124 M	$208 M	$325 M
Net Cash Flow (undiscounted)	$1,261 M	$2,528 M	$4,298 M
NPV at 5% discount rate	$704 M	$1,482 M	$2,571 M
NPV at 8% discount rate	$505 M	$1,104 M	$1,942 M
Internal rate of Return (IRR %)	37.7%	63.0%	92.9%
Payback period	27 months	19 months	15 months
1st 5 years			
Silver cash costs before credits (US$/ounce)	$9.70	$9.70	$9.70
Silver cash costs after credits (US$/ounce)	$2.94	$2.01	$0.86
Life of Mine			
Silver cash costs before credits (US$/ounce)	$13.87	$13.87	$13.87
Silver cash costs after credits (US$/ounce)	$5.06	$3.85	$2.39

18.9.12 Sensitivities

The financial model has been used to test the sensitivity of the project to a range of variables. The chart below (Figure 18-15) represents the sensitivity of the 40,000tpd acid-chloride heap leach Base Case to changes in key variables by charting the impact of ±20% deviations from the Base Case set point on NPV (at 5% discount rate).

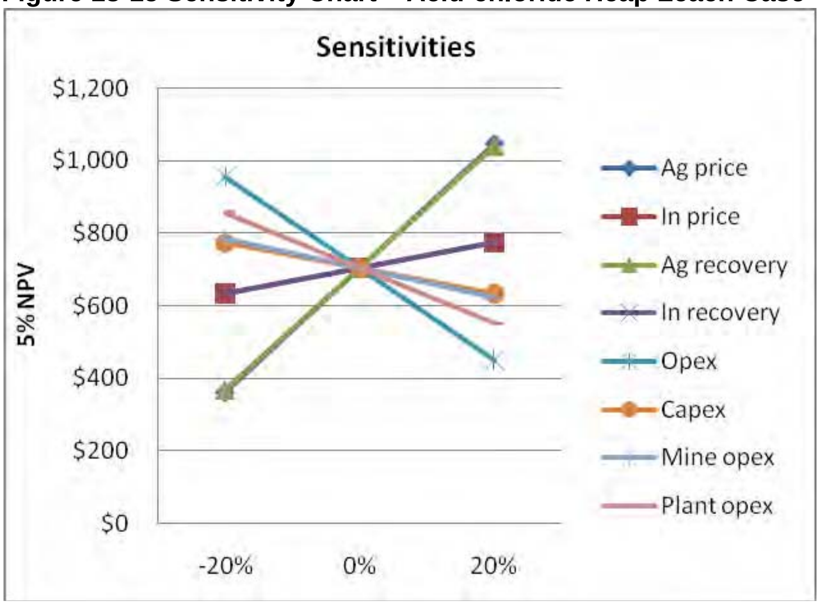
- The Base Case project is most sensitive to silver price and recovery, with each 1% change in either silver price or recovery impacting NPV by around $17M.
- The project is also sensitive to operating cost changes (mining, processing and G&A), with each 1% change impacting NPV by around $12.7M.
- Changes in processing costs (~$7.6M NPV change for each 1% change) have a greater impact than changes in mining costs (~$4.1M NPV change for each 1% change).
- NPV changes by ~$3.5M for each 1% change in capital costs.

Table 18-21 Acid-chloride Heap Leach Sensitivities

Variable	*Variable changed -20%*	*Variable changed 0%*	*Variable changed 20%*
Ag price	$360	$704	$1,047
In price	$633	$704	$774
Ag recovery	$367	$704	$1,040
In recovery	$633	$704	$774
Opex	$957	$704	$450
Capex	$774	$704	$634
Mine opex	$785	$704	$622
Plant opex	$856	$704	$551

All values are US$

Figure 18-15 Sensitivity Chart – Acid-chloride Heap Leach Case



The economic analysis is appropriate for a Preliminary Economic Assessment but further studies will be required to demonstrate a higher degree of economic certainty.

18.10 Project Development Plan

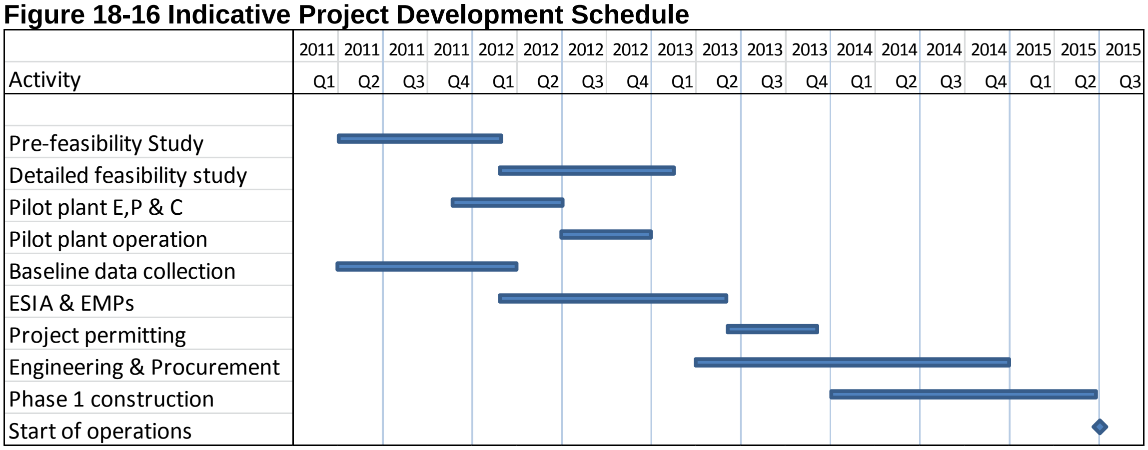
Subject to satisfactory outcomes through the process, SASC proposes to progress towards development of Malku Khota in a number of overlapping but parallel study phases that involve increasing levels of detail.

- On completion of the PEA, SASC proposes to undertake a pre-feasibility study (PFS) and to follow up with a detailed feasibility study (DFS) before advancing to construction.
- The main aim of the PFS will be to demonstrate technical and financial potential and to define the project that will be studied in a subsequent DFS. The DFS would describe the project in detail and ultimately provide the basis for the investment decision.

- Both the PFS and the DFS will include additional definition drilling to progressively promote Inferred Resources to the Measured and Indicated categories and ultimately to demonstrate sufficient Mineral Reserves in the DFS to support an investment decision.
- Further metallurgical testwork will also be required for both the PFS and DFS phases. SASC is currently considering a pilot plant to be constructed on site. The idea is that a pilot plant will be designed and partly procured in the latter part of the PFS in order to be constructed and operated during the DFS.
- Environmental and social baselining (data collection) will be carried out in parallel with the PFS with the aim of collecting at least 12 months of data to feed into the environmental and social impact assessment (ESIA) process.
- The ESIA process starts towards the end of the PFS and would ultimately be based on the project defined by the DFS. The ESIA ultimately leads to the development of the environmental management plans (EMPs) that are designed to mitigate the impacts. These processes eventually lead into the project permitting process.
- Rather than starting out at full capacity, it is possible that a starter “module” of 5,000-10,000tpd capacity would be built initially, stepping up to full capacity progressively over a number of phases. This concept will be examined in subsequent study phases.

Feasibility studies and the parallel environmental, social and permitting processes are inherently uncertain and it is not possible to express a forward-looking timetable with a high degree of accuracy. Notwithstanding the obvious limitations, an indicative timeline for the development of Malku Khota could be as shown in Figure 18-16.

Figure 18-16 Indicative Project Development Schedule



- PFS - Q2, 2011 through Q1, 2012.
- DFS – Q1, 2012 through Q1, 2013.
- Pilot plant operation starting Q3, 2012.
- Environmental and social baseline data collection in parallel with the PFS.
- ESIA in parallel with the DFS, with a completion lag of 3 months, ending Q2, 2013.
- Project permitting – Q2, 2013 through Q4, 2013.
- Construction – Phase 1: Q1, 2014 through Q2, 2015.
- Start of operations - Q2-3, 2015.

Estimated costs to take the project through the PFS and DFS stages are as follows:

MK Site Operations	$4,359,973
SASC Owners Team	$966,720
Resource Definition Drilling	$11,545,617

Resource Modeling & Estimation	$269,219
Mining Engineering	$303,534
Geotechnical Fieldwork and Studies	$1,020,000
Metallurgical and Materials Testing	$1,395,550
Mineral Processing Engineering	$2,114,196
Project Infrastructure Engineering	$1,123,333
Other Feasibility Study Subjects	$504,800
Environmental, Social and Permits	$1,404,160
Expenses & Contingencies	$3,004,462
Grand Total	$28,011,563

18.11 Alternative Case – Cyanide Heap Leach

The project is also amenable to a cyanide leach process which would focus only on recovery of silver with gold credits. The value added by the indium, lead, zinc and gallium from the acid leach process is significant and allows for greater exploitation of the deposits; longer mine life, more metal production and higher NPV. As well, the indium and gallium are regarded as strategic metals that give the project future upside potential. For these reasons the acid leach option is preferred and the cyanide heap leach option is a fallback option that would be pursued in the event that the acid-chloride heap leach option proves not to be viable for some reason. The Alternative Case is a 20,000tpd cyanide heap leach operation.

The cyanide heap leach scenario would be different from the acid chloride leach scenario in the following parameters:

- Mining for the cyanide option would be similar to the Base Case except at 20,000tpd not 40,000tpd as a result of reducing the quantity of economic ore available for processing
- Heap leaching would also be similar to the Base Case except at the lower treatment rate and using cyanide as the leaching agent not acid.
- Recovery of gold and silver would be achieved using the well-known, less capital and operating intensive, Merrill-Crowe process.
- Other infrastructure would be similar to the Base Case, but smaller due to the lower treatment rate, fewer reagent and personnel requirements, lower power requirements, reduced reclamation requirements, and shorter mine life.

Since the 2009 PEA, the focus has been on the acid-chloride option and a modest amount of test work has been performed to advance the cyanide heap leach option. However, as the cyanide leach process is a well proven technology the accuracy of the operating and capital estimates are as good as or better than the information available for the acid-chloride leach process.

- The cyanide leach scenario for the Malku-Khota ores have been reviewed and refined using more recent consumable and labor rates.
- Process operating costs and associated impacts on the calculations used in the mine block model have been included and updated in a new cyanide leach scenario model.
- Material balances and equipment requirements have been updated.
- Capital costs have been updated to reflect the latest operating scenario information.

18.11.1 Operating Scenario

Ore from the mine will be three stage crushed to -3/8”. The crushed ore will be stacked in 8 m lifts, piped with drip irrigation tubing, and leached with a weak sodium cyanide leach solution. The leach solution percolating through the heap and to the bottom of the leach pile will be collected in lined pregnant leach solution ditches and will flow by gravity to a pregnant leach solution pond.

The pregnant solutions will be pumped continuously to a Merrill-Crowe precious metals recovery plant. The pregnant leach solution entering the metal recovery plant will sequentially be clarified, vacuum de-aerated and contacted with zinc dust in precipitation filters to precipitate the silver and gold content. It has been assumed in this conceptual study that if the cyanide concentration is kept sufficiently high during the precipitation step, any copper value in the pregnant solution will not precipitate. Further it is assumed the sulfidization, acidification, recycle, thickening (SART) process can then be utilized to remove and recover the copper values in the leach solutions. Cyanide will be recycled from the SART circuit and returned to the leach solutions. The recycled cyanide and barren solution from the metal recovery and SART circuits will be returned back to the heap leach pad for reuse in the leaching process. The projected principal parameters of the envisioned process are presented in the following heap leach design criteria and a block flow diagram for the crushing and leach pad circuits are shown in Figure 18-17 and Figure 18-18.

Heap Leach Design Criteria

Lift Height 8.0m
Application Rate =9 L/h/m^2 for 120 days
Density =1.65 t/m^3
Tons/m2/lift =13.2,
Daily Tons =20,000 tonnes
Surface area required: Primary – 363,636m^2

Ag mass balance:
Head Ag =33 ppm
Operating recovery =70% of Ag (possible additional dissolution with time)
PLS Grade 43.07 ppm Ag at 30 days, 10.77 ppm Ag average grade
PLS flow to metal recovery 1,806 m^3/hr
Daily Ag production=462 kg Ag

The leach cycle on a single-lift basis would be 120 days, and it would take ~234,000 m^2 of area to be piped and under various stages of leaching or resting. Additional metal dissolution is likely if the ore is leached in multiple lifts but this has other inventory issues as mentioned below.

Conceptual Leach Design

Once the commercial single-lift mass balance is developed, the decision whether to leach a single-lift or in a multiple-lift must be chosen. There are trade-offs to either approach. The positive factor with leaching this ore in a multiple-lift is the low PLS grade. Although silver recoveries within the heap will climb to increase with increasing lifts, the low barren grade will help to keep the inventories low as compared to inventories experienced in the acid-leach scenario.

Multiple-lift Leach vs. ON/Off “dynamic” Pad

Some added dissolution from the ore in the underlying ore lifts of a multiple-lift leach design can mitigate some of the inventory effects, but until the current testing is completed, it is difficult to estimate what this might be. An on/off pad of 120-day leach duration may be a better economic trade-off under these circumstances. At 120 days, the ore must be taken off the pad to provide room for fresh ore. As for the acid-chloride Base Case, the cyanide heap leach alternative case is based on the On/Off Pad option with no “lock-up” of values in the heap.

Future Column Testing and Piloting

A factorial matrix column test program will be necessary to determine the levels of application rate, cyanide concentration, lime addition, and solution management schemes. In fact, a two- or three-phased test program could be necessary. The first phase may be shorter columns (two to four meters) testing the optimal solution chemistry and then taking this to a full-depth program in a Phase 2 program which would be designed to confirm the basic assumptions in this initial scale-up. Ultimately, a pilot plant program will likely not be necessary to prove up the cyanide leach parameters as they are generally well understood by the industry.

18.11.2 Metal Recovery Plant

Silver and gold values present in the pregnant cyanide leach liquor will be recovered by precipitation with zinc dust in the well-known Merrill-Crowe process. Merrill-Crowe recovery begins with clarification of the pregnant cyanide leach solution to ensure the purest product possible since any suspended solids in the pregnant leach solution would be collected in the filtered precipitate. Clarification is generally accomplished by passing the solutions through a diatomaceous earth pre-coated filter. The next step involves removing dissolved oxygen from the pregnant solution by passing the solution through a vacuum splash tower. Finally, the pregnant solution is contacted with zinc powder just prior to entering a pressure filter where the resulting precipitate is collected. The zinc powder is introduced as slurry into the suction side of the filter press feed pump. The precipitate is periodically collected from the filter press as the chambers fill. Precious metal precipitates are generally dried and can be smelted into doré bars. It has been assumed in this conceptual study that if the cyanide concentration is kept sufficiently high during the precipitation step, copper values in the pregnant solution will not precipitate with the silver and gold. Further it is assumed that the developmental SART process can then be utilized to recover the copper values and recycle cyanide. A bleed stream of barren solution from Merrill-Crowe recovery will be diverted to the SART process circuit. SART involves using a source of sulfide ion, such as sodium hydrosulfide, to precipitate copper as copper sulfide and convert the copper-bound cyanide to free cyanide for recycling. The bleed stream will be acidified to a pH of 5 and copper sulfide will precipitate immediately on addition of the sulfide ion. Following filtration of the copper sulfide product the solution is then re-neutralized and the solution with the recovered free cyanide is returned to the heap leach operation. The metallurgical testing has indicated cyanide consumption up to 3 kg/ tonne. Much of this consumption is due to the copper content. It is presumed for this study that the SART process will keep the overall cyanide consumption below 1 kg/ tonne. A listing of principal reagents and projected consumption levels and costs for the cyanide leach process is presented in Table 18-24

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Figure 18-17 Cyanide Leach Crushing Circuit

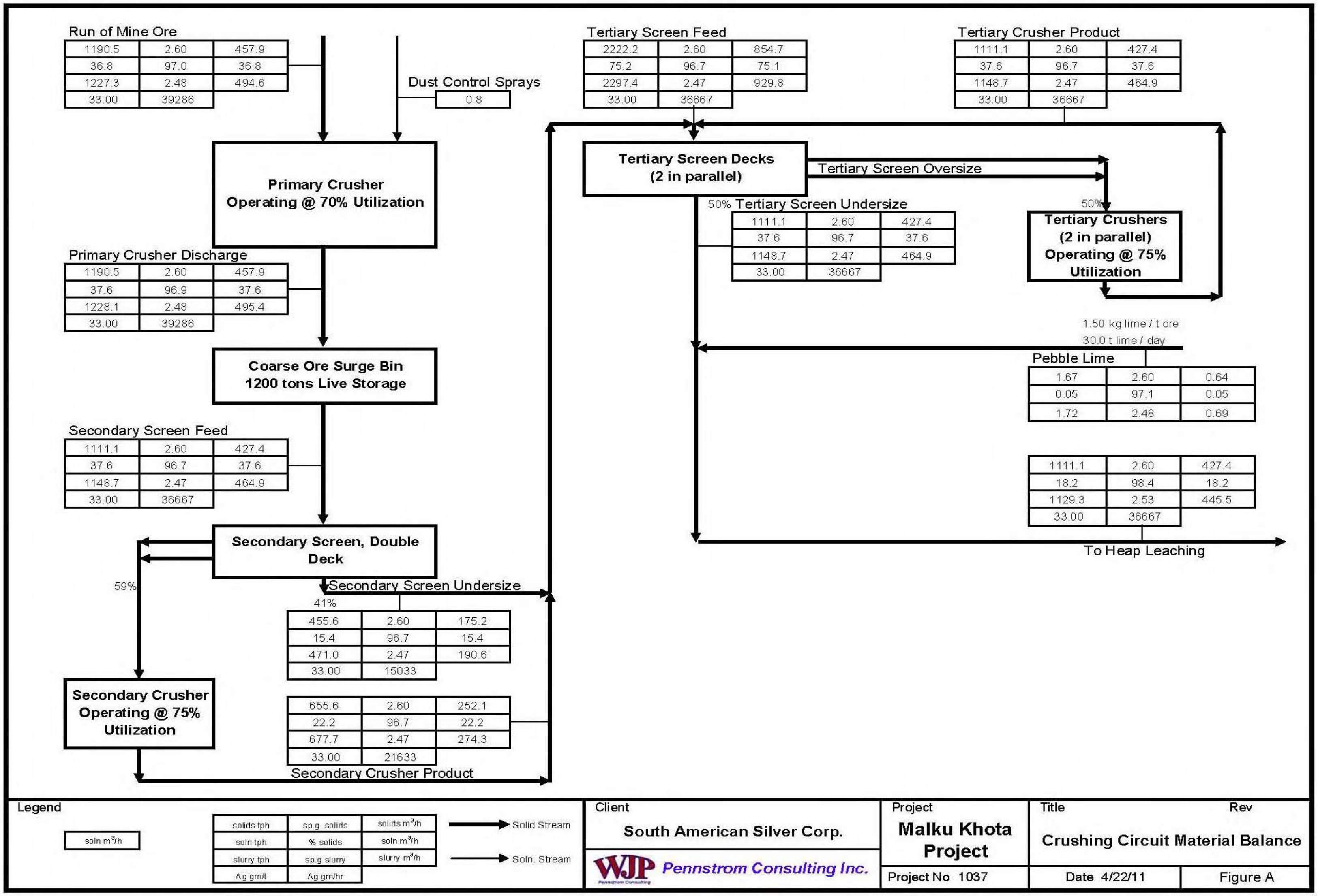
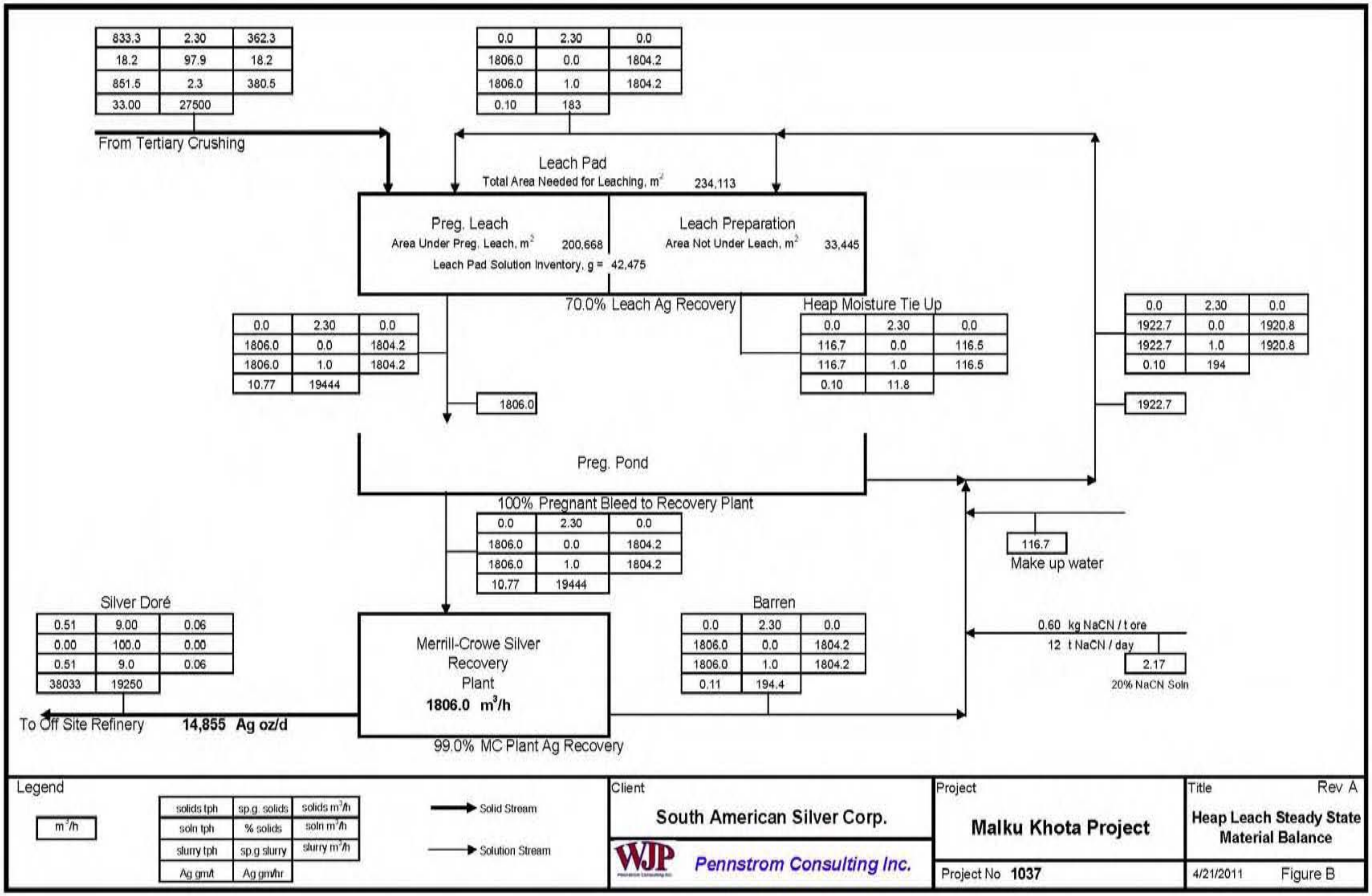


Figure 18-18 Cyanide Heap Leach Circuit



18.11.3 Capital Costs

Mining

Open pits for Limosna, Wara Wara and Sucre were developed for the 20,000tpd cyanide case on the basis of the design parameters shown in Table 18-22.

Table 18-22 Cyanide Case - Pit Design Parameters

Parameter	Value used for pit design
Silver price	$14.70 per ounce
Silver recovery	70%
Mining costs	$1.21 per tonne mined
Processing costs	$3.44 per tonne of leach feed
G&A costs	$1.00 per tonne of leach feed
Wall slope angle	43 degrees

Capital cost schedules for mine pre-production and mining fleet for the cyanide case were prepared based on the open-pit design. The main capital costs arising, which were subsequently used in the financial modeling are as follows:

Initial Capital		
Pre-stripping	Year -1: 0.4Mt	$4.0M
Mining fleet	Year -2	$36.2M
Mining fleet	Year -1	$21.9M
Mining fleet	Year 1	$0.5M

Sustaining Capital		
Life of Mine	Year 2 on	$107.5M
	Average	$0.41/ tonne mined

Process Facilities – Initial Capital

Capital cost estimates for processing facilities listed below are factored on the 40,000tpd acid-chloride heap leach case. Cost estimates for the Merrill-Crowe Plant are based on consultant's data base of similar facilities.

Table 18-23 Cyanide Case - Initial Capital Cost Summary

Area	Estimated Cost (US$x1000)
Process Plant	
Crushing & Screening	$21,500
Leaching	$5,980
Metal Recovery	$29,500
Power Supply	$5,800
Site Development	$2,450
Utilities	$2,610
Office, Lab & Workshops	$2,095
Reagents/ Solids Handling	$4,320
Process Plant Direct Costs	$74,255
Leach Pads & Residues	
Leach Pads & Ponds	$59,100
Residue Facility Pond	$9,540

Area	Estimated Cost (US$x1000)
Leach Pads Direct Costs	$68,640
Total Direct Costs	$142,895
Other Costs	
Camp & Catering	$7,830
Field CM, Office, QA/QC	$5,740
Engineering (EPM)	$13,210
Freight, Taxes & Duties	$6,470
Vendor Reps & Commissioning	$860
Spares & First Fills	$2,180
Owner's Costs	$17,480
Other Project Costs	$53,770
Total Process Plant and Infrastructure	$196.665

Process Facilities – Sustaining Capital

Sustaining capital for the leach pads and residue dams and ponds were estimated as part of the process of estimating initial capital for the facilities. Life-of-Mine Sustaining Capital in the processing facilities equates to $0.33 per tonne mined.

18.11.4 Operating Costs

Open Pit - Operating Cost

Year-by-year operating costs were developed for the 20,000tpd cyanide heap leach case based on the sequenced open-pit design. Life-of-Mine operating costs are estimated at $311.5M, which equates to $1.19 per tonne mined.

Process Facilities – Operating Costs

Processing costs are factored on the acid-chloride heap leach case.

- NaCn will be delivered dry and mixed on site.
- As for the acid-chloride heap leach case, costs are based on “On-off” loading and unloading of the heap leach pads and progressive construction of the residue dam walls after Year 2.

Table 18-24 Cyanide Case - Operating Cost Schedule

Item	Quantity		Rate		Cost/ tonne
Crushing & Grinding					
Power	3.71	kWh/ t	$0.042	/ kWh	$0.16
Liners & grinding media	1	lot	$0.21	/ tonne	$0.21
Conveyor & screen parts	1	lot	$0.30	/ tonne	$0.30
Total - Crushing & Grinding					$0.67
Leaching & Refining					
Power	0.84	kWh/ t	$0.042	/ kWh	$0.04
Maintenance	1	lot	$0.10	/ lot	$0.10
NaCn	0.60	kg/ tonne	$1740.00	/ tonne	$1.04
Lime (CaO)	1.50	kg/ tonne	$105.00	/ tonne	$0.16

Item	Quantity		Rate		Cost/ tonne	
Cement	0.50	kg/ tonne	$85.00	/ tonne	$0.04	
Pad piping & materials	1	lot	$0.15	/ tonne	$0.15	
Merrill-Crowe Reagents	1	lot	$0.25	/ tonne	$0.25	
Total - Leaching & Refining					$1.78	
Other Processing Costs						
Solution/ residue handling	1	lot	$0.40	/ tonne	$0.40	
Heap leach handling	1	lot	$0.43	/ tonne	$0.43	
Labor	1	lot	$0.16	/ tonne	$0.16	
Total - Other Processing					$0.99	
Total - Processing Costs					$3.44	/ tonne

18.11.5 Project Economics

The main parameters for the economic evaluation of the cyanide option remain the same as the acid-chloride heap leach except as noted below.

- Table 18-25 is a summarized extract from the year-by-year mine production plan for the 20,000tpd cyanide heap leach case.

Table 18-25 Cyanide Case - Mining Tonnes and Grade

Year	Ore tonnes	Waste tonnes	Strip Ratio	Ag grade (g/t)
1	5,000,000	2,221,500	0.58	41.07
2	7,000,000	3,304,900	0.47	53.17
3	7,000,000	3,270,400	0.47	58.35
4	7,000,000	5,305,000	0.76	52.82
5	7,000,000	6,290,800	0.90	45.12
6	7,000,000	2,868,500	0.41	42.15
7	7,000,000	5,611,100	0.80	44.13
8	7,000,000	12,231,100	1.75	44.31
9	7,000,000	12,140,500	1.73	34.20
10	7,000,000	12,224,400	1.75	27.48
11	7,000,000	12,017,100	1.72	29.78
12	7,000,000	11,255,800	1.61	34.75
13	7,000,000	11,226,800	1.60	26.80
14	7,000,000	11,857,300	1.69	30.96
15	7,000,000	12,044,300	1.72	27.15
16	7,000,000	5,085,300	0.73	31.38
17	7,000,000	4,837,100	0.69	35.37
18	5,811,900	2,419,400	0.42	40.47
19	866,900	122,300	0.14	79.93

- Except for by-product credits, only silver prices are relevant for the cyanide case. The following silver prices were used in the financial evaluation:
  - Base Case: $18.00 per ounce
  - Middle-Case: $25.00 per ounce
  - Recent Case: $35.00 per ounce
- Silver recoveries of 70% were used to evaluate the cyanide option.

- As for the acid-chloride heap leach case, the cyanide option financial analysis uses a Pay Factor of 97% for silver and provision for TC/RCs and transport of $0.36 per ounce.
- As for the acid-chloride case, by-product credits for gold are included $0.72/ tonne.
- Operating Costs used in the financial model are rates shown in the table below. Mining costs are based on the mining schedule developed specifically for the 20,000tpd cyanide case. Processing and G&A costs are factored on the acid-chloride case.

Operating Cost Summary

Mining	$1.19	per tonne mined
Processing	$3.44	per tonne of leach material
G&A	$1.00	per tonne of leach material

- Initial Capital Costs are summarized in the financial model as shown below.

Capital Cost Summary

Mine pre-production	$4.0 M
Mining Fleet	$58.6 M
Process Plant & Infrastructure	$74.3 M
Leach Pads and Residue Facilities	$68.6 M
Indirect Costs	$53.8 M
Total Initial Capital	$259.3 M

- Working Capital requirements are based on 90 days between the date of sale (production) and receipt of payment.
- Deferred and sustaining capital requirements for the production phase are estimates based on the specific 20,000tpd cyanide case. Closure costs use the figures derived for the acid-chloride heap leach case.

Sustaining Capital

Mining (US$/ total tonne)	$0.41
Process Plant (US$/ ore tonne)	$0.33

Closure

Salvage Value (US$ million)	$2.0
Reclamation Cost (US$ million)	$10.0

The financial indicators in Table 18-26 are outputs of the financial model for the 20,000tpd cyanide Alternative Case for the three different metal price cases.

Table 18-26 Cyanide Case - Key Financial Indicators

Measure	Base Case	Mid-Case	Recent
Silver Price (per ounce)	$18.00	$25.00	$35.00
Recovered silver (ounces)	108,818,628	108,818,628	108,818,628
Project cash flow ($US million)	$712	$1,454	$2,514
NPV at 5% discount rate ($US million)	$366	$796	$1,410
Internal Rate of Return	27.0%	44.4%	64.8%
Payback period	36 months	26 months	21 months
Life on Mine	19 years	19 years	19 years

Measure	Base Case	Mid-Case	Recent
Averages 1st 5 years			
Ag production (ounce/ years)	7,524,099	7,524,099	7,524,099
Ag grade (g/t)	50.65	50.65	50.65
Silver cash costs (per ounce)	$5.10	$5.10	$5.10
Cash flow pa (US$ million/ year)	$80.6	$132.1	$205.7

18.11.6 Sensitivities

Table 18-27 represents the sensitivity of the 20,000tpd cyanide heap leach case to changes in key variables by comparing the impact of ±20% deviations from the Base Case set point on NPV (at 5% discount rate).

Table 18-27 Cyanide Case - Sensitivities

Variable	*Variable Changed -20%*	*Variable Changed 0%*	*Variable Changed 20%*
Ag price	$145	$366	$587
Ag recovery	$149	$366	$582
Opex	$461	$366	$271
Capex	$410	$366	$322
Mine opex	$399	$366	$332
Plant opex	$413	$366	$318

All values are US$

18.12 Acid-chloride and Cyanide cases compared

Table 18-28 compares financial model inputs and outputs for the 40,000tpd acid-chloride heap leach Base Case to the 20,000 tpd cyanide heap leach case, both using Mid-Case metal pricing.

Table 18-28 Financial Indicator Comparison – Acid-chloride vs. Cyanide

Measure	Unit	40,000 tpd Acid Heap	20,000 tpd Cn Heap
Metal Prices			
Silver	/ounce	$25.00	$25.00
Indium	/kg	$570.00	$0.00
Lead	/lb	$1.00	$0.00
Zinc	/lb	$1.00	$0.00
Copper		$3.70	$0.00
Gallium	/kg	$570.00	$0.00
Recoveries			
Silver	%	73.6%	70.0%
Indium	%	81.0%	0.0%
Lead	%	51.1%	0.0%
Zinc	%	62.0%	0.0%
Copper	%	84.8%	0.0%
Gallium	%	26.9%	0.0%
Case Variables			
Gold credits	$/t ore	$0.72	$0.72
Mine Opex	$/t mined	$1.08	$1.19
Processing Opex	$/t ore	$6.41	$3.44
G&A Opex	$/t ore	$0.80	$1.00

Measure	**Unit**	**40,000 tpd Acid Heap**	**20,000 tpd Cn Heap**
Mining Pre-production Capital	$/t ore	$3,788	$3,958
Mining Fleet Capital	$'000	$83,085	$58,609
Plant & Infrastructure Capital	$'000	$324,556	$196,665
Mine Sustaining Capital	$/t mined	$0.27	$0.41
Plant Sustaining Capital	$/t ore	$0.25	$0.33
Salvage Value at end	$'000	-$2,000	-$2,000
Reclamation Cost	$'000	$10,000	$10,000
Output Summary			
Total tonnes	tonnes	642,341,628	260,012,277
Ore processed	tonnes	198,831,830	123,678,749
Strip Ratio		2.23	1.10
Mining Operations	years	15	19
Average Silver grade	g/t	33.60	39.09
Average Indium grade	g/t	7.35	0.00
Recovered Silver	ounce	158,083,641	108,818,628
Recovered Indium	kg	1,183,630	0
Recovered Lead	lbs	191,401,353	0
Recovered Zinc	lbs	135,057,299	0
Recovered Copper	lbs	88,114,119	0
Payable Gallium	kg	212,963	0
Gross Revenue - Silver	million	$3,834	$2,639
Net Revenue	million	$5,247	$2,689
Operating Costs	million	$2,126	$858
Silver cash costs before credits	$/ounce Ag	$13.87	$8.13
Silver cash costs after credits	$/ounce Ag	$3.85	$7.29
Net Operating Profit	million	$3,121	$1,830
Initial Capital	million	$372	$244
LOM Sustaining Capital	million	$209	$120
Net Cash Flow	million	$2,528	$1,454
NPV at 2.5%	million	$1,922	$1,066
NPV at 5%	million	$1,482	$796
NPV at 10%	million	$915	$461
IRR		63.0%	44.4%
Payback period	years	1.6	2.2
Averages in 1st 5 years			
Silver production	ounces	13,250,708	7,524,099

Whilst both the 40,000tpd acid-chloride heap leach and the 20,000tpd cyanide heap leach are financially robust projects, the acid-chloride heap leach is preferred as it provides for greater exploitation of the deposits, longer mine life, more metal production and higher NPV. As well, the indium and gallium are regarded as strategic metals that give the project future upside potential. Both options will be held open for further study during the PFS.

19 INTERPRETATION AND CONCLUSIONS

The technical and economic evaluations in this PEA demonstrate that the silver-indium deposits at Malku Khota have sufficient value to support a decision to advance to PFS and, assuming satisfactory findings, then to advance to DFS. The decision to proceed to construction will, amongst other things, be contingent on the results of the DFS.

The operational plans described in this study are preliminary in nature and include Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the operational LOM plans can be realized.

20 RECOMMENDATIONS

The technical and economic evaluations in this PEA demonstrate that the silver-indium deposits at Malku Khota have sufficient value to support a decision to advance to PFS and, assuming satisfactory findings, then to advance to DFS. The decision to proceed to construction will, amongst other things, be contingent on the results of the DFS.

The essential objective of the PFS will be to confirm that a potentially viable project worthy of further examination in a DFS exists. This will involve identifying and evaluating obstacles to the project, collecting additional information and examining and selecting between development alternatives. The DFS will ultimately put the economic case for the project and be the basis for the environmental and social review process, the separate but related project permitting process, and financial commitment to its ultimate development.

It is recommended that the following areas are addressed in the PFS as they have the greatest potential to impact project outcomes.

- Carry out infill drilling of the deposits in the early stage of the PFS to ensure that sufficient Inferred Resources are promoted to Measured and Indicated categories to support the decision to proceed to DFS.
- Maintain a focus on metallurgical and materials testwork, particularly:
    - leaching testwork to improve the understanding of the relationship between recoveries and the different ore types in the orebody;
    - hydrodynamic and geotechnical testing of heap leach ores to verify the design assumptions based on preliminary tests carried out during the PEA;
    - metals recovery testwork to allow detailed design criteria to be developed for the design phase.
- Carry out the following work related to open pit mining:
    - Bring all leach pad material within the pit boundaries to Indicated or Measured resource categories in order to allow the tonnages to be used in the Pre-Feasibility study.
    - Undertake ARD testwork on all expected waste materials to ensure that no long-term waste storage issues exist.
    - Carry out geotechnical test work on all the pit areas as well as foundations of the proposed waste dump locations, considering different rock types, wall slope orientations, and potential faulting.
    - Develop rock strength parameters suitable for accurate blasting estimation and crusher design.
    - Assemble a comprehensive cost database for labor, explosives and equipment.
    - Carry out trade-off studies pertaining to the electrification of the pit for the shovels and drills for operating cost optimizations.
    - Carry out condemnation drilling beneath all waste dump locations to ensure mineralized material is not trapped beneath these areas.
- Investigate the following process enhancement opportunities:
    - Preliminary testwork indicates that leaching recoveries improve at finer material sizes suggesting that vat leaching or possibly agitated leaching might produce

higher economic returns than heap leaching. These options should be investigated as part of the PFS.

- Early-stage testwork suggests that significant recovery gains could be achieved at higher leach reaction temperatures. Higher solution temperatures are also a benefit in metals recovery circuits. This would allow more value to be extracted from the orebody and warrants further investigation.
- There is evidence that power can be generated economically by using Bolivia's cheap natural gas at a significant saving on power from Bolivia's electrical grid. This should be investigated further for its potential benefit to the project
- Acid consumption is a significant cost of operation and the potential to reduce costs and simplify logistics by making acid on site should be studied in the next stage.

- Continue to develop the cyanide case while it remains a viable alternative to the acid-chloride leach case noting that the competitiveness of the cyanide case or a blended cyanide/acid leach case increases as the silver price increases relative to other metals.
- Develop a detailed plan for a pilot plant to be built on site and operated during the DFS.
- Carry out PFS-level investigations and engineering studies into:
  - Important infrastructure aspects, specifically power supply, water supply, transport and logistics.
  - hydrological, hydrogeological and geotechnical aspects of the site to facilitate detailed design and cost estimating.
- Carry out a study to develop a better understanding of the market for the products that will arise from the project. The study will need to provide sufficient detail regarding the payment terms including "payability", TC/RCs, penalties and transport costs for trade-off studies to be undertaken.
- Advance environmental and social baselining to ensure that sufficient data is gathered) in advance of the ESIA process starting in earnest (minimum 12 months required).

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22 DATE AND SIGNATURE PAGES

Allan Armitage
Pierre Desautels
Gordon Zurowski
William J. Pennstrom Jr.
Ralph Fitch
Felipe Malbran

QP CERTIFICATE – ALLAN ARMITAGE

To Accompany the Report titled Preliminary Economic Assessment Update, Technical Report for the Malku Khota Project, Department of Potosi, Bolivia dated May 10, 2011 (the "Technical Report")

I, Allan E. Armitage, Ph. D., P. Geol. of #35, 1425 Lamey's Mill Road, Vancouver, British Columbia, hereby certify that:

1. I am currently a consulting geologist with GeoVector Management Inc., 10 Green Street Suite 312 Ottawa, Ontario, Canada K2J 3Z6

2. I am a graduate of Acadia University having obtained the degree of Bachelor of Science – Honours in Geology in 1989.

3. I am a graduate of Laurentian University having obtained the degree of Masters of Science in Geology in 1992.

4. I am a graduate of the University of Western Ontario having obtained a Doctor of Philosophy in Geology in 1998.

5. I have been employed as a geologist for every field season (May – October) from 1987 to 1996. I have been continuously employed as a geologist since March of 1997.

6. I have been involved in mineral exploration for gold, silver, copper, lead, zinc, nickel, uranium and diamonds in Canada, Mexico, Honduras, and the Philippines at the grass roots to advanced exploration stage, including resource estimation since 1991.

7. I am a member of the Association of Professional Engineers, Geologists and Geophysicists of Alberta and use the title of Professional Geologist (P.Geol.).

8. 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 of my professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

9. I am responsible for section 17, Mineral Resource Estimate of the Technical Report.

10. I have not examined the property in the field.

11. I have no prior involvement with the property that is the subject of the Technical Report.

12. I am independent of South American Silver Corp., as defined by Section 1.4 of NI 43-101.

13. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

14. I have read NI 43-101 and Form 43-101F1 (the "Form"), and the Technical Report has been prepared in compliance with NI 43-101 and the Form.

15. I consent to the filing of the Technical Report with and stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

16. Signed and dated this tenth day of May, 2011 at Vancouver, British Columbia.

Signed and Sealed

Allan Armitage, Ph.D., P.Geo

QP CERTIFICATE – PIERRE DESAUTELS

To Accompany the Report titled Preliminary Economic Assessment Update, Technical Report for the Malku Khota Project, Department of Potosi, Bolivia dated May 10, 2011 (the "Technical Report")

I, Joseph Rosaire Pierre Desautels, as one of the authors of the Technical Report, do hereby certify that:

1. I reside at 290 Harvie Road, Barrie, Ontario, Canada, L4N 8H1.
2. I am a graduate of Ottawa University (B.Sc. Hons., 1978).
3. I am a member in good standing of the Association of Professional Geoscientists of Ontario, Registration number 1362.
4. I am a Principal Geologist with AGP Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 246, Barrie, Ontario, L4N 0Z7.
5. I have practiced my profession in the mining industry continuously since graduation.
6. 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 purpose of NI 43-101.
7. My relevant experience with respect to resource modeling includes 30 years experience in the mining sector covering database, mine geology, grade control and resource modeling. I was involved in numerous projects around the world in both base metals and precious metals deposits.
8. I visited the Malku Khota deposit between 11 and 13 November 2010.
9. I am responsible for the preparation of section 12, 13 and 14 of the Technical Report.
10. 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.
11. I have no prior involvement with the property that is the subject of the Technical Report.
12. I am independent of the Issuer as defined by Section 1.4 of NI 43-101.
13. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
14. Signed and dated this 10th day of May, 2011 at Barrie, Ontario.

Signed and Sealed

____

Pierre Desautels, P.Geo

QP CERTIFICATE - GORDON ZUROWSKI

To Accompany the Report titled Preliminary Economic Assessment Update, Technical Report for the Malku Khota Project, Department of Potosi, Bolivia dated May 10, 2011 (the "Technical Report")

I, Gordon Zurowski, of Stouffville, Ontario, as one of the authors of the Technical Report, do hereby certify that and make the following statements:

1. I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 246, Barrie, Ontario, L4N 0Z7.
2. I am a graduate of University of Saskatchewan, B.Sc. Geological Engineering, 1989.
3. I am a member in good standing of the Association of Professional Engineers of Ontario, Registration #100077750.
4. I have practiced my profession continuously since graduation.
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 purpose of NI 43-101.
6. My relevant experience includes the design and evaluation of open pit mines for the last 22 years.
7. I am responsible for the preparation of mining-related sections in Section 18 of the Technical Report including 18.5, 18.6.1, 18.7.1, 18.8.1, 18.9.1, 18.11.3, 18.11.4, and 18.11.5.
8. I have no prior involvement with the property that is the subject of the Technical Report.
9. I visited the Malku Khota project between the 11th and 13th of November 2010.
10. As of the date of this Certificate, to 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.
11. I am independent of South American Silver Corp. as defined by Section 1.4 of NI 43-101.
12. I have read NI 43-101and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
13. Signed and dated this 10th day of May, 2011, at Barrie, Ontario.

Signed and Sealed

Gordon Zurowski, P.Eng

QP CERTIFICATE – WILLIAM J. PENNSTROM JR.

To Accompany the Report titled Preliminary Economic Assessment Update, Technical Report for the Malku Khota Project, Department of Potosi, Bolivia dated May 10, 2011 (the "Technical Report")

I, William J. Pennstrom, Jr., do hereby certify that:

1. I am self employed as a Consulting Process Engineer and I am President of:
   Pennstrom Consulting Inc.
   2728 Southshire Rd.
   Highlands Ranch, CO 80126

2. I graduated in 2001, with a Master of Arts degree in Management from Webster University, St. Louis, Missouri.

3. I graduated in 1983 with a Bachelors of Science degree in Metallurgical Engineering from the University of Missouri - Rolla, Rolla, Missouri (presently known as Missouri S&T).

4. I am a Founding Registered Member of the Society for Mining, Metallurgy, and Exploration (SME).

5. I am a recognized Qualified Professional (QP) Member with expertise in Metallurgy of the Mining and Metallurgical Society of America (MMSA).

6. I have worked in the Mineral Processing Industry for a total of 32 years since before, during, and after my attending the University of Missouri.

7. 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 purpose of NI 43-101.

8. I have reviewed the metallurgical testing programs described in Section 16 and am responsible for the process plant design described in Sections 16 and 18 of the Technical Report. I believe the Technical Report correctly estimates the metallurgical parameters for processing the Malku Khota ores represented by the samples that have been tested.

9. I have not visited the Malku Khota project site.

10. Pennstrom Consulting Inc. has not worked on the Malku Khota project prior to June 2009.

11. I may not be considered "independent" of South American Silver Corp. (the "Issuer") under section 1.4 of NI 43-101 due to my holding options to acquire common shares of the Issuer.

12. I have read National Instrument 43-101 and Form 43-101F1 (the "Form") and, to my knowledge, the Technical Report has been prepared in compliance with NI 431-101 and the Form.

13. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

14. Signed and dated this 10th day of May, 2011 at Highlands Ranch, Colorado.

Signed

William J. Pennstrom Jr.

QP CERTIFICATE – RALPH FITCH

To Accompany the Report titled Preliminary Economic Assessment Update, Technical Report for the Malku Khota Project, Department of Potosi, Bolivia dated April 2011 (the "Technical Report")

I, Ralph Gordon Fitch, as one of the authors of the Technical Report, do hereby certify that:

1. I reside at 3243 South Adams Way, Denver CO 80210 USA.

2. I am a graduate of The University of London, Imperial College with a BSc.(Special) Hons., A.R.C.S. in geology in 1965. I have practiced my profession continuously since 1965.

3. I am a member in good standing of The Australasian Institute of Mining and Metallurgy, Australia (member 100188, member since 1973), member of the Society of Economic Geologists (SEG 261085), member of SME (1010990).

4. I am the Executive Chairman, South American Silver Corp. (the "Issuer"), with a business address at 2696 S Colorado Blvd., Suite 240, Denver CO 80222, USA.

5. Since 1965, I have been involved in mineral exploration, evaluation and management of exploration activities for gold, silver, copper, lead, zinc, indium, gallium, tantalum, nickel, platinum and diamonds amongst others; mostly within West and South Africa, Australia, USA and South America and specifically in Bolivia since 1995.

6. 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 purpose of NI 43-101.

7. My relevant experience with respect to mining and exploration includes 46 years experience in these sectors covering early stage reconnaissance exploration through pre-feasibility and as a mine geologist. I have spent the last 16 years as a CEO or Executive Chairman of a Canadian listed Company directing exploration in Bolivia, Chile, Peru and the USA.

8. In my capacity as Executive Chairman and prior to that CEO, of the Issuer or predecessor Company General Minerals Corporation, I have been involved with the Malku Khota Project since it was first acquired in July 2003.

9. My most recent personal inspection of the Malku Khota Project was from [date] to [date].

10. I am responsible for the preparation of section 1,2,3,18,19,20,21,22 and 23 of the Technical Report.

11. I am not independent of the Issuer applying the test set out in Section 1.4 of the NI43-101 due to my position as Executive Chairman of the Issuer.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

13. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

14. Signed and dated this 10th day of May 2011, Denver, Colorado, USA.

Signed

Ralph Fitch

QP CERTIFICATE – FELIPE MALBRAN

To Accompany the Report titled Preliminary Economic Assessment Update, Technical Report for the Malku Khota Project, Department of Potosi, Bolivia dated May 10, 2011 (the "Technical Report")

I, Felipe Bernardo Malbran, as one of the authors of the Technical Report, do hereby certify that:

1. I reside at Las Fresas 5161, Santiago, Chile.

2. I am a graduate of University of Chile with a Bachelor of Science degree in geology in 1984 and as geologist in January 1987. I have practiced my profession since 1983 and continuously since 1987.

3. I am a member in good standing of The Australasian Institute of Mining and Metallurgy, Australia (member 301110), member of the Society Of Economic Geologists (SEG 366075), member of Institute of Mining Engineers of Chile (member 2350), member of the Geologists Association of Chile (CGCh).

4. I am the Executive Vice President, Exploration with South American Silver Corp. (the "Issuer"), with a business address at Padre Mariano 10, suite 408, Santiago, Chile.

5. Since 1987, I have been involved in mineral exploration and evaluation for gold, silver, copper, lead, zinc and tantalum; in Chile, Peru, Bolivia and Argentina.

6. 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 purpose of NI 43-101.

7. My relevant experience with respect to mining exploration includes 26 years experience in the mining exploration sector covering exploration, geologic setting, drilling and geochemical sample methodology and preparation and data verification. I was involved in numerous projects in South America in both base metals and precious metals deposits.

8. In my capacity as Executive Vice President, Exploration of the Issuer, I have been involved with the Malku Khota Project since the beginning in July 2003.

9. My most recent personal inspection of the Malku Khota Project was in April 2011.

10. I am responsible for the preparation of section 4, 5, 6, 7, 8, 9, 10 and 11 of the Technical Report.

11. I am not independent of the Issuer applying the test set out in Section 1.4 of the NI43-101 due to my position as Executive Vice President, Exploration of the Issuer.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

13. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

14. Signed and dated this 10th day of May, 2011 at Santiago, Chile.

Signed

Felipe Malbran, Geo

23 ADDITIONAL REQUIREMENTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

The Malku Khota property is an exploration property and as such has not reached the stage of a development or producing property. This section is not therefore relevant to this report.

24 ILLUSTRATIONS

For ease of reference, illustrations are included in the relevant sections of the document.