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TIMMINS GOLD CORP.

NI 43-101 F1 TECHNICAL REPORT ON THE PRELIMINARY FEASIBILITY STUDY FOR THE SAN FRANCISCO GOLD PROJECT SONORA, MEXICO

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TABLE OF CONTENTS

List of Tables

List of Figures

LIST OF ABBREVIATIONS

Name	Abbreviation
Accurassay Laboratories	Accurassay
Acme Analytical Laboratories Ltd.	ACME
Adsorption/desorption/recovery	ADR
ALS Chemex Laboratories	ALS Chemex
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian National Instrument 43-101	NI 43-101
Centimetre(s)	cm
Compania Fresnillo S.A. de C.V.	Fresnillo
Defiance Mining Corporation	Defiance
Degree(s), Degrees Celsius	^{o} ,^{o}C
Digital elevation model	DEM
Dirección General de Minas	DGM
Discounted cash flow	DCF
Diversified Drilling, S.A. de C.V.	Diversified
Explotaciones Mineras Del Noroeste S.A. de C.V.	Explotaciones Mineras
Geomaque de Mexico, S.A. de C.V.	Geomaque de Mexico
Geomaque Explorations Inc.	Geomaque
Golder Associates Ltd.	Golder Associates
Grams per metric tonne	g/t
Hectare(s)	ha
Inch(es)	in
Independent Mining Consultants, Inc.	IMC
Inductively Coupled Plasma – Emission Spectrometry	ICP-ES
Internal diameter	ID
Internal rate of return	IRR
Impuesto al Valor Agregado (or VAT)	IVA
Kappes, Cassiday and Associates	Kappes Cassiday
Kilogram(s)	kg
Kilometre(s)	km
Life of mine	LOM
Litre(s)	L
McCelland Laboratories Inc.	McCelland
METCON Research Inc.	METCON
Metre(s)	m
Mexican peso	MXN
Micon International Limited	Micon
Million (eg Million tonnes, Million ounces, Million years)	M (Mt, Moz, Ma)
Milligram(s)	mg
Millimetre(s)	mm
Molimentales del Noroeste de S.A. de C.V	Molimentales
North American Datum	NAD
Net present value, at discount rate of 5%/y, 10%/y, 15%/y	NPV, NPV_5, NPV_{10}, NPV_{15}
Net smelter return	NSR
Not available/applicable	n.a.
Ounces (troy)/ounces per year	oz, oz/y
Parts per billion, part per million	ppb, ppm
Percent(age)	%

Name	Abbreviation
Quality Assurance/Quality Control	QA/QC
Run of mine	ROM
Servicios Industriales Peñoles, S.A. de C.V.	Peñoles
SGS Mineral Services	SGS
Sol & Adobe Ingenieros Asociados S.A. de C.V.	Sol & Adobe.
Specific gravity	SG
Square kilometre(s)	km^2
Three-dimensional	3D
Timmins Gold Corp.	TMM
Timmins Goldcorp Mexico, S.A. de C.V.	Timmins
Tonne (metric)/tonnes per day	t, t/d
Tonne-kilometre	t-km
Tonnes per cubic metre	t/m^3
TSL Laboratories Inc.	TSL
United States Dollar(s)	USD or $
Universal Transverse Mercator	UTM
Value Added Tax or Impuesto al Valor Agregado	VAT or IVA
Year	Y, y

1.0 SUMMARY

Timmins Gold Corp. (TSX-V:TMM) (TMM) has retained Micon International Limited (Micon) to prepare a Preliminary Feasibility Study on its San Francisco project in the state of Sonora, Mexico. The purpose of this technical report is to support disclosure of the results of that study in a form compliant with Canadian National Instrument (NI) 43-101.

Prior to this, Micon prepared a technical report describing the project, dated December 20, 2005. Later, that report was updated in support of disclosure of TMM's 2006 drilling results in a report dated February 23, 2007. The latter report was co-authored by Michael G. Hester, FAusIMM of Independent Mining Consultants, Inc. (IMC) and included a mineral resource estimate for the project completed by Mr. Hester. That resource estimate remains valid and has provided the basis for the preparation of the Preliminary Feasibility Study described herein.

Property Description and Location

The San Francisco property is situated in the north-central portion of the state of Sonora, Mexico approximately 150 km north of the city of the state capital, Hermosillo. In this report, the term San Francisco project ('the project') refers to the area related to the exploitation or mining concessions optioned by TMM, while the term San Francisco property ('the property') refers to the entire land package (mineral exploitation and exploration concessions) optioned and owned by TMM.

The project is comprised of two previously mined open pits (San Francisco and La Chicharra) together with heap leach processing facilities and associated infrastructure located close to the San Francisco pit.

TMM advises that it holds its interest in the San Francisco property through its wholly-owned Mexican subsidiary Timmins Goldcorp Mexico, S.A. de C.V. (Timmins). Timmins originally acquired the rights to the exploitation concessions on April 18, 2005 upon signing an option agreement with Geomaque de Mexico, S.A. de C.V. (Geomaque de Mexico). The option agreement has now been superceded by an acquisition agreement. Briefly, the purchase price was set at USD 5,000,000 plus 10,000,000 shares at a deemed price of USD 0.50 per share. The existing equipment has a purchase price of USD 3,500,000 plus VAT which amount is due without interest at the end of the three years from May, 2007. Only the equipment purchase price remains outstanding. The security documentation to place a lien against the equipment is being finalized.

In 2005 Timmins staked a number of exploration concessions surrounding the exploitation concessions. Interspersed within the Timmins concessions were four mineral concessions controlled by other parties, but these do not impact the main area of interest covering the San Francisco project. All concessions are subject to a semi-annual fee and the filing of reports in May of each year covering the work accomplished on the property between January and

December of the preceding year. The total semi-annual fee payable to the Mexican government for the group of concessions is presently estimated to be USD 32,970.

The Mexican mining laws were changed in 2005 and as a result of these changes all mineral concessions granted by the Dirección General de Minas (DGM) became mining concessions and there are no longer separate specifications for a mineral exploration or exploitation concession. A second change to the mining laws was that all mining concessions are granted 50 years provided the concessions remain in good standing. As part of this change, all former exploration concessions which were previously granted for 6 years became eligible for the 50-year term.

In 2006, Timmins concluded an access agreement with an agrarian community (an "ejido" in Mexico) called "Los Chinos" whereby Timmins was granted access privileges to 674 ha, the use of the ejido's roads, as well as being able to perform all exploration work on the area covered by the agreement. The agreement is for a period of 10 years with an option to extend the access beyond the 10 year period. In consideration for the ejido granting the access privileges to a portion of its land, Timmins paid the ejido the sum of USD 30,000.

Accessibility, Climate, Local Resources, Infrastructure and Physiography

The project is located in the Arizona-Sonora desert in the northern portion of the Mexican state of Sonora, 2 km west of the town of Estación Llano (Estación), approximately 150 km north of Hermosillo and 120 km south of the United States/Mexico border city of Nogales along Highway 15 (Pan American highway). The closest accommodations are located in Santa Ana, a small city located 21 km to the north on Highway 15.

Physiographically, the San Francisco property is situated within the southern Basin and Range Province, characterized by elongate, northwest-trending ranges separated by wide alluvial valleys. San Francisco is located in a relatively flat area of the desert with the topography ranging between 700 and 750 m above sea level.

The climate at the project site ranges from semi-arid to arid. The average ambient temperature is 21°C, with minimum and maximum temperatures of -5ºC and 50ºC, respectively. The average rainfall for the area is 330 millimetres (mm) with an upper extreme of 880 mm. The desert vegetation surrounding the San Francisco mine is composed of low lying scrub, thickets and various types of cacti, with the vegetation type classified as Sarrocaulus Thicket.

History

After conducting exploration on the project between 1983 and 1992, Compania Fresnillo S.A. de C.V. (Fresnillo) sold the property in 1992 to Geomaque Explorations Ltd. (Geomaque). After conducting further exploration, Geomaque decided to bring the project into production in 1995. Due to economic conditions, mining ceased and the operation

entered into the leach-only mode in November, 2000. In May, 2002, the last gold pour was conducted; the plant was mothballed, and clean-up activities at the mine site began.

In 2003, Geomaque sought and received shareholder approval to amalgamate the corporation under a new Canadian company, Defiance Mining Corporation (Defiance). On November 24, 2003, Defiance sold its Mexican subsidiaries (Geomaque de Mexico and Mina San Francisco), which held the San Francisco gold mine, to the Astiazaran family of Sonora and their private company for a total consideration of USD 235,000.

Since June, 2006, the Astiazaran family and their company Desarrollos Prodesa S.A. de C.V. have been extracting sand and gravel intermittently from both the waste dumps and the leach pads for use in highway construction as well as other construction projects.

Timmins conducted a review of the available data and started a reverse circulation drilling program in August and September, 2005. This was followed by a second drilling program comprised of both reverse circulation and diamond drilling in 2006, based on the results of the 2005 drilling program.

Geological Setting and Mineralisation

The San Francisco project is a gold occurrence with trace to small amounts of other metallic minerals. The gold occurs in granitic gneiss and the deposit contains principally free gold and occasionally electrum. The associated mineralogy, the possibility of associated tourmaline, the style of mineralization, and fluid inclusion studies suggest that the San Francisco deposits may be of mesothermal origin.

The San Francisco deposits are roughly tabular with multiple phases of gold mineralization. The deposits strike 60˚ west to 65˚ west, dip to the northeast, range in thickness from 4 to 50 m, extend over 1,500 m along strike and are open ended. Another deposit, the La Chicharra zone, was mined during the last two years of production, as a separate pit.

Exploration

After a review of the available geological data, Timmins identified a number of exploration targets in and around the existing San Francisco pit as well as some secondary targets located on Timmins exploration concessions. Timmins conducted its first exploration drilling program on the San Francisco project in August and September, 2005.

The drilling program conducted by Timmins from September to November, 2006 was based primarily on the results of the 2005 drilling program and on the results of the historical drilling programs conducted by Geomaque and Fresnillo.

During 2007, Timmins has conducted field work and exploration drilling to evaluate the extent of the gold mineralization in the other zones on the property, but no mineral resource

estimate from that work was available for inclusion in this study. A total of 5,123 m of exploration drilling were completed in 2007 which included 1,327 m of condemnation drilling. The total expenditures for the 2007 drilling program were approximately USD 629,000.

Drilling

In the fall of 2005, Timmins conducted a reverse circulation drilling program comprised of 14 holes on the San Francisco project site. The details and results of the drilling program were published previously in a Micon technical report entitled "Technical Report on the San Francisco Mine Property, Estación Llano, Sonora, Mexico." This technical report was published on the SEDAR website on April 28, 2005 by TMM.

Between September and November, 2006, Timmins conducted a follow-up drilling program which consisted of 56 holes comprised of 28 diamond drill holes and 28 reverse circulation drill holes. While the drilling was primarily concentrated to the north and northwest of the present San Francisco pit and to the north and northwest of the existing La Chicharra pit, a number of widely spaced holes were drilled to test specific geological and geochemical targets around the San Francisco pit and to the south and west of the La Chicharra and La Severiana areas.

The details of the 2006 exploration program and its results were discussed in a February, 2007 Micon technical report entitled "NI 43-101 Technical Report and Resource Estimate for the San Francisco Gold Property, Estación Llano, Sonora, Mexico". This report was filed on the SEDAR website on February 27, 2007 by TMM.

This report repeats the description of the Frebruary, 2007 resource estimate by IMC on the main San Francisco deposit, including the sampling methods, assaying, data quality control and verification procedures.

Mineral Processing and Testwork

The San Francisco mine operated during the period 1995-2000, when approximately 13.5 million tonnes of ore at a grade of 1.13 g/t gold was treated by heap leaching, and 300,834 oz of gold were recovered. Metal recovery from the pregnant solution was effected through carbon columns. Loaded carbon from the columns was transferred to a Zadra elution plant for precious metal extraction and the production of Doré bullion. Average gold recovery over the mine life was about 63%. Mining operations ceased in 2001 as a result of low gold prices, although leaching and rinsing of the heap continued for approximately one year after this.

In Section 9 of its 2006 San Francisco Scoping Study, Sol & Adobe Ingenieros Asociados S.A. de C.V. (Sol & Adobe) noted that:

- Norberg Inc. Minerals Research and Test Centre conducted tests to evaluate impact work indices and paddle abrasion index and that the results indicated that the material was characterized as very hard and abrasive.

- Metallurgical testwork was performed on the San Francisco project by four separate organizations, the first being Servicios Industriales Peñoles, S.A. de C.V. (Peñoles) of Monterrey, Mexico. Additional work was performed by McCelland Laboratories Inc. (McCelland), Kappes, Cassiday and Associates (Kappes Cassiday) both of Sparks, USA and METCON Research Inc., (METCON) of Tucson, USA. The testwork performed by Peñoles and McCelland included both roll and column leach tests. Work completed at Kappes Cassiday included bottle roll tests on drill cutting samples and column tests on bulk material crushed to minus 5/8 in. METCON conducted column tests on samples designated as schist and granite and representing four different crush sizes for each type of material.

- Independently of ore type, schist or granite, gold extraction averaged 57% for a leach cycle of 72 days. Cyanide consumption ranged from 0.0554 to 0.085 kg/t the lowest to 0.105 kg/t, the highest. Lime consumption averaged from 0.77 to 0.93 kg/t. In plant, reported cyanide consumption averaged 0.11 kg/t, whereas lime gave 0.96 kg/t, values considered very close to the ones reported in several metallurgical tests.

Timmins shipped four samples from the San Fransisco mine to Process Research Associates (PRA), in Richmond, B.C., representing the major ore types. Bottle roll testing was carried out on the ‘’as received’’ nominally ¾ in and ½ in material and a sample crushed to a P80 of ¼ in, considered to be the finest size that could practically be used for heap leaching. The finer crush increased extractions and it was decided that column testing should be carried out at this size.

Four columns with an internal diameter of 102 mm were were filled to 2.44 m, one column for each ore type. Each column contained 25 – 28 kg material, agglomerated with 0.05 kg/t of hydrated lime, 5 kg/t of cement and approximately 5% moisture. Leaching was carried out on the granite and gneiss samples for 75 days and gold extractions of 79.6% and 85.6% were achieved respectively. Cyanide and lime consumptions were 1.12 and 0.95 kg/t and 0.12 and 0.11 kg/t, respectively. The gabbro and pegmatite ore leaching was continued to 97 days and extractions of 65.8% and 90.2% were achieved for each, respectively. Reagent consumptions were similarly low with the exception of the pegmatite test in which 0.21 kg/t of lime was consumed.

PRA’s standard acid base accounting on four waste rock samples indicated that they could be classified as non-acid generating.

Timmins then conducted definitive leach testing at their on-site laboratory. The results of bottle roll tests were generally lower than those obtained during the PRA testwork and also

exhibited considerable variation. It was believed that the small sample size and presence of free gold may have had an impact on the variability, so it was decided to proceed with tests using the 2.3 m columns.

Six 2.3 m column tests were conducted, three on granitic material, and one each on partly altered granite, gabbro, and gneiss. All the tests were conducted on material crushed to 100% minus ½ in with the exception of test 146809-810 which was crushed to 100% minus ¼ in. Leach times varied from 44 to 66 days as the tests were stopped once the extraction kinetics had slowed and the curves flattened.

The granitic material crushed to minus ¼ in produced a lower extraction rate (56%), compared to the two tests at 100% minus half inch, which averaged approximately 75%. The partly altered granite produced an extraction of 55%, the gabbro and the gneiss both approximately 47%. Although recoveries were improved over the bottle roll tests, they were still considerably less than those achieved during the PRA testwork.

A final series of test carried out on site used columns 6.0 m tall with 140 mm ID. Due to time constraints, leaching was conducted for a maximum of 47 days only, but all samples were still leaching when the columns were shut down for washing and residue assaying. The results of these columns are given in Table 1.1, together with the weighting attributed to each ore type in the sectional geological model prepared by Timmins geological staff so that a weighted average recovery could be calculated for the in-pit resource. It should be noted that the forecast recoveries are given for material crushed to 100% minus ½ in and assume a secondary leaching cycle is conducted on the heap. Whilst the same recoveries could possibly be obtained from coarser material, the leach time required would be longer, and would therefore impact on the planned production rate.

Table 1.1
Actual and Forecast Leach Recovery

Domain name	Weight %	Actual Column Recovery after 47 Days (%)	Forecast Column Recovery after 90 Days (%)
Pegmatite	11.4	65.2	71.7
Gabbro	7.7	52.5	57.8
Gneiss	63.5	61.1	67.2
Granite	17.1	77.9	85.7
Granite/Gneiss (altered)	0.2	67.3	74.0
Overall weighted recovery		**63.7**	**70.1**

Mineral Resource Estimate

In 2006, IMC was engaged by TMM/Timmins to estimate the mineral resources for the San Francisco mine, using the historical Geomaque data along with the results of Timmins' 2005 and 2006 exploration drilling programs. IMC developed a three-dimensional (3D) block

model and used floating cone techniques to develop a mineral resource within a constrained pit outline.

The resource estimate completed by IMC in January, 2007, is compliant with the current CIM standards and definitions required by NI 43-101 and it supersedes the historical resource estimate for the San Francisco mine. To fulfill the criterion for potential economic viability, only material lying within a floating cone pit shell at a gold price of USD 500 per ounce and additional cost and recovery parameters developed by Timmins and IMC has been reported as a mineral resource in Table 1.2.

IMC does not know of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which would adversely affect the estimated mineral resources. However, the reader should be cautioned that mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1.2
IMC Mineral Resource Estimate for the San Francisco Project
(0.23 g/t Gold Cut-off Grade)

Category	Resource (000 t)	Grade (g/t gold)	Contained Gold (oz)
Measured Mineral Resource	5,352	0.912	156,930
Indicated Mineral Resource	22,296	0.781	559,860
Total Measured + Indicated Resources	**27,648**	**0.806**	**716,790**
Inferred Mineral Resource	2,506	0.788	63,490

Table provided by Independent Mining Consultants, Inc.

The drilling database provided to IMC by Timmins consists of 1,133 drill holes amounting to 116,000 m of drilling. There are 62,137 sample intervals of which 61,346 were assayed for gold. The sampling interval is predominantly 2 m (86% of the intervals), though about 7% of the intervals are 1.5 m in length, and about 3% of the intervals are 1 m in length.

IMC was also provided with a block model for the deposit. The provided model is based on 10 m by 10 m by 6 m high blocks and the coordinate limits and block size structure of the model were retained for this current work. The topographic surface was updated to reflect the surface at the termination of Geomaque's mining activity in 2002. An indicator kriging estimation method was used to facilitate the incorporation of new data into the block model. The estimation method is:

1) Block grade estimations are based on 3 m regular bench composites, even though the block height is 6 m. Assays were length weighted for each composite. The reason for the relatively short composite is to unsmooth the resultant block grade distribution to try to better match the likely distribution of mined blocks.

2) Assays were capped at 30 g/t gold prior to compositing.

3) Ore zones were established in the model by indicator kriging, in which a discriminator of 0.125 g/t was used. Composites greater than 0.125 g/t were assigned a value of 1 and composites less than 0.125 g/t were assigned a value of 0. The ones and zeros were kriged to obtain a value between 0 and 1 for each block that may be interpreted as the probability that the block is above 0.125 g/t gold. Blocks with a probability over 0.5 were assigned a code to designate them inside the ore zone, i.e. an ore block.

4) Composites were assigned an ore/waste code as follows. Composites greater than 0.125 g/t gold were marked as ore. Composites below 0.125 g/t gold, but located within ore blocks defined as 0.75 or more probability of being ore were also assigned a code to mark them as in the ore zone. This accounts for internal waste composites.

5) Gold grades were estimated by ordinary kriging. Only ore blocks were estimated and only the ore zone composites were used for the estimation. The search radius for the estimation was 50 m along strike (N65°W), 60 m down dip (30°NE) and 20 m in the perpendicular (near vertical) direction. A maximum of 10 and a minimum of 1 composite were used to assign grade with a limit of 2 composites per hole. The indicator kriging to develop the ore zone was based on a maximum of 10 and a minimum of 2 composites, again with a maximum of 2 per hole. The relatively small maximum number of composites was, again, to limit over-smoothing the grade distribution. The search radii used represent 100% of the estimated variogram ranges of the gold variograms.

6) Based on historic data, an in-situ block density of 2.66 t/m^3 was assigned for the ore blocks and 2.77 t/m^3 for the waste blocks.

7) A resource classification code (measured, indicated, and inferred resources) was also assigned to the model. More details on this assignment are included below.

To validate the resulting block model, IMC compared the model estimate of the contents of the existing open pit with a table of historic production from the San Francisco pit between 1996 and the mine closure in 2002. Table 1.3 compares the historic production with the new IMC model for this material, showing a good comparison between the new model and reported historic production.

Table 1.3
Comparison of the IMC Model with Historic Production for the San Francisco Project

Description	Tonnes (000 t)	Grade (g/t gold)	Gold (000 oz)
Historic Production	13,490	1.127	488.7
IMC Model at 0.4 g/t cut-off	13,707	1.113	490.5
% Difference	+1.6	-1.2	+0.037

Table provided by Independent Mining Consultants, Inc.

For resource classification purposes only, a special gold grade kriging was then done. The gold grade estimates from this kriging were not used, but the number of samples and kriging standard deviation for each kriged block were used to classify the resource.

1) The maximum search radii for the special kriging were set to 38 m along strike, 45 m down dip, and 20 m perpendicular. This represents about 75% of the variogram range; IMC generally assumes that measured/indicated resources should be defined within 67% to 75% of the variogram range. The variogram was also normalized to a sill of 1 and a nugget of 10% of the sill. A maximum of one composite per drill hole was allowed in the kriging and the ore zones used in the grade kriging were respected. This kriging procedure provides a count of the number of holes within 75% of the maximum search radius and also calculates a kriging standard deviation based on these data. The number of holes and kriging standard deviation were stored in the model.

2) Probability plots of block kriging standard deviations by the number of holes were constructed.

3) First, all blocks in the ore zone were set to a default of inferred resource. For blocks with the closest composite outside the 38 m by 45 m search, that is their final classification. They were not examined in the special kriging.

4) The plots of kriging standard deviations indicate that blocks estimated with five or more holes generally have a standard deviation less than 1.0. These were classified as indicated resource.

5) Blocks kriged with four holes and with a kriging standard deviation less than 1 were classified as indicated resource. This is about 99.5% of the blocks kriged with four holes. Blocks kriged with three holes and with a kriging standard deviation less then 0.9 were classified as indicated resource. This is about 82% of the blocks kriged with three holes. Blocks kriged with two holes and with a kriging standard deviation less than 0.8 were also classified as indicated resource. This is about 35% of those blocks. Blocks kriged with one hole and a kriging standard deviation less than 0.7 (about 4% of those blocks) were also classified as indicated resource.

6) All blocks with a kriging standard deviation less than 0.45 were then re-classified as measured resource. This is about 10% of all blocks kriged with the special kriging.

Visually, the described method appears to give good results. Indicated resources are not extrapolated far outside of the drilling data and measured resources are developed only in well-drilled areas. Blocks kriged with one or two holes can generate indicated resources only very close to the holes.

IMC then performed a floating cone analysis on the block model to constrain the reported mineral resource estimate to that material having demonstrated economic potential. Table 1.4 shows the economic parameters used for this analysis. The parameters are a combination of IMC and Timmins inputs. The mineral resource given in Table 1.2, above lies within the resulting optimized pit shell.

The USD 1.00 per total tonne owner mining cost was estimated by IMC. The processing and G&A costs were provided by Timmins. The G&A cost is based on a fixed cost of USD 600,000 per year and an ore production rate of 3 million tonnes per year. The gold recovery of 64% was a Timmins estimate. Historic recovery is reported at 61.5% while Geomaque operated the property. The slope angles used for the floating cone evaluation are based on inter-ramp angles recommended by Golder Associates and adjusted by IMC to allow for haul roads.

Table 1.4
Floating Cone Parameters for the San Francisco Project

Parameters	Costs, Etc.
Mining Cost per Tonne Mined	USD 1.00
Processing Cost Per Ore Tonne	
Crushing	USD 0.86
Leaching/Plant	USD 1.11
Maintenance	USD 0.12
Sales	USD 0.10
TOTAL	USD 2.19
G&A Cost Per Ore Tonne	USD 0.20
Gold Recovery	64%
Gold Price Per Troy Ounce	USD 500
Cut-off Grades	
Internal Cut-off (g/t)	0.23
Breakeven Cut-off (g/t)	0.33
Slope angles based on Golder Associates 1996 report	

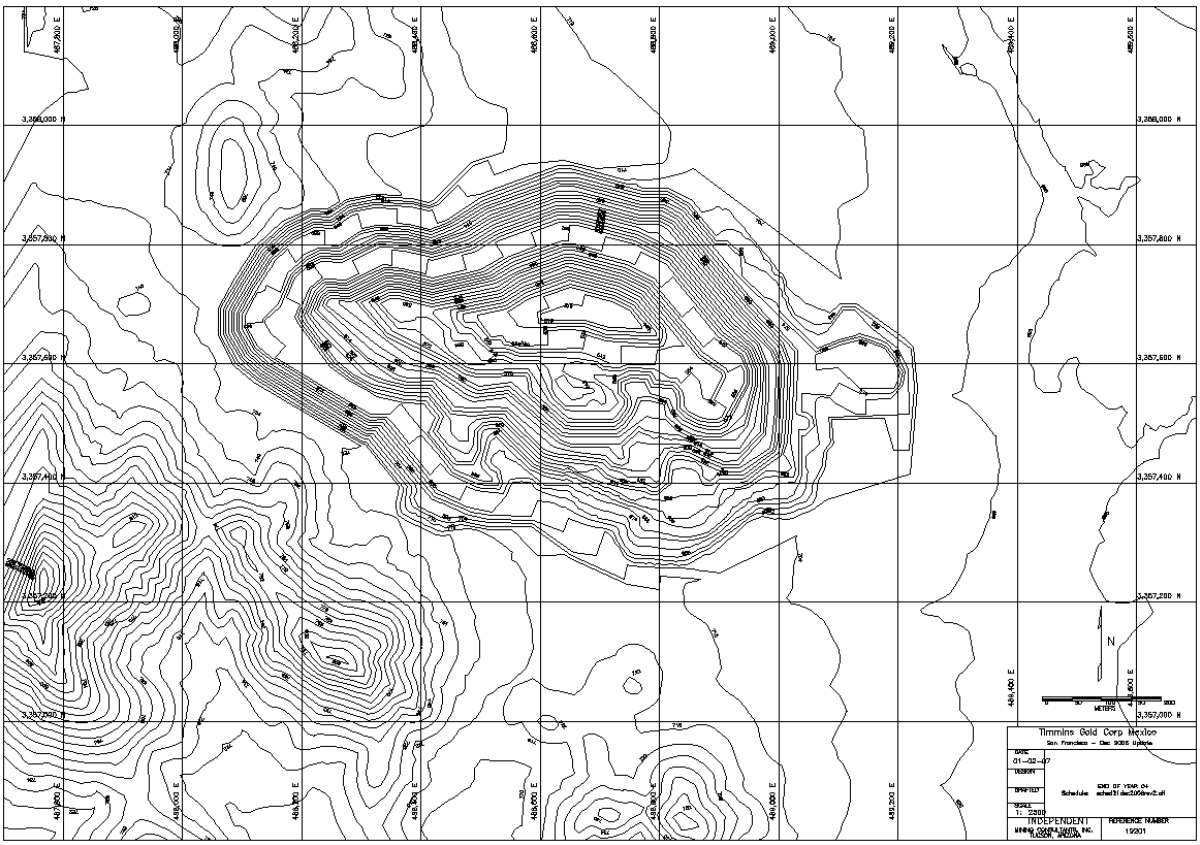
Table provided by Independent Mining Consultants, Inc.

Mineral Reserve Estimate

IMC then performed a second floating cone analysis on the block model, this time attributing no value to the inferred resource material in the block model. All other parameters were as given in Table 1.3, above. This optimized shell provided the basis for an open pit design including ramps and safety berms that could be used to delimit a mineral reserve estimate for the project. The ultimate open pit is shown in Figure 1.1

During the course of its work towards the preparation of this study, Micon has identified increases in some of the operating costs forecast for the project since the time at which IMC conducted its open pit optimization, design and production scheduling in 2006 and 2007.

Figure 1.1
Final Limits of San Francisco Open Pit



Most notable amongst these is the increase in mining costs per tonne, which have now been priced to include the intended mining contractor's cost of ownership as a cash cost, rather than maintaining the previous assumption of an owner-operated fleet where cost of ownership would have been considered as a capital expense. The revised average mining cost is USD 1.72 per tonne of material based on a mining contractor operation.

However, the rise in forecast gold recoveries and, most significantly, the higher forecast gold price, have off-set such changes, so that the cut-off grades used in the pit design are now seen to be conservative. Micon therefore considers that IMC's resource estimate remains valid. Table 1.5 compares the parameters used in pit optimization with those resulting from the completed preliminary feasibility study.

Modelling of the San Francisco deposit has been undertaken by IMC in a manner such that the resource tonnage and grade estimates already provide sufficient allowance for grade dilution and losses in mining recovery that no further modifying factors are required before reporting these as a mineral reserve. The comparison to prior production given in Table 1.3 (above) bears out this assumption.

Table 1.5
Comparative Parameters for the San Francisco Project

Parameters	Resource Estimate Jan-2007	Preliminary Feasibility Study	Preliminary Feasibility Study
Processing Method	**Crusher Feed**	**Crusher Feed**	**Low Grade (ROM)**
Mining Cost per Total Tonne	USD 1.00	USD 1.72	USD 1.72
Processing Cost Per Ore Tonne			
Crushing	USD 0.86	USD 0.86	nil
Leaching/Plant	USD 1.11	USD 0.87	USD 0.87
Maintenance	USD 0.12	USD 0.12-	USD 0.12
Sales	USD 0.10	USD 0.23	USD 0.23
TOTAL	USD 2.19	USD 3.80	USD 3.80
G&A Cost Per Ore Tonne	USD 0.20	USD 0.30	USD 0.30
Gold Recovery	64%	70%	40%
Gold Price Per Troy Ounce	USD 500	USD 686	USD 686
Cut-off Grades			
Internal Cut-off Grade (g/t)	0.23	0.15	0.10
Breakeven Cut-off Grade (g/t)	0.33	0.27	0.21

Table compiled by Micon. Data for Resource Estimate column provided by Independent Mining Consultants, Inc.

On the basis that the information presented in Section 18 of this report, economic viability of the proposed extraction and treatment of the portion of the measured and indicated mineral resource found within the designed open pit has been demonstrated.

Micon therefore considers the measured and indicated mineral resources within the open pit design to be a mineral reserve in terms of the CIM definitions.

Notwithstanding the fact that engineering of certain aspects of the project have reached an advanced stage, Micon considers that, overall, the basic project engineering is at the level normally associated with a preliminary feasibility study. In this study, no break-down of the resources by rock type and recovery on a bench-by-bench or annual basis was available. Only a manual, sectional estimate was available, providing a life-of-mine average. This resulted in "*uncertainties associated with the modifying factors*" in terms of NI 43-101, and hence Micon has classified both the Measured and Indicated Mineral Resources within the pit as a Probable Mineral Reserve.

Table 1.6 sets out the Mineral Reserves of the San Francisco Project. These reserves are valid as of February 29, 2008. Note that in addition to the Mineral Reserve tonnage, total waste rock within the final pit outline is estimated to be 46.0 Mt, giving a stripping ratio (waste:ore) ratio of 2.0:1.

Table 1.6
Mineral Reserve Estimate

Case	Reserve Class	Cut-off Grade (g/t gold)	Reserve (000 t)	Grade (g/t gold)	Gold (000 oz)
High Grade Crusher feed	Probable	0.50	12,000	1.05	403.7
Low Grade Crusher feed	Probable	0.23	4,653	0.88	132.0
Sub-total Crusher feed	Probable		16,653	1.01	535.7
Low Grade ROM leach	Probable	0.28	5,981	0.39	75.3
Grand Total	**Probable**		**22,634**	**0.84**	**611.0**

Compiled by Micon from schedules provided by Independent Mining Consultants, Inc.

Mining Production Schedule

IMC prepared a production schedule for the open pit, generating a crusher feed of 4.0 Mt/y of higher grade material using a cut-off grade 0.50 g/t gold, together with lower grade material above a cut-off of 0.23 g/t gold which was to be heap leached without crushing. This production schedule has been adjusted by Timmins to reflect a slightly slower production rate of 3.3 Mt/y of crusher feed, in order that the lower grade material can be leached concurrently with the higher grade, crushed ore. Micon's review confirms that the Timmins schedule honours the total bench-by-bench volumes and grades of material to be moved, as delineated in the IMC schedule.

Table 1.7 shows the production schedule for the main zone.

Table 1.7
Mine Production Schedule

	Unit	Pre-production	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Crush-Leach Ore	000 t	464	3,300	3,300	3,300	3,300	2,989	16,653
	g/t Au	0.883	1.023	1.091	1.054	0.972	0.868	1.001
ROM Ore	000 t	447	1,784	1,789	1,367	594	0	5,981
	g/t Au	0.383	0.391	0.393	0.393	0.393		0.392
Total Ore	000 t	911	5,084	5,089	4,667	3,897	2,989	22,634
	g/t Au	0.638	0.801	0.845	0.860	0.884	0.868	0.840
Waste	000 t	4,804	12,647	10,733	8,323	6,045	3,507	46,059
Total Tonnes	000 t	5,715	17,731	15,822	12,990	9,939	6,496	68,693
Strip ratio		5.3	2.5	2.1	1.8	1.6	1.2	2.0

All mining activities will be carried out by a mining contracting company. The contractor will provide all the required mining equipment and personnel to produce the tonnes scheduled in Table 1.7. Timmins will provide contract supervision, geology, engineering and planning and survey services using its own employees.

Processing

The gyratory primary crusher remains in place but requires some fairly extensive repair work to rebuild worn areas. Estimates for the rebuild work have been obtained from Metso, and have been incorporated in the financial model. There is no surge bin, feeder or impact idlers under the primary crusher as would normally be installed to protect the conveyor belts and provide a uniform feed to the following equipment. However, it was decided that the configuration should remain the same as before.

Secondary crushers were part of the original circuit but these were dismantled as were the 'grasshopper' conveyors for feeding ore onto the heaps. Much of the conveyor hardware for feeding the leach pads is on site, although the belts have been removed. For the revamped plant, Timmins intends to install secondary and tertiary crushing, with the latter in closed circuit. Sandvik have provided a detailed proposal for the refurbishment of the secondary and tertiary crushing and screening plant, which has been incorporated into the capital estimate

It is Timmins intention to construct a new 47 ha leach pad adjacent to the existing pads. Phase one of the new leach pad will occupy 25 ha and be divided into 4 sections to facilitate control of pregnant liquor concentrations. Phase 2, which is comprised of the remaining 22 ha, will be constructed in the third year of operation. The new pads will require some re-grading of the existing terrain to ensure correct solution flow to the elution plant.

The adsorption plant consists of 2 lines of carbon columns each with 5 tanks through which the carbon is advanced counter currently. One line of columns contains approximately 2.0 tonnes of carbon per column and the other 2.5 tonnes per column. Gold is adsorbed on the carbon to a concentration of approximately 5,000 g/t. Desorption of the carbon is achieved in a Zadra circuit using stainless steel electrodes in a stainless steel electrolytic cell. Adequate carbon regeneration and handling facilities are installed and should require only a minimum of rehabilitation work to return them to service.

In total, the processing and maintenance staff will require 116 people. It is intended to operate two 12 hour shifts, seven days per week, a four day on/four day off schedule thus requiring four crews of up to 29 persons each; on day shift, there will be a need for heap leach solution piping movements and also additional people in the laboratory. Overall the forcast labour is considered to be higher than might be required once the plant has reached stable operating state.

Table 1.8 indicated the annual production schedule for the processing plant.

Table 1.8
Annual Process Production Schedule

		2008	2009	2010	2011	2012	2013	TOTAL
High Grade Ore treated	000 t	464	3,300	3,300	3,300	3,300	2,989	16,653
	g/t Au	0.88	1.02	1.09	1.05	0.97	0.87	1.00
	Au (kg)	410	3,376	3,600	3,477	3,207	2,594	16,664
Gold recovery	%	70.00	70.00	70.00	70.00	70.00	70.00	70.00
	Au (kg)	287	2,363	2,520	2,434	2,245	1,816	11,665
Low Grade Ore treated	000 t	447	1,784	1,789	1,367	594	-	5,981
	g/t Au	0.38	0.39	0.39	0.39	0.39	-	0.39
	Au (kg)	171	698	703	537	233	-	2,342
Gold recovery	%	40.00	40.00	40.00	40.00	40.00	40.00	40.00
	Au (kg)	68	279	281	215	93	-	937
Total Ore Heaped	000 t	911	5,084	5,089	4,667	3,894	2,989	22,634
Gold production	Au (kg)	355	2,642	2,801	2,649	2,339	1,816	12,602
Gold inventory	Au (kg)	44	113	120	114	100	-	
Gold sales	Au (kg)	311	2,573	2,794	2,656	2,352	1,916	12,602
Gold sales	**Au (oz)**	**9,995**	**82,734**	**89,829**	**85,377**	**75,615**	**61,598**	**405,148**

Manpower

Table 1.9 indicates the total manpower proposed for the San Francisco project, excluding employees provided by the mining contractor and the commercial security personnel.

Table 1.9
Manpower structure

Description	Number
General Manager	1
Accounting	6
Purchasing	4
Human Resources	14
Subtotal G&A	**25**
Environmental	3
Geology	7
Mining	5
Subtotal Technical services	**15**
Crushing	42
Maintenance	43
Adsorption/Desorption/Recovery	18
Laboratory	13
Subtotal Processing Department	**116**
Grand Total	**156**

Environmental and Social Considerations

Timmins is in the process of renewing its permits for the mine operations. In order to comply with the Mexican legal and administrative framework, the following laws and regulations must be considered:

- Mining Act and Regulations (LM)
- General Law of Ecological Equilibrium and Environmental Protection (LGEEPA)
- General Law of Sustainable Forestry Development (LDFS)
- National Waters Law (LAN)
- General Law for Integral Waste Management and Prevention (LGPGIR)

As a nation, Mexico is a signatory to a significant number of International Environmental Treaties. Additionally, in all the Commercial Treaties to which Mexico is a party, environmental regulations are included.

- Vienna Convention for the Protection of the Ozone Layer, Vienna 1986 and Montreal Protocol Related to the Substances that Deplete the Ozone Layer, Montreal 1987.
- Convention on the Control of Transboundary Movement of Hazardous Wastes, Basel 1989.
- UN Convention on Climate Change, New York 1992.
- Convention on Biological Diversity, Rio de Janeiro 1992.
- Convention on International Trade in Endangered Species of Flora and Fauna, Washington 1973.
- North America Free Trade Agreement, side agreement on environmental matters, 1993

Permitting is expected to be completed early in 2008 as indicated in the work schedule submitted as part of the MIA to government (Figure 1.1). Timmins' application for a Change of Land Use permit was approved by SEMARNAT on February 22, 2008. This key permit for the project is supported by a compensation payment from Molimentales del Noroeste de S.A. de C.V. (Molimentales) that is to be spent on reforestation and restoration activities.

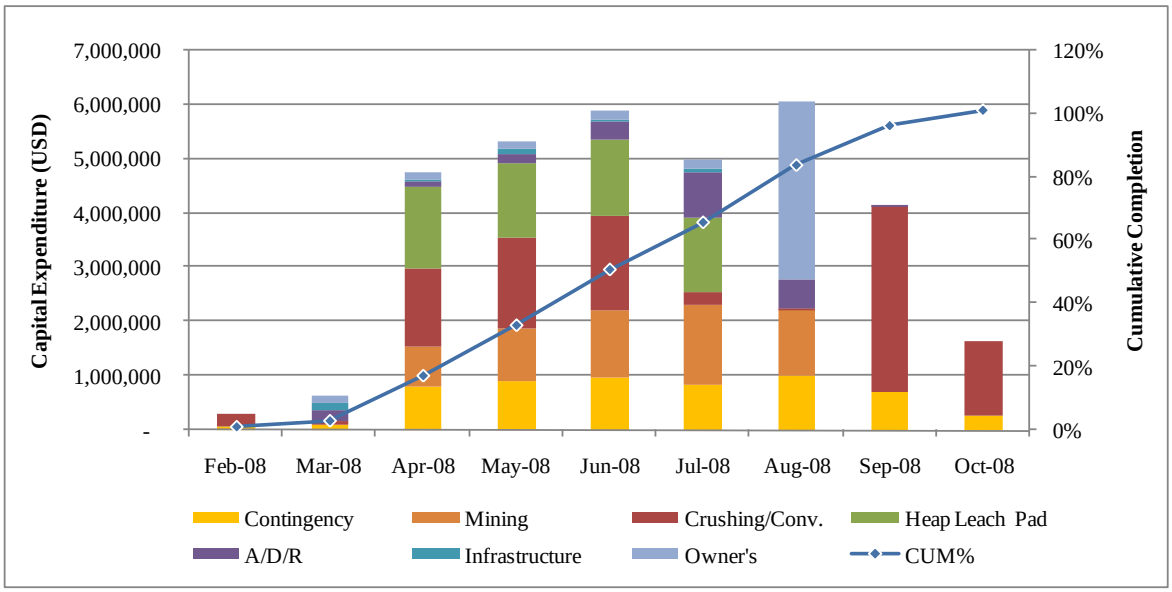
Figure 1.1
San Francisco Mine Work Schedule

Work Program	2008					
	JAN	FEB	MAR	APR	MAY	JUN
Obtain Authorizations	▬	▬				
Acquisition of Parts and Equipment	▬	▬	▬			
Maintenance and Refurbishment of Process Equipment		▬	▬	▬		
Exploration Program		▬	▬	▬	▬	▬
Preparation of Mine and Leach Pad Areas			▬	▬	▬	▬
Start of Operations						▬

Implementation Schedule

The project construction and implementation schedule, represented by capital expenditure, is shown in Figure 1.2.

Figure 1.2
Project Construction Schedule



The project implementation schedule calls for the completion of heap leach pad construction by mid-2008, in order that loading of the pad can commence in the third quarter, and leaching can commence at the end of that quarter. Refurbishment of the crushing plant must therefore be completed around mid-year. Expenditures shown later than this in the schedule represent deferred payments in respect of the work carried out on the secondary crushing and conveying systems by Sandvik. The pumping, irrigation and ADR plant must also be commissioned during the third quarter.

Such a timetable for development is ambitious, though Micon notes that refurbishment of the ADR plant was well advanced at the time of its visit in 2007. Critical path items are seen to be pre-stripping of the open pit and construction of the leach pad and irrigation/ADR systems. While the crushing system is important to the achievement of planned levels of recovery, it would be possible to commission the leach pad with uncrushed ROM ore if the crushing system fell behind schedule.

Capital Expenditure

The overall initial capital estimate for the project is given in Table 1.10.

Table 1.10
Initial Capital Cost Estimate
(USD 000)

	Total
Mining	5,675
Primary Crusher Rehabilitation	759
Secondary Crushing	9,483
Leach Pad Construction	5,641
Adsorption Plant Rehabilitaion	2,171
Infrastructure	406
Indirect Costs	4,006
Contingency	5,628
Total Initial Capital	**33,769**

Mining capital costs represent the principal mining contractor establishment charges, which are needed in order to refurbish and equip the existing workshop structure. Pre-production mining charges are included for pre-stripping of waste rock.

Light vehicles and equipment for the owner's supervisory and technical services teams are included under infrastructural capital, below.

Cost estimates for the primary and secondary crushing circuits have been based on supplier quotes for all of the major work items. Similarly, the leach pad construction cost has been based on a bill of quantities taken from Golder's design for the pad and contractor's quoted unit rates. The ADR plant work has been undertaken by a local contractor for a known cost rate.

The contingency of 20% reflects appropriately the level of engineering work carried out to date. It is assumed that all Impuesto al Valor Agregado (IVA, or VAT) will be recouped, and is therefore not provided for in the cost estimate.

Sustaining capital expenditure is given in Table 1.11.

Table 1.11
Life-of-Mine Sustaining Capital Expenditure
(USD 000)

	Total
Mining (demobilization)	1,200
Crushing/Screening/Conveying	2,048
Leach Pad Ph 2 – pad construction	4,840
Leach Pad Ph 2 - solution distr./recov.	440
ADR plant	434
Infrastructure	81
Closure/ Redundancies	3,616
Total Sustaining Capital	**12,659**

Phase 2 of the leach pad construction has been provided for at a unit rate 10% higher than that in Phase 1, to allow for increased cut and fill requirements. Additional pumps, pipes, valves and dripper lines are also allowed for.

A provision has been made for unspecified ongoing capital replacement expenditure in the crushing, screening and conveying plant, to cover any additional heap construction, conveying equipment needs, and/or replacement of existing equipment additional to the working cost allowance for maintenance. This allowance has been set at 4% per year of the initial capital cost for this plant.

Provision has been made for working capital represented by 7 and 30 days of accounts payable and receivable respectively, as well as 45 days stores and 30 days of product inventory.

Operating Costs

Table 1.12 shows a summary of the project operating costs over the life of the mine, and as a cost per tonne of ore heaped, and per ounce of gold sold. Note that only high grade ore is crushed before it is is placed on the leach pad. Lower grade ore is heaped uncrushed (i.e., as run-of-mine ore), and so the costs applicable to each are shown separately in the table.

Table 1.12
Cash Operating Costs

	LOM Total USD 000	High Grade Ore USD/t	Low Grade Ore USD/t	LOM Average USD/t	LOM Average USD/oz Au
G&A	6,821	0.30	0.30	0.30	16.84
Mining	113,070	4.97	5.06	5.00	279.08
Crushing	22,161	1.33	-	0.98	54.70
Processing	19,736	0.87	0.87	0.87	48.71
Laboratory	3,686	0.16	0.16	0.16	9.10
Social & Envir. Mgmt	1,677	0.07	0.07	0.07	4.14
Total	167,152	7.71	6.47	7.38	412.57

Revenue

Revenue projections are based on a gold price which is forecast to decline in real terms from the 2008 price level, towards a lon-term median price. The average price of USD 850/oz for 2008 is taken from the median of analysts forecasts published in early 2008 by the London Bullion Market Association (LBMA). The long-term median price used is USD 522.39/oz, being calculated from annual average prices in real terms for the period 1995 to 2007.

For the base case, this results in a weighted average gold price of USD 686.63/oz before royalties and refining charges.

Cash Flow Projection

Table 1.13 shows a summary of the project LOM undiscounted cash flow and the discounted cash flow valuation at discount rates of 5%, 10% and 15% per year. It will be seen from the above that the net present value of the project cash flow at a discount rate of 10%/y (NPV_{10}) for the base case evaluates to a approximately USD 25.5 million, and that the cash flow has an IRR of 38.5%. The average cash cost of production equates to USD 412.57/oz gold, or USD 7.38 per tonne treated (including uncrushed ROM material leached).

Figure 1.3 shows graphically the base case cash flow. Using an annual discount rate of 10%, the discounted cash flow has a payback period of 2.5 years, or approximately half the LOM.

Table 1.13
Summary of Project Base Case Discounted Cash Flow

IRR 38.5%	USD 000	LOM Total (Undisc)	NPV_5	NPV_{10}	NPV_{15}	USD per tonne	LOM Average USD/oz
Revenue	Gross Sales	278,188	241,900	212,654	188,763	12.29	686.63
less	Refining charges	(709)	(613)	(536)	(473)	(0.03)	(1.75)
less	Bullion delivery	(2,334)	(2,047)	(1,818)	(1,632)	(0.10)	(5.76)
less	Royalty	-	-	-	-	-	-
	Net Sales Revenue	275,145	239,240	210,300	186,658	12.16	679.12
Cash op.costs	G&A	6,821	5,921	5,200	4,614	0.30	16.84
	Mining	113,070	99,032	87,640	78,273	5.00	279.08
	Crushing	22,161	19,118	16,688	14,720	0.98	54.70
	Processing	19,736	17,174	15,111	13,426	0.87	48.71
	Laboratory	3,686	3,202	2,814	2,498	0.16	9.10
	Social & Env.Mgt	1,677	1,456	1,278	1,134	0.07	4.14
	Total	167,152	145,903	128,731	114,665	7.38	412.57
Net Cash Operating Margin		107,994	93,337	81,570	71,993	4.77	266.55
Capital Exp.	Initial/exp. capital	33,769	33,359	32,974	32,609	1.49	83.35
	Sustaining capital	12,659	10,651	9,079	7,827	0.56	31.25
Change in Working Capital		-	569	979	1,277	-	-
Net cash flow before tax		61,565	48,757	38,538	30,280	2.72	151.96
Taxation payable		17,313	14,969	13,072	11,518	0.76	42.73
Net cash flow after tax		**44,252**	**33,788**	**25,466**	**18,761**	**1.96**	**109.22**

Sensitivity Study

Sensitivity of the NPV_{10} to changes in gold price, operating and capital costs has been analyzed (Figure 1.4). While project revenues are directly proportional to process recovery and grade, project operating costs are largely insensitive to these factors, and hence gold price may also be used as a proxy in this model both for changes in recovery and reserve grade as revenue drivers.

Figure 1.3
Base Case Cash Flow Profile

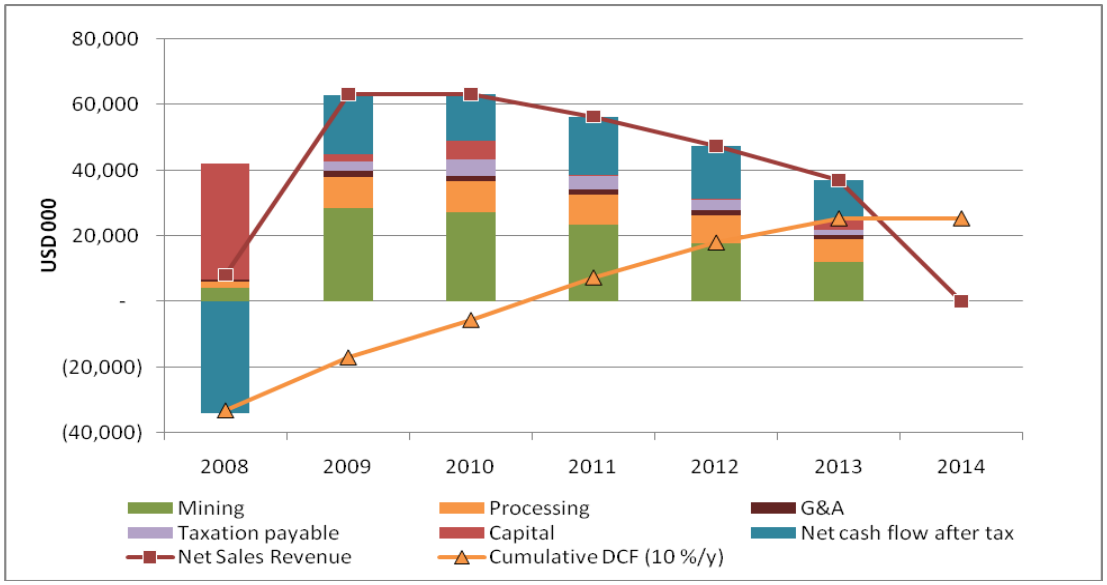
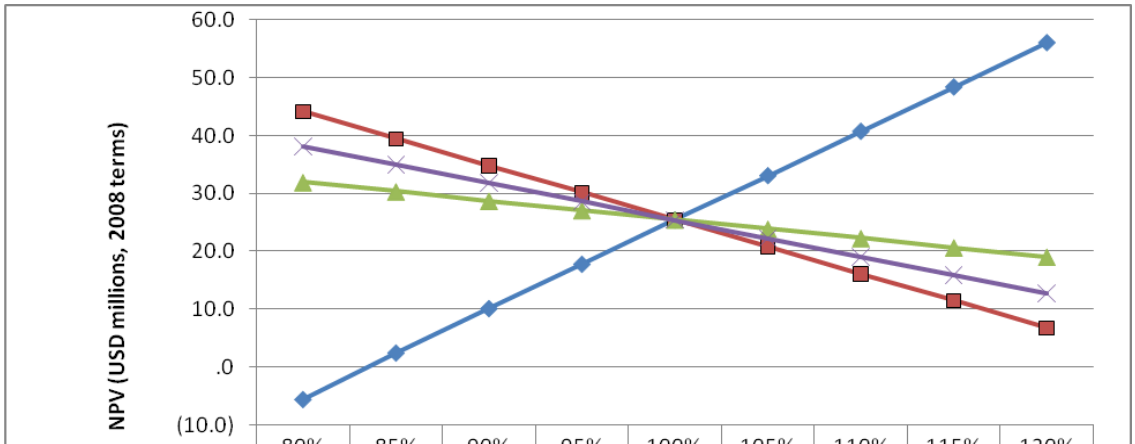


Figure 1.4
Sensitivity of NPV to Prices, Operating and Capital Costs



	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue drivers	(5.5)	2.5	10.2	17.8	25.5	33.1	40.8	48.4	56.1
Operating costs	44.1	39.5	34.8	30.1	25.5	20.8	16.1	11.5	6.8
Capital costs	31.9	30.3	28.7	27.1	25.5	23.8	22.2	20.6	19.0
Mining cost	38.2	35.0	31.8	28.6	25.5	22.3	19.1	15.9	12.7

Taking as the starting point the base case NPV_{10} evaluation of USD 25.5 million, the sensitivity results show that, as expected, the project is most sensitive to the revenue drivers described above. An adverse change of 20% is just sufficient to produce a slightly negative

NPV_{10}. The project is also moderately sensitive to operating costs, with a 20% adverse change sufficient to reduce NPV_{10} to less than USD 7 million.

Mining costs are a very significant portion of total operating costs, and the sensitivity to mining costs is evidence of the need to carefully manage the mining contractor under the proposed project implementation.

The project is least sensitive to project capital costs, with a 20% adverse change reducing NPV_{10} by around USD 6.5 million to USD 19.0 million.

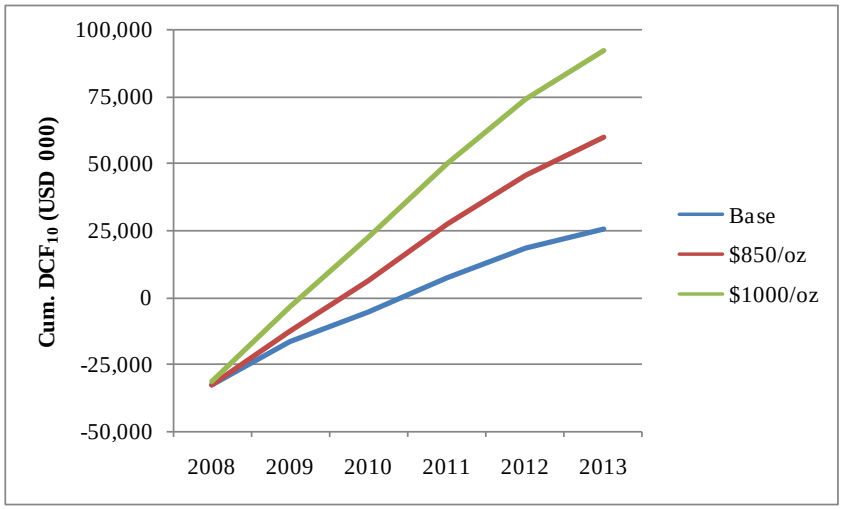
Two further sensitivity studies were carried out to reflect more optimistic gold price scenarios, wherein the gold price is maintained at (i) USD 850/oz and (ii) USD 1,000/oz over the LOM period. These scenarios reflect maintenance of the LBMA forecast average for 2008, and maintenance of prices recently achieved in the spot market, respectively.

Table 1.14 summarizes the results of this sensitivity study, which are also presented graphically in Figure 1.5. These results demonstrate the increase in IRR and NPV_{10} and the shortening of the discounted pay-back period achievable with a higher gold price.

Table 1.14
Summary of Results for Price Sensitivity Study

	Pre-Tax		After Tax		Discounted Payback Period (y)
	IRR (%)	NPV_{10} (USD 000)	IRR (%)	NPV_{10} (USD 000)	
Base Case	51.7	38,538	38.5	25,466	2.43
USD 850/oz	86.7	85,981	66.2	59,605	1.68
USD 1,000/oz	129.2	131,797	97.7	92,622	1.15

Figure 1.5
Comparison of Discounted Cash Flows for Gold Price Scenarios



Interpretation and Conclusions

Micon has reviewed the results of the historical exploration programs and the results from the, 2005 and 2006 drilling programs conducted by Timmins and, in light of the observations made in the Conclusions and Recommendations Section of this report, supports the concepts outlined by Timmins for further development of the project towards production. It is Micon's opinion that the property also merits further exploration, that Timmins' proposed exploration plans are properly conceived and justified, and that the work program and budget proposed by Timmins for the next phase of exploration are appropriate and warranted by the exploration results achieved to date.

Micon concurs with the intention of Timmins to bring the project into production and recommends that:

1) Timmins proceed with development of the San Franscisco open pit mine, crushing, heap leaching and gold recovery plant as described in the preliminary feasibility study summarized in this report;

2) Timmins should prepare bench-by-bench estimates for the distribution of rock types comprising the Mineral Reserve within the open pit which can be used to improve the precision of the annual leaching recovery forecasts;

3) Prior to construction of the second phase of its heap leach pad, Timmins should investigate the potential for further expansion of the open pit towards the north-west, near the proposed leach pad area. Micon considers that this would involve both exploration drilling and engineering studies to further delimit the open pit potential using a gold price forecast prepared at that time;

4) Timmins continues to compile the San Francisco data into a single database for the property which will assist in preparing further computer-generated resource estimates; and to document and review the general QA/QC program that has been set-up within the company and apply this program in all future exploration programs;

5) In future exploration drilling, all gold assays should be duplicated and samples which show a range of assays greater than 10% should be assayed by the screen metallics procedure.

Given the amount of work conducted previously at the San Francisco project on the known exploration targets and areas of mineralization, the property should be regarded as an advanced-stage exploration project with significant economic potential.

2.0 INTRODUCTION AND TERMS OF REFERENCE

At the request of Mr. Arturo Bonillas, President and Mr. Bruce Bragagnolo, CEO of Timmins Gold Corp. (TSX-V:TMM) (TMM), Micon International Limited (Micon) has been retained to provide a preliminary feasibility study for the San Francisco gold project located in the state of Sonora, Mexico. TMM advises that it holds its interest in the San Francisco property through its wholly-owned Mexican subsidiary Timmins Goldcorp Mexico, S.A. de C.V. (Timmins) which holds two mining concessions, three exploration concessions, and seven exploitation concessions through a wholly-owned subsidiary, Molimentales.

The study is based on the resource estimate prepared in 2007 by Independent Mining Consultants Inc. (IMC), which in turn was based upon the results of drilling conducted in 2005 and 2006 by Timmins, as well as drilling conducted previously by Geomaque Explorations Ltd. (Geomaque). IMC was retained by Timmins to update/develop a resource block model and estimate the mineral resources for the San Francisco project in compliance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards and definitions required by Canadian National Instrument 43-101 (NI 43-101) "Standards of Disclosure for Mineral Projects". That report, entitled "NI 43-101 Technical Report and Resource Estimate for the San Francisco Gold Property, Estación Llano, Sonora, Mexico" was dated February 23, 2007 and published on SEDAR February 27, 2007.

IMC later developed a preliminary open pit design and 4.0 Mt/y production schedule for the deposit, which provided a basis from which Timmins staff have developed the mining and processing schedules, and the capital and operating cost estimates used in this study. Metallurgical testwork was carried out by Process Research Associates (PRA), of Richmond, B.C., and also by Timmins staff at the project site. Micon has reviewed the engineering and environmental work carried out on the project, and has accepted that as meeting the standards normally expected of a preliminary feasibility study in terms of NI 43-101. This report presents the results of that work, together with Micon's analysis of the project economics, its conclusions with respect to project viability, and its recommendations regarding project development.

All currency amounts are stated in US dollars (USD) or Mexican pesos (MXN), as specified, with costs and commodity prices typically expressed in US dollars. Quantities are generally stated in metric units, the standard Canadian and international practice, including metric tons (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, grams (g) and grams per metric tonne (g/t) for gold and silver grades (g/t Au, g/t Ag). Wherever applicable, imperial units have been converted to Système International d'Unités (SI) units for reporting consistency. Precious metal grades may be expressed in parts per million (ppm) or parts per billion (ppb) and their quantities may also be reported in troy ounces (ounces, oz), a common practice in the mining industry. A list of abbreviations used in this report is provided above.

The qualified persons responsible for the preparation of this report are Mr. William J. Lewis, P.Geo., Mr. Michael G. Hester, FAusIMM, Mr. R. James Leader, P.Eng., Mr. Ian R. Ward, P.Eng., and Mr. Christopher A. Jacobs, CEng MIMMM.

Mr. Lewis, a Senior Geologist with Micon, is responsible for the independent summary and review of the 2006 exploration on the San Francisco project and the comments on the propriety of Timmins' 2006 exploration drilling program and the plans and budget for the next phase of exploration. Mr. Lewis is also the author or co-author of previous NI 43-101 Technical Reports on the project (Lewis, 2005, and Lewis and Hester, 2007). Mr.Lewis conducted a visit, his second, to the San Francisco property on September 10, 2006, with the assistance of Miguel Angel Soto y Bedolla, Chief Operating Officer of Timmins and Daniel Maya, a property geologist at the San Francisco mine. The purpose of the site visit was to review both the progress of the drilling program and the Quality Assurance/Quality Control (QA/QC) program instituted by Timmins.

Mr. Hester, a Vice President and Principal Mining Engineer with IMC, is responsible for the independent update/development of the block model and the resource estimate for the San Francisco project. Mr. Hester conducted his most recent site visit to the property on February 14, 2007. The main purpose of the visit was to examine the condition of the San Francisco and La Chicharra pits, and also the condition of the San Francisco waste dumps which will be expanded if the operation is re-started. Mr. Hester is co-author with Mr. Lewis of a previous NI 43-101 Technical Report on the property (Lewis and Hester, 2007).

A multi-disciplinary team from Micon comprising Mr. Leader, Senior Mining Engineer, Mr. Victor Bryant, IEng, AIMMM, Vice President and Senior Metallurgist, Ms. Jenifer Hill, R.P.Bio., Environmental Scientist, and Mr. Jacobs, Senior Consultant mineral economist and project manager, visited the site during the period September 25-28, 2007. Mr.Leader has reviewed the mine planning and production scheduling carried out by Timmins, and also the terms of the proposed mining contract and related cost estimates. Mr. Jacobs reviewed the capital and operating cost estimates for the project and compiled the discounted cash flow analysis and sensitivity studies used to evaluate project economics. Mr. Jacobs also supervised the work of Ms. Hill, and has taken responsibility for that work.

Mr. Ian R. Ward, P.Eng., President of Micon, has reviewed and taken responsibility for the work of Mr. Bryant, who undertook the site visit and has compiled descriptions of the metallurgical testwork and the crushing, heap leaching and adsorption/desorption/recovery (ADR) processes.

The review of the San Francisco project was based on published material researched by Micon, as well as data, professional opinions and unpublished material submitted by the professional staff of Timmins or its consultants. Much of these data came from reports prepared and provided by Timmins. IMC based its block model on the drilling database and the block model for the deposit provided by Timmins.

3.0 RELIANCE ON OTHER EXPERTS

Micon has reviewed and analyzed data provided by Timmins, its consultants and the previous operator of the project, and has drawn its own conclusions therefrom, augmented by its direct field examination. Micon has not carried out any independent exploration work, drilled any holes or carried out an extensive program of sampling and assaying on the property. However, previous sampling (Lewis, 2006) was conducted to independently substantiate the mineralization at the San Francisco project and further samples were not obtained during the 2006 and 2007 site visits.

To complete the resource estimate on the San Francisco project IMC was provided with both the drilling database and a block model for the deposit by Timmins. The exact source of the block model is uncertain but it could be the model IMC developed as part of its work for Geomaque in 1997. However, there has been a substantial amount of new drilling conducted by both Geomaque and Timmins since the 1997 model was developed and this material as well as an update of the topographic surfaces to reflect the end of Geomaque's mining activity in 2002 was added to the model.

While exercising all reasonable diligence in checking, confirming and testing it, Micon has relied upon Timmins' presentation of the project data, including data from the previous operator, in formulating its opinion with respect to the San Francisco property.

The English translations for the various agreements under which TMM and its wholly-owned Mexican subsidiary, Timmins, hold title to the mineral concessions for this project have been reviewed by Micon. Micon, however, offers no legal opinion as to the validity of the mineral title claimed. A description of the property, and ownership thereof, is provided for general information purposes only. A legal opinion regarding the mineral concessions has been provided to Micon by Timmins and Micon has relied on this expert opinion as Timmins validation regarding title to the mineral concessions. The legal opinion regarding the mineral title was undertaken by Roberto Herrera Piñon, a mining lawyer and master in corporate law, located in Hermosillo, Mexico. The legal opinion regarding the mineral concessions was dated April 26, 2007.

Micon has relied upon a legal opinion dated April 25, 2007 and prepared by Francisco Manuel Cordova Celaya, Dr. En Derecho regarding the transfer of the San Francisco project to Molimentales and Timmins. The legal opinion was conducted and executed in Hermosillo, Mexico.

The existing environmental conditions, liabilities and remediation have been described where required by NI 43-101 regulations. However, it should be noted that these statements are provided for information purposes only and Micon offers no opinion in this regard.

The description of geology, mineralization, exploration and mineral resource estimation methodology used in this report are taken from reports prepared by various companies or their contracted consultants, as well as from various government and academic publications. The conclusions of this report rely in part on data available in published and unpublished reports supplied by the companies which have conducted the exploration on the property, and information supplied by Timmins. The information provided to Timmins was supplied by reputable companies and Micon has no reason to doubt its validity.

Micon and IMC are pleased to acknowledge the helpful cooperation of Timmins management and consulting field staff, all of whom made any and all data requested available and responded openly and helpfully to all questions, queries and requests for material.

4.0 PROPERTY DESCRIPTION AND LOCATION

Timmins' San Francisco property is located in the north-central portion of the Mexican state of Sonora, which borders on the American state of Arizona, and is approximately 150 km north of the city of Hermosillo, which is the capital of Sonora. The latitude and longitude for the project site are approximately 30°21'13" N, 111°06'52" W. The UTM coordinates are 3,357,802 N, 489,017 E and the datum used was NAD 27 Mexico. The project is located 2 km west of the town of Estación Llano (Estación) and is accessed via Mexican State Highway 15 (Pan American highway) from Hermosillo.

The term San Francisco project refers to the area related to the exploitation concessions optioned by TMM, while the term San Francisco property refers to the entire land package (mineral exploitation and exploration concessions) optioned and owned by TMM. The location of the San Francisco property is shown in Figure 4.1.

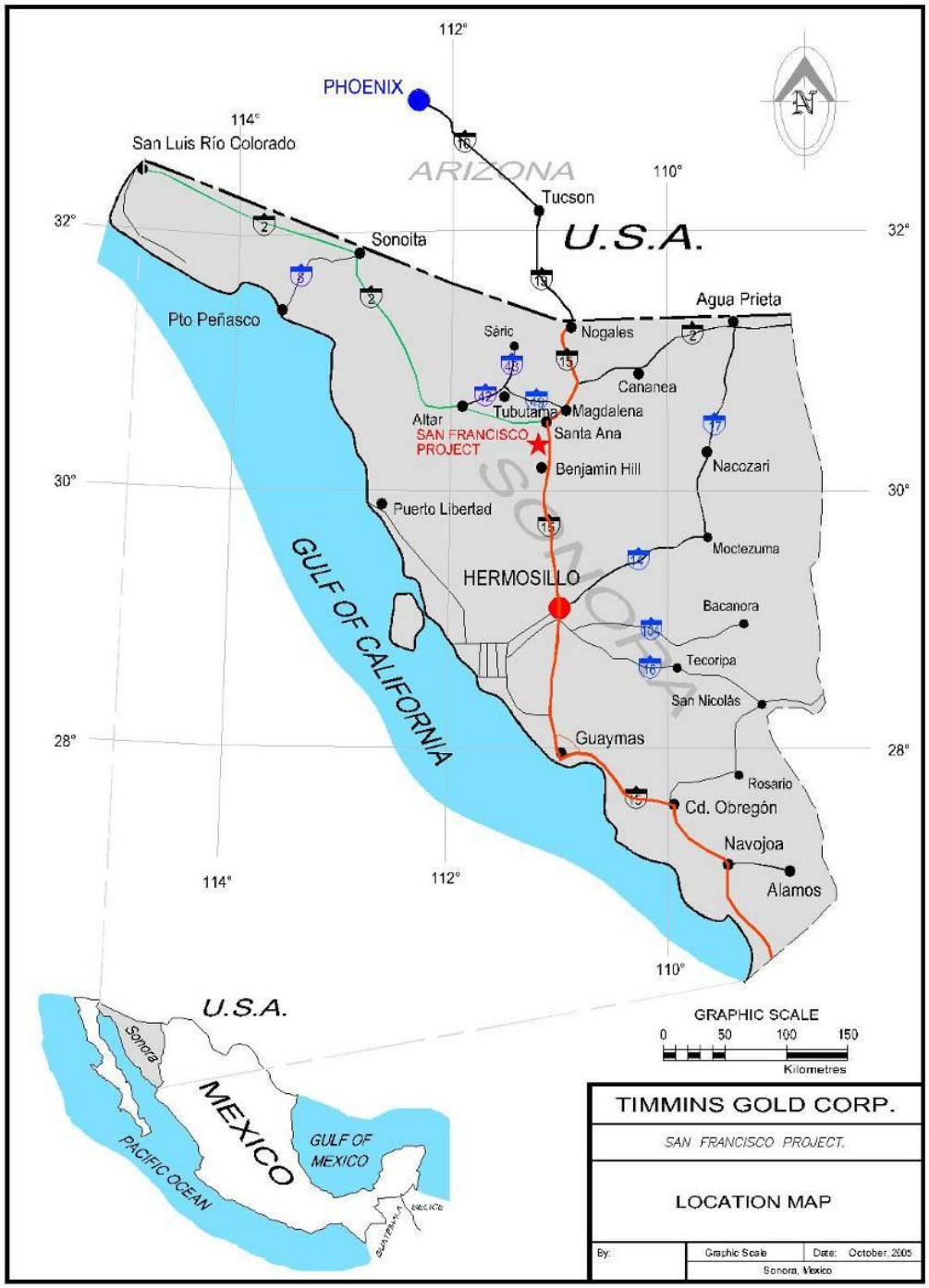
The San Francisco project is comprised of two previously mined open pits (San Francisco and La Chicharra) together with heap leach processing facilities and associated infrastructure located close to the San Francisco pit. Gold mineralization occurs principally as free gold and occasionally as electrum. Trace to small amounts of chalcopyrite, galena, sphalerite, covellite, bornite, argentite-acanthite and pyrrhotite are present. The exploration program carried out by Timmins in 2005 and 2006 focused on exploring the lateral extensions of the mineralized zones to the northwest and southeast of the San Francisco pit area. Additionally the 2006 exploration program focused on exploring potential lateral extensions of the mineralized zone contained in the La Chicharra pit.

TMM advises that it holds the San Francisco property through its wholly-owned Mexican subsidiary via fourteen mining concessions. All the concessions are contiguous and each varies in size for a total property area of 63,540.94 ha. All concessions are subject to a semi-annual fee and the filing of reports in May of each year covering the work accomplished on the property between January and December of the preceding year.

TMM advises that it originally acquired the mining concessions covering the San Francisco project via an option agreement between its Mexican subsidiary (Timmins) and Geomaque de Mexico, S.A. de C.V. (Geomaque de Mexico) on April 18, 2005. The option agreement was subsequently superseded by an acquisition agreement. In terms of the acquisition agreement, Timmins had the option to earn a 50% interest in the exploitation concessions by spending USD 2,500,000 on exploration and development over a two-year period and after Timmins had earned its interest, the property would have been a held as joint venture with Timmins acting as the operator.

In a press release dated March 19, 2007, TMM announced that it had entered into an agreement to increase from 50% to 100% its interest in Molimentales, a company specifically formed to own 100% of the past-producing San Francisco mine.

Figure 4.1
San Francisco Project Location Map



Map provided by Timmins

The purchase price for the mining concessions and other rights was USD 5 million split into two equal payments with the first payment completed upon the execution of the formal agreement and the balance due by October 31, 2007, and the issuance of 10 million common shares of TMM. The shares are subject to a voting trust agreement and a pooling agreement which allows for the release from the pool in equal increments over a three year period. The vendor has the right to appoint a nominee to the board of directors or advisory committee of TMM during the term of the pooling agreement. The purchase price for the gold plant and equipment as well as the other infrastructure was a further USD 3.5 million which may be paid at any time over the next three years without interest.

On October 29, 2007, TMM announced that it had paid the full and final USD 2.5 million to complete the acquisition of the San Francisco mine in a press release.

The Astiazaran family and their private Mexican company retained their ownership of the existing heap leach piles and waste dumps which were thus not part of the mine acquisition. Timmins is currently negotiating to acquire the existing heap leach piles and waste dumps.

The information for the individual exploitation and exploration mineral concessions is listed in Table 4.1. A map of the mineral concessions for the San Francisco property is provided in Figure 4.2.

In 2006, Timmins concluded an access agreement with an agrarian community (an "ejido" in Mexico) called "Los Chinos" whereby Timmins was granted access privileges to 674 ha, the use of the ejido's roads, as well as being able to perform all exploration work on the area covered by the agreement. The agreement is for a period of 10 years with an option to extend the access beyond the 10 year period. In consideration for the ejido granting the access privileges to a portion of its land, Timmins paid the ejido the sum of USD 30,000.

Other parties control four mineral concessions which are contained within the area of the mineral concessions owned by Timmins but none of these concessions impact the main area of the San Francisco project.

The semi-annual fee, payable to the Mexican government to hold the group of contiguous exploitation and exploration concessions is USD 57,795. The semi-annual fees refer to the payments which are made twice per year to the Mexican government to hold the concessions.

Table 4.1
San Francisco Project Summary of Mineral Concessions

Mineral Concession Name	Title Number	Owner	Location (UTM Nad 27 Mex)	Mineral Concession Type[3]	Area (hectares)	Location Date	Expiry Date[3]	Bi-Annual Fee (US $)[4]
San Francisco	198971	Molimentales del Noroeste, de C.V.	488,675.174 E 3,359,396.801 N	Exploitation	48.0000	Nov 13, 1993	Feb. 10, 2044	700
San Francisco Dos	209618	Molimentales del Noroeste, S.A. de C.V.	488,675.174 E 3,359,396.801 N	Exploitation	315.6709	Dec 4, 1996	Aug. 2, 2049	1,300
San Francisco Cuatro[1]	219301	Molimentales del Noroeste, S.A. de C.V.	488,675.174 E 3,359,396.801 N	Exploitation	5,189.7041	Aug 18, 2000	Feb. 25, 2053	7,000
Llano II	197203	Molimentales del Noroeste, S.A. de C.V.	483,652.702 E 3,356,290.081 N	Exploitation	500.0000	Oct 23, 1986	Aug. 18, 2043	5,500
Llano III	197202	Molimentales del Noroeste, S.A. de C.V.	483,652.702 E 3,356,290.081 N	Exploitation	500.0000	Oct 23, 1986	Aug. 26, 2043	5,500
Llano IV	222787	Molimentales del Noroeste, S.A. de C.V.	488,675.174 E 3,359,396.801 n	Exploitation	500.0000	Aug 31,2004	Aug 30, 2054	1,200
Llano V		Molimentales del Noroeste,S.A. de C.V.	488,675.174 E 3,359,396.801 n	Exploitation	500.0000	Aug 31, 2004	Aug 30,2054	1,200
Timmins	226519	Timmins Goldcorp México, S.A. de C.V.	488,675.174 E 3,359,396.801 N	Exploration	337.0000	Aug. 26, 2005	Jan. 23, 2056	200
Timmins III Fraccion 1	227237	Timmins Goldcorp México, S.A. de C.V.	481,529.246 E 3,371,837.280 N	Exploration	346.0004	Feb. 15, 2006	May 25, 2056	190
Timmins III Fraccion 2	227238	Timmins Goldcorp México, S.A. de C.V.	481,529.246 E 3,371,837.280 N	Exploration	54.2835	Feb. 15, 2006	May 25, 2056	35
Timmins II Fraccion Sur[2]	228260	Timmins Goldcorp México, S.A. de C.V.	488,675.174 E 3,359,396.801 N	Mining Concession	20,370.0604	Nov. 17, 2005	Mar. 13, 2056	9,200
Pima[2]	228261	Timmins Goldcorp México, S.A. de C.V.	486,058.775 E 3,375,493.728 N	Mining Concession	15,772.0000	Nov. 17, 2005	Mar. 13, 2056	7,250
Total:	-	-	-	-	**43,432.7193**	-	-	**32,970**

NOTES:

1 The San Francisco Cuatro claim was regularized to the original application of 5,189.7041 ha in place of the previous 453.7127 Ha claim.

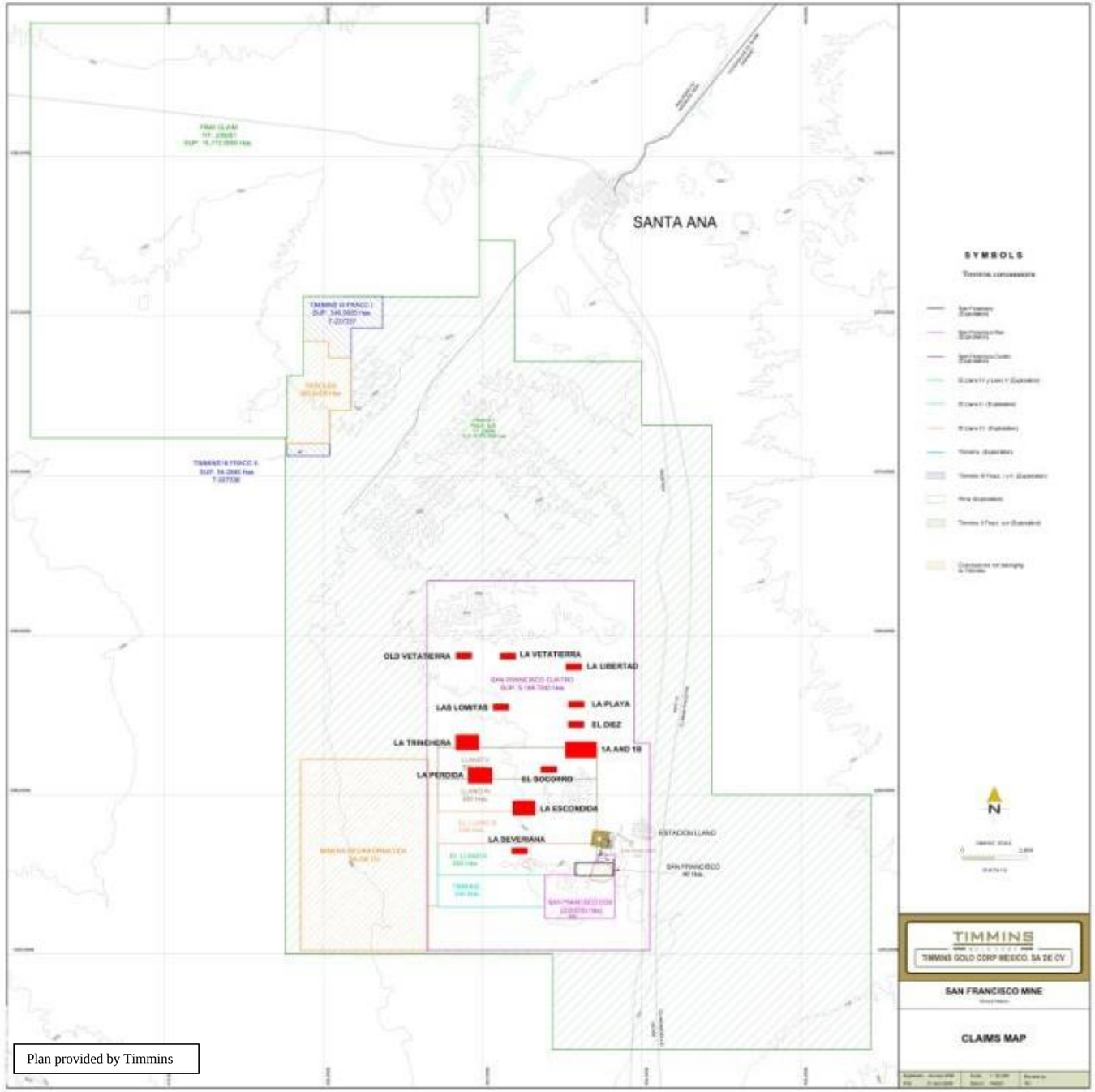
2 The Timmins II claim, originally staked with a surface of 39.403.0000 Ha., was titled for the Direccion General de Minas (DGM), after survey works with a surface of 36,142.0604 Ha. In the past year due to a change in exploration strategy the Timmins II claim was divided into two claims, Timmins II Fraccion Sur and Pima.

3 In the last year the mining law was modified and all concessions granted by the DGM are exploitation mining concessions, this is the case with the Timmins II Fraccion Sur and Pima claims. Also the new mining law grants the new mining concessions for a term of 50 years which is the same length of time granted to the old exploitation concessions under the old mining law and have eliminated the initial period of 6 years for the exploration concessions.

4 Fees are estimated in US dollars based on payments in 2007. Inflation and other factors are considered by the Mexican government each year.

Table provided by Timmins

Figure 4.2
San Francisco Property - Plan Showing Mineral Concessions



Plan provided by Timmins

Prior to 2005, exploration concessions were valid for six years and could not be extended but could be converted into one or more exploitation concessions after the six-year period concluded, provided the bi-annual fee and work requirements were in good standing.

When the Mexican mining laws were changed in 2005, all mineral concessions granted by the Dirección General de Minas (DGM) became mining concessions and there are no longer separate specifications for mineral exploration or exploitation concessions. A second change to the mining laws resulted in all mining concessions being granted for 50 years provided the concessions remained in good standing. As part of the latter change, all former exploration concessions which were previously granted for 6 years became eligible for the 50 year term.

For any concessions to remain valid the semi-annual fees must be paid and a report has to be filed by May of each year which covers the work conducted during the preceding year. Concessions are extendable provided that the application is made within the five-year period prior to the expiry of the concession and the bi-annual fee and work requirements are in good standing.

The semi-annual fees are based on a number of factors. Prior to January, 2006, exploration and exploitation mineral concessions had two different fees based on the type of mineral concession and the amount of time since they were issued. After January, 2006, in accordance with the 2005 changes in the mining laws, a single fee per hectare was implemented, at a rate which escalates based on the age of the title. See Table 4.2 for the present fee rates per hectare.

Table 4.2
San Francisco Project Semi-Annual Fee Rates

Year	Type of Concession	Fee for the Period after Issue	Fee per Hectare (MXN)
Prior to January 2006	Exploration	For the first year	2.0230
		For the second through fourth years	6.01
		After the fifth year of issue	12.43
	Exploitation	For the first and second years	25.06
		For the third and fourth years	50.34
		After the fifth year of issue	88.29
After January 2006	All concessions are mining concessions	For the first and second years	4.60
		For the third and fourth tears	6.88
		For the fifth and sixth years	14.24
		For the seventh and eighth years	28.64
		For the ninth and tenth years	57.26
		After the tenth year of issue	100.79

Table provided by Timmins

All mineral concessions must have their boundaries orientated astronomically north-south and east-west and the lengths of the sides must be one hundred metres or multiples thereof, except where these conditions cannot be satisfied because they border on other mineral concessions. The locations of the concessions are determined on the basis of a fixed point on the land, called the starting point, which is either linked to the perimeter of the concession or

located thereupon. Prior to granting a concession the company must present a topographic survey to the DGM within 60 days of staking. Once this is completed the DGM will usually grant the concession. The starting point (white cairn on the hilltop) for one of the mineral concessions comprising the San Francisco project is shown in Figure 4.3.

Since the San Francisco project is located on a number of exploitation concessions upon which mining has previously been conducted, all exploration work continues to be covered by the environmental permitting already in place and no further notice is required to be given to any division of the Mexican government. The original environmental permitting of the San Francisco mine site is good for the duration of the exploitation concessions. Water for any drilling programs at the San Francisco project is obtained from the onsite water wells.

Micon is unaware of any outstanding environmental liabilities attached to the San Francisco project and is unable to comment on any remediation which may have been undertaken by previous owners.

Figure 4.3
View of the Starting Point for One of the Concessions on the San Francisco Property

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The San Francisco property is readily accessible from Hermosillo, the state capital of Sonora, via Mexican State Highway 15 (Pan American Highway), north from Hermosillo for 150 km or 120 km south of the United States/Mexico border city of Nogales. The San Francisco mine site is 2 km west of the town of Estación Llano (Estación). The major population centre for the region is Magdalena de Kino (Magdalena) to the north with a population of over 50,000 inhabitants.

The mineral concessions are located approximately due west and north of Estación with the closest accommodations located in Santa Ana, a small city located to the north on Highway 15.

Timmins maintains guarded gates across the access road to the mine and immediate project area. Exploration can be conducted year round with the desert "monsoon season" occurring between July and September. If and when economic resources of mineralization are exploited, materials needed to supply the mine could be transported by either truck (utilizing Mexican State Highway 15) or by rail (utilizing the Ferrocarril del Pacifico railway**),** both of which pass through the community of Estación.

Timmins has been granted the temporary occupation of surface rights at the San Francisco mine by the DGM for the duration of the exploitation concessions. See Table 4.1 for the details regarding the exploitation concessions. In the case of an exploration concession, the holder is granted temporary occupancy for creation of land easement needed to carry out exploration for the duration of the mineral concession. In order to commence mining, the holder of the concession would be required to negotiate the surface rights with the legal holder of these rights or to acquire the surface rights through a temporary expropriation, which is the case with the San Francisco project surface rights. Annual payments for the surface rights on the San Francisco project is MXN 243,769, indexed to inflation, or approximately USD 22,500.

Water for the drilling programs is available from the wells located on the mine site. The water table in the area of the mine is located approximately 25 m below the surface. A typical water well is shown in Figure 5.1.

The project is located in the Arizona-Sonora desert in the northern portion of the Mexican state of Sonora. The climate at the project site ranges from semi-arid to arid. The average ambient temperature is 21°C, with minimum and maximum temperatures of -5ºC and 50ºC, respectively. The average rainfall for the area is 330 mm with an upper extreme of 880 mm. The wet season or desert "monsoon" season is between July and September and heavy rainfall can hamper exploration at times.

The San Francisco property is situated within the southern Basin and Range physiographic province which is characterized by elongate, northwest-trending ranges separated by wide

alluvial valleys. San Francisco is located in a relatively flat area of the desert with the topography ranging between 700 and 750 m above sea level.

The desert vegetation surrounding the San Francisco mine is composed of low lying scrub, thickets and various types of cacti, with the vegetation type classified as Sarrocaulus Thicket. The state of Sonora is well known for its mining and cattle industries, although US manufacturing firms are starting to move into the larger centres as a result of the North American Free Trade Agreement (NAFTA). See Figure 5.2 for a view of the desert surrounding the San Francisco property.

Figure 5.1
View of the Water Well Located on the San Francisco Project

Figure 5.2
View of the Sonora Desert Surrounding the Property

6.0 HISTORY

6.1 SAN FRANCISCO PROPERTY AND GOLD MINE

The San Francisco gold mine is a past producing heap leach operation which was in production between 1995 and 2002. However, during the last two years of operation gold was being recovered from the leach pads only, with no mining being conducted within the San Francisco and La Chicharra open pits.

Placer mining and small scale underground mining began in the San Francisco mine area during the early 1940s. This limited work is what drew Fresnillo to the area in 1983. In 1985, three diamond drill holes and 30 conventional percussion drill holes were completed on the property. The results of these drill holes were encouraging enough to warrant drilling additional diamond drill holes during 1986. In 1987, 540 m of underground development was conducted, including a decline and a number of drifts and cross-cuts. The decline was completed to the 685 m elevation above sea level, where numerous 1.8 by 1.5 m drifts and cross-cuts were developed. Fresnillo drilled 10 diamond drill holes and 25 reverse circulation drill holes in 1988, and an additional 226 reverse circulation drill holes in 1989. In addition, metallurgical testing and an induced polarization survey were completed in 1989. In 1990 and 1991, Fresnillo completed an additional 108 reverse circulation drill holes. Fresnillo decided to sell the property in 1992, at which time it was acquired by Geomaque. See Figure 6.1 for an example of one of the rotary drill site locations southeast of the main pit. As part of the Geomaque purchase, Fresnillo retained a 3% NSR and the option to re-acquire a 50% interest by paying Geomaque twice what had been expended. Geomaque completed a feasibility study in 1993 and drilled a further 69 reverse circulation drill holes in 1994. Geomaque acquired the NSR and option back from Fresnillo in 1995 for USD 4,700,000.

Geomaque conducted its activities in Mexico through its subsidiaries, Geomaque de Mexico, S.A. de C.V. (Geomaque de Mexico) and Mina San Francisco, S.A. de C.V. (Mina San Francisco).

Geomaque began construction of the San Francisco mine in 1995, with production beginning in late 1995. Production began at the rate of 3,000 t of ore per day or 30,000 ounces of gold per year. However, as a result of the discovery of additional reserves, an expansion of the mining fleet, crushing system and gold recovery plant were undertaken in an effort to increase production to 10,000 t of ore per day. Due to the prevailing market conditions in February, 2000, Geomaque announced a revised mine plan whereby higher grade ore with lower stripping ratio would be mined from the San Francisco pit and the La Chicharra deposit, which is located to the west of the San Francisco pit.

The San Francisco deposits consisted of the El Manto, the San Francisco, the En Medio, and the El Polvorin deposits. All these deposits were later incorporated into the main San Francisco pit. Another deposit, the La Chicharra zone, was mined during the last two years of production as a second pit.

Mining ended and the operation entered into a leach only mode in November, 2000. In May, 2002, the last gold pour was conducted, the plant was mothballed, and clean-up activities at the mine site began. See Figure 6.2 for a photographic overview of the San Francisco pit and leach pad taken from a hill to the southwest of the mine site.

Figure 6.1
Location of One of the Rotary Drill Sites Located to Southeast of the Main Pit

Figure 6.2
View of the San Francisco Gold Mine with Estación Llano in the Background

In 2001, to settle debts related to lease arrangements of construction equipment to Geomaque de Mexico, Butler Machinery Co. (Butler) accepted a payment of USD 500,000, the proceeds in excess of USD 500,000 on the sale of certain equipment from the San Francisco mine and a 1% NSR on any future gold production from the unmined resources in the main pit of the San Francisco mine. No present value was ascribed to the rights at the time of the agreement. Micon has been advised by Timmins that the contract between Timmins and Geomaque de Mexico states that the agreement between Geomaque and Butler has ended and it is understood by Micon that Geomaque will continue to be responsible for covering the 1% NSR, should the property be returned to production. Therefore, Timmins advises that it has free and clear title to the property.

Geomaque signed a Surface Rights Agreement with a group of rights holders (the "Ejido Jesus Garcia Heroe De Nacozari" or Ejido Jesus Garcia for short). Based on a letter agreement dated July 7, 1999, the Ejido Jesus Garcia agreed to transfer to the company a surface area of 800 ha, for a total consideration of USD 1,000,000, of which USD 75,000 was due and payable on signing of the agreement. The letter agreement and its efficacy were the subject of litigation between Geomaque and the Ejido Jesus Garcia whereby the company sought to have the agreement declared void, its deposit returned and other remedies, and the Ejido Jesus Garcia sought to have the agreement held effective and sought, inter alia, the payment of the balance of the purchase price and other relief.

In the summer of 2003, Geomaque sought and received shareholder approval to amalgamate the corporation under a new Canadian company, Defiance Mining Corporation (Defiance).

On November 24, 2003, Defiance sold its Mexican subsidiaries, Geomaque de Mexico and Mina San Francisco, to the Astiazaran family and their private Mexican company for a total consideration of USD 235,000. The Mexican subsidiaries held the San Francisco gold mine and the sale relieved Defiance of long-term liabilities totalling USD 1,900,000, including a USD 925,000 surface rights purchase obligation, approximately USD 760,000 in reclamation provisions and other payables totalling USD 263,000. The litigation of the surface rights between the Ejido Jesus Garcia and Geomaque de Mexico was settled in favour of Geomaque de Mexico on January 20, 2005. Geomaque de Mexico has been granted by the DGM the temporary occupation of surface rights at the San Francisco mine for the duration of the exploitation concessions. Annual payments for the surface rights to the Ejido Jesus Garcia is MXN 243,769, indexed to inflation, or approximately USD 31,736.

Since June, 2006, the Astiazaran family and their company, Desarrollos Prodesa S.A. de C.V. (Prodesa) retained ownership of the the waste dumps and the leach pads, and have been extracting sand and gravel intermittently for use in highway construction and other construction projects. Timmins is currently negotiating the purchase of these heap leach piles and waste rock dumps. See Figure 6.3 for a view of the gravel extraction from the old leach pads at the San Francisco mine site.

Figure 6.3
Extraction of Gravel from the Leach Pad for Construction Use

TMM was incorporated on March 17, 2005 under the Business Corporations Act of British Columbia. TMM originally acquired the exploitation concessions covering the San Francisco project through its wholly-owned Mexican subsidiary, Timmins, via an option agreement with Geomaque de Mexico on April 18, 2005. That option agreement was subsequently superseded by an acquisition agreement. In terms of the acquisition agreement, Timmins had the option to earn a 50% interest in the exploitation concessions by spending USD 2,500,000 on exploration and development over a two-year period and after Timmins has earned its interest, the property will be operated as a joint venture with Timmins acting as the operator.

In a press release dated March 19, 2007 TMM announced that it had agreed to increase its interest from 50% and had entered into an agreement to acquire a 100% interest in Molimentales, a company specifically formed to own 100% of the past producing San Francisco mine.

On October 29, 2007, TMM announced that it had paid the full and final USD 2.5 million to complete the acquisition of the San Francisco mine in a press release.

During August and September, 2005, Timmins conducted a reverse circulation drilling program comprised of 14 reverse circulation holes based on the results of previous drilling conducted by both Fresnillo and Geomaque. The 2005 drilling program focused on confirming and exploring extensions of the gold mineralization to the northwest and southeast of the existing San Francisco pit. The results of the drilling program confirmed the extension of the gold mineralization to the northwest beyond the limits of the present pit and

the presence of a higher grade gold zone. To the southeast the 2005 drilling results disagreed with the previous drilling conducted by Geomaque, with only erratic values detected. However, drill hole TF-06 ended in 6.10 m averaging 2.817 g/t gold and this drill hole will be re-interpreted as part of a future exploration program focusing on reassessing the mineralization to the southeast.

In 2006, Timmins conducted an intensive exploration drill program which was based on the analysis of Geomaque's drilling results, the 2005 Timmins drill results and the geological, geochemical data and a structural re-interpretation of the gold mineralization controls within the known deposit. The drilling program consisted of 28 reverse circulation and 28 diamond drill holes with three general target areas covered by the program. The first area covered by the drilling program was the immediate area north and northwest of the existing San Francisco pit, with a particular emphasis placed on drilling in the area covered by the former crusher. The second area covered by the 2006 drilling program was located to the north and south of the La Chicharra pit. The La Chicharra pit was the second pit mined by Geomaque at the project site and is located west of the San Francisco pit on the other side of a small mountain. The third area covered by the drill program investigated places where direct observations by Timmins geologists and previous geological mapping indicated favourable lithology, hydrothermal alteration and geochemical results for the continuation of the mineralization. The details of the 2006 exploration program and its results were discussed in a February, 2007 Technical Report entitled "NI 43-101 Technical Report and Resource Estimate for the San Francisco Gold Property, Estación Llano, Sonora, Mexico." This report was filed on the SEDAR website on February 27, 2007 by TMM.

Table 6.1 provides a summary of the significant historical drilling data at the San Francisco project upon which Timmins based part of its 2006 drilling programs.

Timmins subsequently conducted a follow-up drilling program in 2007. Details of the 2007 exploration program were outlined in a November, 2007 press release by TMM. Although, the results of the Timmins exploration program have been commented on in Sections 10 and 11 of this report, the results of the 2007 exploration program have not been detailed in this report and only the details mentioned in the press release included in this report.

6.2 HISTORICAL RESOURCE ESTIMATES

A discussion regarding the historical mineral resource estimate was contained in a December 20, 2005 technical report entitled "Technical Report on the San Francisco Mine property, Estación Llano, Sonora, Mexico." The technical report was posted to the SEDAR website on April 28, 2006.

The historical resource estimate for the San Francisco mine has been superseded by the January, 2007 updated resource estimate by IMC. In 2006, IMC was asked by Timmins to update/develop a resource block model and to estimate the mineral resources for the San Francisco mine using the historical Geomaque data along with the results of Timmins' 2005

Table 6.1
Summary of Significant Historical Reverse Circulation Hole Intervals which Support the 2006 Drilling Program on the San Francisco Project

Area Of Drilling	Drill Hole No.	Total Depth (m)	Drill Hole Angle	Section Line	Location of Drill Hole (UTM)		Elevation[1]	Mineralized Intersections				Comments
					Easting	Northing		From (m)	To (m)	Distance (m)	Gold (g/t)	
San Francisco northwestern projection of the mineralization	CI-571	80.00	-70	1500	488,017.005	3,357,837.232	735.010	8.00	12.00	4.00	2.930	Isolated holes.
	CI-572	80.00	-70	1500	488,186.252	3,358,199.730	724.700	46.00	50.00	4.00	0.300	
	CI-573	80.00	-70	1500	488,123.520	3,358,068.070	735.170	10.00	14.00	4.00	0.340	
	CI-574	76.00	-70	1600	487,844.510	3,357,699.810	740.770	8.00	12.00	4.00	1.140	
	CI-607	60.00	-65	1600	487,839.230	3,357,688.190	742.510	1.50	7.50	6.00	0.400	
							And	13.50	16.50	3.00	0.400	
	CI-608	60.00	-65	1740	487,703.720	3,357,724.500	757.820	21.00	22.50	1.50	0.340	
							And	54.00	57.00	3.00	0.560	
	CI-609	69.00	-65	1500	488,047.090	3,357,896.430	746.670	1.50	3.00	1.50	1.780	
							And	10.50	18.00	7.50	0.930	
							And	27.00	33.00	6.00	3.340	
							And	45.00	48.00	3.00	0.410	
							And	67.50	69.00	1.50	0.500	
North Pit Crusher Area	CI-581	50.00	-70	1000	488,685.030	3,358,089.050	709.720	6.00	8.00	12.00	0.920	Very spaced holes at the former crusher area.
							And	48.00	50.00	2.00	4.380	
	CI-583	50.00	-70	1160	488,469.300	3,358,000.380	718.350	2.00	10.00	8.00	0.310	
	CI-596	120.00	-90	920	488,774.800	3,358,075.110	709.090	12.00	24.00	12.00	0.788	No more exploration was done because the crusher was installed to start the operation.
							And	36.00	42.00	6.00	0.410	
							And	60.00	62.00	2.00	1.280	
							And	72.00	80.00	8.00	1.220	
	CI-899	60.00	-90	1160	488,519.710	3,358,117.110	713.410	2.00	10.00	8.00	0.280	
							And	18.00	24.00	6.00	0.510	
	CI-900	58.00	-90	1220	488,465.330	3,358,143.710	714.790	8.00	12.00	4.00	0.620	
	692	120.00	-90	880	488,689.650	3,357,817.890	712.190	62.00	76.00	14.00	1.345	
							And	82.00	86.00	4.00	0.440	
							And	104.00	106.00	2.00	0.840	
	1044	258.00	-90	900	488,670.490	3,357,825.410	698.240	32.00	42.00	10.00	0.510	
							And	56.00	62.00	6.00	2.443	
							And	126.00	140.00	14.00	0.677	
							And	148.00	150.00	2.00	2.600	
							And	176.00	180.00	4.00	0.880	
							And	186.00	196.00	10.00	0.928	
							And	210.00	216.00	6.00	0.650	
							And	226.00	240.00	14.00	1.511	
	861	156.00	-70	920	488,653.400	3,357,833.580	704.890	40.00	48.00	8.00	0.915	
							And	74.00	78.00	4.00	1.250	
							And	118.00	136.00	18.00	0.910	
							And	146.00	156.00	10.00	0.928	
	CI-152	64.00	-70	940	488,626.050	3,357,824.330	713.840	44.00	56.00	12.00	1.533	
	1080	138.00	-70	960	488,634.470	3,357,888.940	713.600	70.00	76.00	6.00	1.740	
							And	96.00	102.00	6.00	2.086	
	1025	264.00	-90	1040	488,524.210	3,357,840.540	716.180	16.00	24.00	8.00	0.912	
							And	46.00	48.00	2.00	1.470	
							And	56.00	62.00	6.00	1.993	
							And	170.00	180.00	10.00	0.472	
							And	230.00	238.00	8.00	1.405	

Note: [1] Metres above sea level
Table provided by Timmins

and 2006 exploration drilling programs. The updated resource estimate for the San Francisco mine was completed by IMC in January, 2007 and is discussed in Section 17 of this report.

The updated resource estimate by IMC is compliant with the current CIM standards and definitions required by NI 43-101 and is therefore reportable as a mineral resource by TMM. However, the reader should be cautioned that mineral resources that are not mineral reserves do not have demonstrated economic viability.

IMC is in the process of preparing a new mineral resource and reserve estimate based on the results of the 2007 exploration drilling program. This update of the resources and reserve will be presented in a future Technical Report as they were not available in time for this Technical Report.

There are no known resource estimates for any of the other portions of the San Francisco property.

6.3 HISTORICAL PRODUCTION

Historical production occurred at the San Francisco gold mine between 1996 and 2002. Production was conducted using open pit mining methods with the gold recovered using the heap leaching method. During its production phase the San Francisco mine extracted 13,490,184 t at a grade of 1.13 g/t gold for a total of 488,680 contained ounces of gold. A total of 300,281 oz gold and 96,149 oz of silver were recovered, with the gold recovery estimated to be 61.4%.

Table 6.2
San Francisco Project Geomaque Annual Production 1996 to 2000

Year	Dry Crush on Pads (t)	Grade (g/t)	Ounces on Pad	Gold/Silver Ounces Doré	Gold Ounces Doré	Gold Recovered (%)
1996	1,735,550	1.32	73,655	46,787	36,127	49.0
1997	2,288,662	1.12	82,412	75,847	54,519	66.2
1998	3.074,902	1.05	103,803	86,940	58,808	56.7
1999	3,010,639	1.14	110,345	98,726	64,371	58.3
2000	3,380,431	1.09	118,465	104,953	69,100	58.3
2001					17,092	
2002					264	
Total	**13,490,184**	**1.13**	**488,680**		**300,281**	**61.4**

Note: 301,893 tonnes of mineral and 975,900 tonnes of waste rock were mined in 1995.
Table taken from the 2006 San Francisco Scoping Study by Sol & Adobe Ingenieros Asociados S.A. de C.V.

Other mines or exploratory shafts within the district are El Durazno (gold/silver), El Aguaje (gold), El Jabali (manganese), La Jarra (gold), El Refugio (gold), Caracahui (copper/gold), Sonora Copper (copper/gold), Las Animas (gold/copper), La Colorada (gold), Libertad (gold) and the La Chicharra (placer gold). Production statistics for these mines or exploratory shafts are unavailable and in some cases there is very little published data on these workings.

7.0 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The following descriptions of the regional geology were extracted from Prenn (1995).

> "The San Francisco property is situated in a belt of metamorphic rocks that hosts numerous gold occurrences along the trace of the Mojave-Sonora megashear, which trends southeast from south-central California into Sonora. The megashear is a left-lateral transform fault which became active during the Jurassic period and exhibits up to 800 km of displacement. Deformation along the megashear occurred along with metamorphism (Calmus et al., 1992) and since the formation of the megashear the area has been subjected to both tectonic compressional and tensional forces."

> "The following description is extracted from Silberman (1992). The northwest-trending range-front faults and numerous low-angle shear zones related to thrust or detachment faults are the most common structures. The Mojave-Sonora megashear as defined by Silver and Anderson (1974) is a regional northwest-trending feature. It separates the Precambrian basement rocks of slightly differing ages. The Jurassic rocks which occupy the zone are strongly deformed along low-angle thrust faults and the associated sedimentary rocks are tightly folded. The south-western boundary of the megashear appears to be a major fault that juxtaposes Precambrian basement rocks against the Jurassic magmatic terrain (Anderson and Silver, 1979). Up to 800 km of left lateral movement has been proposed for this shear after the Middle Jurassic period. Others (Jaques et al., 1989) have suggested that the megashear is a Cretaceous thrust front reactivated as a middle Tertiary detachment. The metamorphism in the area has been postulated to have occurred with the megashear or the magmatic activity of the Middle to Late Jurassic periods (Tosdal et al., 1989). However, others propose a close relationship between deformation and the closing of the marginal basin after its subduction below the volcanic arc, or the result of Late Cretaceous or Tertiary compression associated with uplift and low-grade metamorphism (De Jong et al, 1988). Calmus (1992) believes it is unquestionable that a Cretaceous-Tertiary (Larimide) tectonic event occurred but that it is superimposed upon older Nevada and Lower Cretaceous compressional and extensional phases. Many of the Sonoran gold deposits are located at or near the Mojave-Sonora megashear."

The Cretaceous Represo Formation, comprised of mudstone, sandstone, limestone, and conglomerate, crops out north of the project area at Santa Ana. A Tertiary leucogranite (41 Ma) intrudes the older metamorphic rocks and hosts most of the gold mineralization at the San Francisco project. The granite and granitic gneiss are sub-parallel to the foliation of the metamorphic rocks. The above rocks are cut by younger, post-mineralization andesite and diorite dykes.

The Basin and Range Province which extends into Sonora, from the United States, is characterized by northwest-trending valleys and ranges. Paleozoic rocks, including quartzite

and limestone, overlie the Precambrian locally. The valleys are covered and infilled by recent gravels. See Figure 7.1 for the regional geology map of the San Francisco mine area and location of the San Francisco and La Chicharra pits.

7.2 PROPERTY GEOLOGY

7.2.1 ROCK TYPES

The oldest rocks on the project site are of the San Francisco unit, a metamorphic sequence that is comprised of banded quartz-feldspathic gneiss and augen gneiss, green schist, amphibolite gneiss and some amphibolite and marble lenses (Calmus et al., 1992). The rocks are altered to a quartz and sericite assemblage while biotite is altered to chlorite. The amphiboles are of two types; hornblende and tremolite-actinolite. The protolith of the San Francisco unit is probably a detrital sequence with volcanic tuffs and mafic dykes (Calmus et al., 1992). Foliation of the unit strikes east and dips north. The San Francisco unit hosts the gold mineralization at the San Francisco (main pit), El Manto (north pit), En Medio (in the main pit), and El Polvorin (west pit) deposits. All these pits were later incorporated into the main San Francisco pit.

Below, and in fault contact with the San Francisco unit, is the Jurassic Coyotillo unit, a low-grade metamorphic unit which is comprised of gneiss with schist interbeds, granite gneiss and gneiss (Luna and Gastelum, 1992). Locally the unit contains black phyllites, meta-sandstone, meta-conglomerate, meta-tuff and quartzite.

Lower Cretaceous beds of the Represo Formation occur in the northern part of the project area above and in fault contact with the San Francisco and Coyotillo units. The beds consist of conglomerate, sandstone, siltstone and limestone. A Tertiary quartz diorite stock intrudes the Represo Formation in the northern part of the project area.

Metamorphic rocks of the San Francisco and Coyotillo units are intruded by a Tertiary leucocratic granite which hosts much of the San Francisco deposit. The granite contains visible feldspar and quartz, and is predominantly porphyritic to gneissic. The main intrusion is roughly stratiform, ten or more metres thick, and subparallel to foliation of the metamorphic rocks. An age date of 41 Ma has been obtained on hydrothermal sericite, which yields both a minimum date on the intrusive and a maximum date of the gold mineralization (Calmus et al., 1992). Previously, the intrusive was believed to be contemporaneous with the deformation and metamorphism and therefore regarded as Jurassic in age.

Post-mineral Tertiary andesite and hornblende-diorite dykes cut the older rocks. These dykes range in size from a few centimetres to approximately 30 m wide and are both subparallel to, and cut, the metamorphic foliation. See Figure 7.2 for a geological map of the San Francisco mine site and the locations of the 2005 reverse circulation drill holes.

Figure 7.1
Regional Map of the Geology of the San Francisco Property

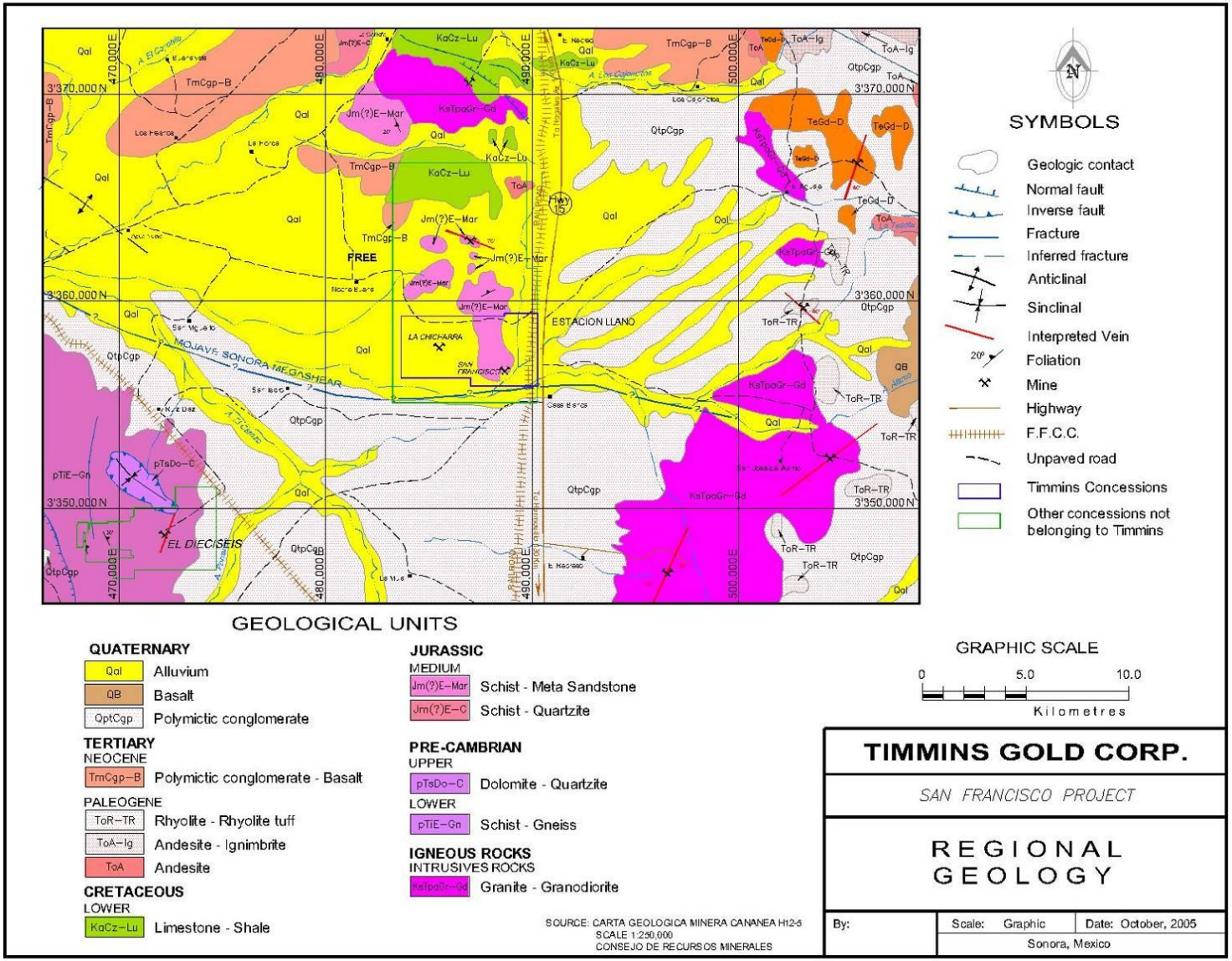
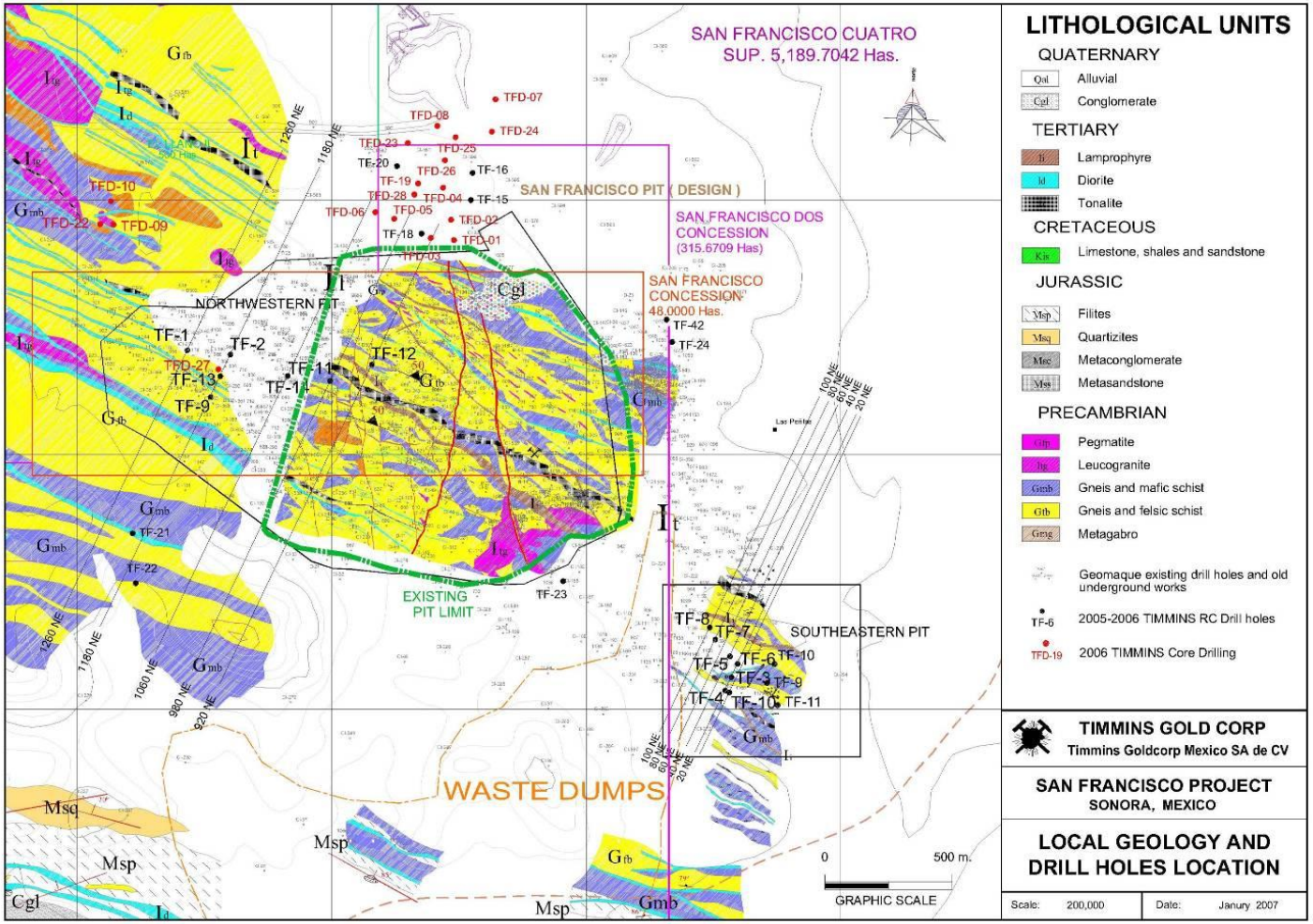


Figure 7.2
San Francisco Mine site Geology Map



Maps provided by Timmins
Note::"Southeastern Pit"was never mined as part of the original San Francisco operation.

7.2.2 STRUCTURAL GEOLOGY

At Cerro Bajarita, located half a kilometre south of the San Francisco deposit, a thrust fault has been described between the San Francisco gneiss and the Coyotillo metamorphic unit. The fault strikes east and dips steeply north at 60˚ to 70˚, possibly due to post-thrust tilting. This thrust may correlate with the thrust separating metamorphic rocks from the overlying sedimentary sequence in the Sierra La Vetatierra mountains in the northern portion of the project.

The normal faults are post-mineral and of the basin and range type. Two post-mineral faults that offset mineralization are the San Francisco fault, with a maximum displacement of 40 m down to the east, and the El Carmen fault. Both faults generally strike north and dip 65˚ to 70˚ east. Other faults strike from north to N40˚W and dip to the east.

The granitic gneiss containing the mineralization at the San Francisco project is intensely fractured with a total of five fracture sets having been identified on the property, although there are only two primary fracture sets. One of the primary fracture sets strikes 36˚ to 60˚ east and dips 70˚ to 90˚ northwest, while the other strikes 64˚ to 73˚ west and dips 46˚ to 66˚ northeast. The regional fracture sets are generally parallel to major faults and perpendicular to foliation planes.

The main vein systems in the region strike 50˚ to 80˚ west and the dips range from northeast to southwest. These vein systems are the San Francisco, La Playa, El Diez, La Chicharra, several in the La Mexicana area, Area 1B and La Escondida. Another system of veins is the La Trinchera, Casa de Piedra, parts of area 1B and La Mexicana veins which strike 60˚ to 80˚ east and dip northwest to southeast. Although the age relation between the two systems is unknown, it is believed that the northeast system is probably the later stage.

The metamorphic foliation in the San Francisco deposit primarily strikes 78˚ west and dips 68˚ to the northeast. However, regionally the foliation is variable but generally ranges from east-west to 60˚ west with varying dips.

The original bedding is recognized in the metavolcanic-sedimentary rocks to the south at Cerro La Bajarita, and is highly variable with strikes ranging from 70˚ west to 80˚ west and dips to the north. The sedimentary beds of the Represo Formation in the northern portion of the property strike 60˚ to 70˚ west and dip to the northeast.

Dykes of intermediate composition in the project area strike predominantly 63˚ west and dip 58˚ to the northeast. Several dykes are intruded along the planes of foliation, and others cut foliation of the metamorphic units. In the Sierra La Vetatierra mountains in the northern portion of the project, dykes strike N60˚E, dip to the northwest, and represent a later system of fractures.

Metamorphic folds, including isoclinal, open symmetrical folds and kink folds, have been described, but no systematic description of folds has been found in the literature.

8.0 DEPOSIT TYPES

At the San Francisco project, Timmins is targeting a large volume low grade disseminated gold deposit contained within a gneiss with schist horizons, and in a granitic gneiss, as the main host rocks for gold mineralization. Mineralization occurs as disseminated free gold in stockworks and in brecciated areas, related to goethite, quartz and tourmaline, as well as in low and high angled vein-shaped structures. The geochemistry is variable with microscopic gold usually occurring at the periphery of goethite crystals as remobilizations and, to a lesser extent, in quartz and pyrite.

The associated mineralogy, the possibility of associated tourmaline and the style of mineralization suggest that the San Francisco deposit might be of mesothermal origin. Others have suggested the same genesis based on these and other factors including fluid inclusion studies (Luna, 1992).

9.0 MINERALIZATION

The San Francisco property is located within the Sierra Madre Occidental metallogenic province which extends along western Mexico from the state of Sonora, south to the state of Jalisco. In the state of Sonora, the most important metal produced in the Sierra Madre province is copper, with the Cananea porphyry copper deposit being the most well known. Gold and silver deposits are next in importance and are hosted mainly in sedimentary rocks and brecciated volcanic domes.

At the San Francisco project, gold occurs principally as free gold and occasionally as electrum. Gold is found, in decreasing abundance, with goethite after pyrite, with pyrite and, to a much lesser extent, with quartz, galena and petzite (Ag_3AuTe_2). Although it is clear that the gold was deposited at the same time as the sulphides the paragenetic relationships are not well understood. There is the possibility that some secondary remobilization may have occurred as evidenced by minor amounts of gold occurring in irregular forms along with or on top of drusy quartz (Prenn, 1995).

The gold occurs in a granitic gneiss and the presence of pyrite (or goethite after pyrite) may be an indication of gold. Stockwork quartz veinlets, some with tourmaline, also exist in the mineralized zone. However, the presence of quartz, even with tourmaline, is not necessarily an indication of the presence of gold. Quartz veinlets with tourmaline without gold mineralization were found hundreds of metres away from the San Francisco deposit. Alvarez (in Prenn, 1995) suggested that some tourmaline was part of the mineralizing system, but could be distinguished from the tourmaline found elsewhere.

The relationship between the quartz and tourmaline at the project is not well understood though at least one event is closely related to the gold mineralization. Calmus (1992) and Perez (1992) described the gold as being in quartz, acicular tourmaline, and albite veins and breccias. It was noted (Perez, 1992) that two types of tourmaline exist: schorl and dravite but these are difficult to distinguish. There is some suggestion that the dravite is associated with the San Francisco zone while the schorl is generally barren of gold. If this can be verified, it could become a valuable exploration tool for the region. Horner (in Prenn, 1995) also noted the possibility of two or more types of tourmaline in the cobbles sampled in the stream beds. Horner believes that only one set of the tourmaline veins is associated with the gold and suggests that bismuth is also associated with one tourmaline quartz vein event.

Other metallic minerals associated with the deposit include trace to small amounts of chalcopyrite, galena, sphalerite, covelite, bornite, argentite-acanthite and pyrrhotite. Trace amounts of molybdenite and wulfenite have also been reported. Metal mineralization is low with copper reaching into the hundreds of ppm, arsenic reaches about 100 ppm, and antimony is rarely over 10 ppm. Petzite was recognized but tellurium values rarely reached over 10 ppm.

The San Francisco deposits are roughly tabular with multiple phases of gold mineralization. The deposits strike 60˚ west to 65˚ west, dip to the northeast, range in thickness from 4 to

50 m, extend over 1,500 m along strike and are open ended. The San Francisco deposits consisted of the El Manto, the San Francisco, the En Medio, and the El Polvorin deposits. All these deposits were later incorporated into the main San Francisco pit. The El Manto deposit (north pit), which lies north of the San Francisco (main pit), is tabular, strikes 65˚ west, dips relatively shallowly to the northeast, and ranges in thickness from 5 to 35 m. The En Medio deposit (in the main pit north of San Francisco) strikes 60˚ west, dips 19˚ to 45˚ northeast and varies in thickness from 4 to 20 m. The El Polvorin deposit (west pit) is a northwest extension of the San Francisco mineralization which strikes 65˚ west, dips moderately to the northeast and ranges in thickness from 4 to 20 m.

Alteration related to the mineralization consists of negligible to locally intense sericitization, coarse-grained pyritization, and rare local silicification. This alteration forms a halo extending a few metres from the mineral deposits, but may also be absent. Supergene alteration consisting of oxidation of pyrite to goethite is also common. Additionally, there is supergene alteration of feldspar to kaolin and sericite.

Analysis by Geomaque of 110 samples in seven mineralized zones showed a silver/gold ratio of less than 1 to 10, with very low values of zinc, copper, molybdenum, bismuth, antimony and mercury. Lead is occasionally high, but not above 1%. Gold shows a good correlation locally with arsenic and lead. However, none of the other elements are good indicators for gold.

10.0 EXPLORATION

A description of the historical exploration work conducted on the exploitation concessions is provided in Section 6.1.

TMM acquired the rights to explore the exploitation concessions on April 18, 2005, upon the signing of an option agreement between its Mexican subsidiary, Timmins, and Geomaque de Mexico. After a period of reviewing the available geological data previously compiled for the mine site and property, Timmins identified a number of exploration targets in and around the existing San Francisco pit, as well as a few secondary targets located on Timmins exploration concessions. A summary of the exploration targets in and around the existing San Francisco project as well as the secondary targets on the property was provided in the December, 2005 NI 43-101 report entitled "Technical Report on the San Francisco Mine Property, Estación Llano, Sonora, Mexico." This report was filed on the SEDAR website on April 28, 2006 by TMM.

Timmins conducted its first exploration drilling program on the San Francisco project in August and September, 2005 and the results of the program were also described in the December, 2005 NI 43-101 report.

The drilling program conducted by Timmins from September to November, 2006 was based primarily on the results of the 2005 drilling program and on the results of the historical drilling programs conducted by Geomaque and Fresnillo. The results of these earlier drilling programs indicated that there were areas to the north and northwest of the existing San Francisco pit and to the north of the La Chicharra pit where the mineralization appeared to continue beyond the former mining areas. Also, during its review of the geological mapping, and geochemical sampling data and after a re-interpretation of the structural controls of the mineralization, Timmins identified a number of favourable drill target areas where there was either a gap in the earlier drilling data or where no drilling had been conducted.

The 2006 exploration drilling program and its results were discussed in a February, 2007 Technical Report which was filed on the SEDAR website on February 27, 2007 by TMM.

In 2007, Timmins conducted a follow-up exploration program based on the results of the 2006 program. The results of the 2007 exploration program were contained in a press release dated November 28, 2007. The results of the 2007 exploration program are discussed below.

10.1 2007 EXPLORATION PROGRAM

During 2007, Timmins conducted field work and exploration drilling to evaluate the extent of the gold mineralization in the other zones on the property, but no mineral resource estimate from the 2007 work was available for inclusion in this study. A total of 5,123 m of exploration drilling were completed in 2007 which included 1,327 m of condemnation drilling. The total expenditures for the 2007 drilling program were approximately USD

629,000. The following paragraphs are based on information contained in the November, 2007 press release.

The 2007 exploration program to the north and northwest of the San Francisco and La Chicharra pits identified several new gold targets. The occurrences are associated with quartz veins and veinlets hosted in a highly sheared, metamorphic sequence of schists, granites, gneisses and gabbros. The geochemical trends which correlate with high resolution aeromagnetic data recently completed for Timmins by the Mexican Geological Survey, confirm that the gold occurrences are associated with at least two principal northwest trending zones identified as the west mineral trend and the east mineral trend.

The west mineral trend includes the La Trinchera, La Perdida, La Escondida, El Gallito Ingles, La Perra and El Socorro gold occurrences. All of these showings either outcrop or are near surface. Thr west mineral trend, which dips to the northeast, exceeds 3 km in width and 10 km in length, and remains open along strike.

The location of the San Francisco and La Chicharra deposits and several geochemical anomalies indicate that the dominant northwest trending structures acted as conduits for the mineralizing fluids. The mineralizing fluids are believed to be both postdeformational and associated with deformation since deposition occurs within favourable lithologies and along foliation planes within the metamorphic sequence. Stong gold geochemical anomalies suggest that there is the possibility of further gold mineralization at depth.

The La Pima silver occurrence, located 30 km northwest of the San Francisco pit, lies along the projected strike extension of the west mineral trend. The La Pima showing is related to the emplacement of an intrusive body into limestone and a sequence of calcareous sandstone and siltstone. Replacement and skarn mineralization consisting of calcite and siderite veinlets, and silver, lead and zinc veins, occurs regularly along a 2 km strike length.

A large portion of the west mineral trend is masked by a thick blanket of soils which have been pierced by outcrops of the metamorphic sequence. The absence of bedrock explosure led Timmins to complete a comprehensive geochemical sampling program over much of the west mineral trend. In total 656 soil samples and 67 chip samples were collected over the 10 km^2 grid. Geological mapping was also completed.

The east mineral trend includes the Cerro Gauna, Area 1B, La Playa, La Mexicana and Old Veta Tierra gold occurrences.

One of the most significant occurrences, Cerro Gauna, lies along the southwest extension of the trend. This sequence is exposed for a vertical distance of over 120 m from the desert floor to the top of the Cerro Gauna ridge. The granite and metamorphic sequence correlates with the mineralized units previously exploited and much of the remaining resource within the San Francisco pit area. A systematic geochemical sampling program was in the progress in the fall of 2007.

The granite extends north, east and northeast of Cerro Gauna trending toward the Area 1B and El Socorro occurrences. Quartz veining with pyrite and tourmaline are all common in the granite. The north extension of the granite, as inferred by areomagnetics and geochemical signatures, is buried by a thick alluvial cover.

Systematic geochemical sampling programs were previously completed on the Area 1B zone in the 1980s by Fresnillo and in the 1990s by Geomaque. Both programs confirmed the presence of a zone of highly anomalous gold values in a surface area measuring 800 m by 450 m which appeared to be associated with granite exposures on the north face of Cerro Gauna. Timmins completed a check sampling program which confirmed the results of the previous operators while a detailed geological mapping and systematic geochemical sampling was continued with the objective of locating drill sites.

Several geophysical programs have been completed on the San Francisco property by previous operators and by Timmins. The first aeromagnetic was completed during the mid-1990s by Geomaque. The survey was flown with 300 m high flight lines. While the survey clearly established that the San Francisco and La Chicharra deposits are related to a well defined, regional, northwest trending change in magnetic field gradient which extends along a 40 to 50 km long corridor, it appears to have failed to identify many more subtle anomalies. A review of the historical data by Timmins determined that the flight lines were too high to identify the subtle anomalies and as a result IP-R and ground magnetic surveys had been incorrectly located in a large number of cases. Timmins believes that the aeromagnetic and ground geophysical surveys completed in 2007 will help in identifying similar mineralized zones along the regional trend.

During May and June, 2007, Timmins completed a 1.227-line-km, high resolution, aeromagnetic and radiometric survey over the plus 40,000 ha San Francisco property. The company also acquired the raw data for an additional 1,569-line-km survey previously completed. In order to provide better resolution than the previous programs, flight lines were completed at elevations between 75 m and 100 m. The magnetic survey data clearly identified the significant gold occurrences associated with the regional structures and faulting which trend north and west, including the San Francisco and La Chicharra deposits. The gold deposits and occurrences appear to be associated with significant gradient changes from magnetic highs to lows. The radiometric survey data correlate well with the magnetic data. Timmins intends to use this relationship to help guide future exploration on the entire property.

To assist in further defining the targets and to obtain a signature of the geophysical responses related to the San Francisco deposit, Timmins contracted Zonge Enginerring and Research (Zonge) to complete a Natural Source AMT (NSAMT) survey over the known zones of mineralization. In total 12 km of NSAMT were completed over the known zones of mineralization. This information will be correlated with ground and aeromagnetic data, geochemical results and geological interpretation to assist in refining drill locations in new target areas. Zonge is in the process of completing the synthesis, filtering and interpretation of the data.

Further details of the 2007 exploration program will be presented at a later date. The information obtained during the 2007 program has not been used to update the mineral resource estimate which is included in this study.

The results of the 2007 core drilling program will be discussed in the next section of this report.

11.0 DRILLING

A description of the historical drilling conducted on the San Francisco project is provided in Section 6.1. Timmins conducted exploration drilling in 2005 and 2006, as described below, and this provided data for the resource estimate prepared in 2007.

Timmins also completed a drill program in 2007, the results of which have been publicly disseminated but which have not yet been incorporated into the present resource estimate. Micon understands a revised resource estimate is being compiled but, at the time of writing, this was not available for inclusion in the preliminary feasibility study.

In the fall of 2005, Timmins conducted a reverse circulation drilling program comprised of 14 holes on the San Francisco project site. The details and results of the drilling program were published previously in a technical report entitled "Technical Report on the San Francisco Mine Property, Estación Llano, Sonora, Mexico." This technical report was published on the SEDAR website on April 28, 2005 by TMM.

Between September and November, 2006, Timmins conducted a follow-up drilling program which consisted of 56 holes comprised of 28 diamond drill holes and 28 reverse circulation drill holes. While the drilling was primarily concentrated to the north and northwest of the present San Francisco pit and to the north and northwest of the existing La Chicharra pit, a number of widely spaced holes were drilled to test specific geological and geochemical targets around the San Francisco pit and to the south and west of the La Chicharra and La Severiana areas (see Figure 4.2).

A total of 13 diamond drill and five reverse circulation holes were focused on confirming the drill intersections of widely spaced Geomaque drilling in the former crusher area to the north of the San Francisco pit, particularly after a re-interpretation of the structural controls of the mineralization indicated that the projected trend of the mineralization was in this direction. These drill holes are concentrated between Sections 880NE and 1040NE, and cover an area of 280 m by 300 m. Diamond drill hole TFD-27 was drilled to twin the previous 2005 reverse circulation hole TF-13 in order to test the high grade mineralization in this hole. The assay results from hole TFD-27 indicated a length of 107.35 m averaging 3.346 g/t gold, influenced by very high gold grades in the weighted intervals. Three diamond drill and five reverse circulation holes were drilled around the area of the San Francisco pit to test favourable lithology and geochemical anomalies.

In the La Chicharra area, five diamond drill and two reverse circulation holes were drilled to test the continuity of the gold mineralization to the north of the existing pit and are concentrated between Sections 2500NE and 3100NE. Another six diamond drill and 13 reverse circulation holes were drilled to test geological and geochemical targets to the south, southwest and west of the La Chicharra pit. These holes were distributed between Sections 2500NE and 3600NE. Finally, two reverse circulation holes were drilled in the La Severiana area to test at depth gold anomalies in placer deposits.

The total estimated expenditure for the exploration program to the end of December, 2006 at the San Francisco property was USD 2,042,312. The 2006 exploration drilling program and its results were discussed in a February, 2007 Technical Report filed on the SEDAR website.

In 2007, Timmins conducted a follow-up exploration and condemnation drilling program which was based partly on the results of the 2006 program. The results of the 2007 drilling were contained in a press release dated November 28, 2007. The results of the 2007 exploration program are discussed below.

11.1 2007 DRILLING PROGRAM

During 2007, Timmins conducted exploration drilling to evaluate the extent of the gold mineralization in the other zones on the property. As noted above, the results have not been included in the present preliminary feasibility study. This information is presented here for the purpose of demonstrating that drilling is ongoing at the San Francisco property. A total of 5,123 m of exploration drilling were completed in 2007 which included 1,327 m of condemnation drilling. The total expenditures for the 2007 drilling program were approximately USD 629,000. The following paragraphs are based on information contained in the November, 2007 press release by TMM.

A total of 40 drilll holes were completed in 2007, including 11 condemnation drill holes.

In the west pit area a total of 7 drill holes were completed which totalled 972.25 m. One of the principal objectives was to test the down-dip extension and strike length of the high gold intersections encountered in drill hole TFD-27 completed during the 2006 drill program (110.75 m averaging 3.078 g/t gold). The 2007 west pit drilling program confirmed the continuity of the high grade since drill holes TFD-29, TFD-30, TFS-31, TFD-32 and TFD-33 all intersected lenses of granite and quartz breccias containing high gold values. The most significant intersection returned 88.80 m grading 1.29 g/t gold.

In the area of the La Chicharra pit a total of 9 drill holes were completed totalling 1,369 m. The results of this drilling extended the strike length by 300 m and confirmed the down dip extension of the La Chicharra deposit to at least 400 m. Hole TDF-61 was drilled 380 m north of the present pit limit with the results confirming the down dip extension of the deposit to at least 400 m.

Nineteen holes totalling 1,700 m of infill drilling were completed in the crusher area and, of this total, 341m in three drill holes were completed during the 2007 drilling program. This portion of the drilling program was designed to increase the confidence of the previously identified mineralized area by increasing the drilling density to be able to classify this material as a mineral resource.

Granite and gabbro are exposed along 400 m of the south wall of the San Francisco pit and as these rock types are two of the principal hosts of the gold-bearing veins and veinlets, a total

of six drill holes were drilled in this area. The six drill holes totalled 450 m and were drilled to test the down dip extent of the gold mineralization found in this area.

Timmins conducted a block model analysis of the San Francisco deposit and identified at least five zones where the drill hole density was not sufficient to satisfy the confidence levels for either an indicated or measured resource. Based on this information Timmins selected the two zones (Southeastern and Polvorines) which were recognized as being the most prospective for upgrading the resources from inferred to an indicated or measured category.

Two drill holes were completed southeast of the present pit adjacent to the waste dumps in order to confirm the the presence of gold mineralization intersected by previous operators. Hole TFD-60, which was drilled along Section 20E, returned 19.90 m grading 4.076 g/t gold from 5.0 m to 24.90 m. Hole TDF-67 was drilled to test the the down dip and strike extent of the same mineralized interval encountered in drill hole TFD-60. This hole, which was drilled 40 m northwest of TFD-60 along Section 40W, ended in 1.20 m (true width) grading 5.76 g/t gold and appears to have just encountered the mineralized zone when it was stopped.

Two drill holes (TFD-68 and TFD-69) were drilled southwest of the present pit in the Polvorines area. The two holes totalled 220 m and were drilled to increase the drill hole density and mineral resource confidence level in this area which previously hosted an inferred resource. Drill hole TFD-68 intersected an 8.10 m (true width) interval grading 2.069 g/t gold and drill hole TFD-69 intersected several mineralized intervals from 1 to 1.5 m wide between 70 and 115 m grading between 0.20 g/t and 0.58 g/t gold.

An 11-hole condemnation drilling program totalling 1,327 m was completed in the area west of the present leach pads. An area 500 m by 500 m was identified as being suitable for locating the future heap pads and/or operating facilities.

12.0 SAMPLING METHOD AND APPROACH

A description of the historical sampling methods conducted on the project is provided in Section 6.1. TMM, through its Mexican subsidiary (Timmins), conducted its initial exploration drilling program on the project in August and September, 2005 and instituted sampling procedures for the reverse circulation drilling program as a result. During the September to November, 2006 drilling program Timmins continued to use the sampling procedures instituted for the 2005 reverse circulation drilling program while instituting a further set of sampling procedures to cover the diamond drilling program. Micon examined both sets of drilling and sampling procedures during the site visit in September, 2006 and is satisfied that they were accurately carried out.

Timmins' 2006 exploration drilling program consisted of a total of 56 drill holes, 28 diamond drill and 28 reverse circulation drill holes, totalling 7,310.62 m. All drill holes were field logged and sampled as the holes were in progress. Once the drilling program was completed the information contained on the hand-written drilling logs (field logs) was transcribed into an Excel® spreadsheet. The Excel® spreadsheet contains the basic drill hole data, individual sample data and assay results, as well as the codes for the lithology, alteration and mineralization. This information was converted to an ASCII file to import it into the database which supports the present resource estimate completed by IMC for Timmins. Previous geological and mineralization interpretation was conducted based on cross-sections which were produced using an AutoCAD® software package.

In 2007, Timmins completed an exploration program which was based partly upon results of the 2006 drilling program. Information obtained during the 2007 program was not available for inclusion in this study. Further details of the 2007 exploration program will be presented separately, at a later date.

12.1 REVERSE CIRCULATION DRILLING

From the reverse circulation drilling, a portion of the material generated for each sample interval was retained in a plastic specimen tray created specifically for the reverse circulation program. The plastic specimen tray constitutes the primary reference for the hole in much the same was as the core does for the diamond drilling. The specimen tray was marked with the drill hole number and each compartment within the tray was marked with both the interval and number for the respective sequential sample it contained. Empty compartments were left for the locations where both the blank and standard samples were inserted into the sequential sample stream and two compartments were filled and identified for each duplicate sample.

Due to the nature of reverse circulation drilling, only rock chip fragments are produced which range from a very fine grained powder up to coarse chips 2 cm in size. Since the stratigraphic contact between the different rock units cannot be identified exactly the holes were sampled on equal 1.52 m (5 ft) intervals from the collar to the toe of the hole. The sample interval was chosen because it represented two samples per drill rod (3.04 m or 10 ft).

In general, this is considered to be the standard sampling length within the industry. It is not too large a sample or sampling interval to dramatically skew the assay results for a drill hole.

Samples were taken in the overlying alluvium as well as within the underlying rock units. The alluvium samples were subject to random assaying, whereas every sample originating from the underlying rock units was assayed. The recovery of the material during the drilling program was excellent, in the order of 90% to 95%, in both near surface sulphide-oxide and sulphide zones. Difficult drilling was encountered in at least two holes; TF-20 and TF-19. The first one was repeated 1 m away from the original site, while for the second one, difficulties prevented the hole from being finished to the programmed depth.

12.2 DIAMOND (CORE) DRILLING

Where core drilling is conducted, the sampling controls start after a run has been completed and the rods are pulled out of the drill hole. Once the core is removed from the core barrel it is placed in core boxes, with the length of each box depending on the type of core stored in it (2.40 m for HQ diameter or 3.00 m for NQ diameter). This follows standard procedures developed for core placement in the core boxes.

Small wooden tags mark the distance drilled in metres at the end of each run. In the case of the San Francisco drilling the drilling rods are measured in imperial units while the tags placed in the boxes and the core logging are measured in metric units. The drill hole number and box number are marked on each filled core box by the drill helper and checked by the geologist. Once the core box is filled at the drill site the box is covered with a lid to protect the core and the box is sent to the core logging facility for further processing.

For diamond drilling where core is produced, a unique record of the exact stratigraphic contact between the different rock units can be identified. Since identification of the exact stratigraphic contact between the different rock units can be established, the stratigraphic contacts are used as the primary basis for separation of the sampling intervals with the maximum sampling length within the stratigraphic unit restricted to approximately 1.0 m or 2.0 m and with no minimum size restriction. The 1.0 m or 2.0 m maximum sampling interval was chosen because it is generally regarded as the standard sampling length within the industry and in general it is not too large an interval to dramatically skew the assay results. However, in addition to the stratigraphic restrictions on the length of the core interval, the size of the sample may have been restricted because of the content or type of mineralization encountered in the drill hole. In general, the core recovery for the diamond drill holes in the San Francisco project was better than 98% and no core loss due to poor drilling methods or procedures was experienced.

A common feature in the sampling process for each of the drilling types is that a unique sample tag is inserted into the sample bag with each sample and each sample bag was marked with its individual sample number. The bags containing the blank and standard samples were added into the sequential numbering system prior to being shipped to the assay preparation facility of ALS Chemex Laboratories (ALS Chemex) and later to the Sonora Sampling

Preparation, S.A. de C.V. (SSP) facility which is the preparation facility for Acme Analytical Laboratories Ltd. (ACME). Both these preparation facilities are located in Hermosillo. Samples identified as the field-duplicate samples during the reverse circulation drilling were split into two separate sequentially numbered samples during the sampling process at the drill.

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

Details of sample preparation, analysis and security are not available for the historical work conducted by Geomaque on the project in the 1990s, although a laboratory was located on the mine site and some of the sample preparation equipment was available for Timmins to use during its exploration programs.

Prior to its initial reverse circulation drilling program in August and September, 2005, Timmins did not conduct any sample preparation or analysis, since no samples were collected.

13.1 SAMPLE COLLECTION AND TRANSPORTATION

The reverse circulation and core drilling sampling was conducted by a team of two or three geologist assistants under close supervision of the TMM/Timmins staff geologists in charge of the program on site. The staff geologists were responsible for the integrity of the samples from the time they were taken until they were delivered to the preparation facilities in Hermosillo.

13.1.1 REVERSE CIRCULATION DRILLING

The sample collected at the drill site was discharged from the drill hole through a hose into a cyclone where it was collected in a plastic pail. Sampling of the material generated during the reverse circulation drilling was conducted at the drill rig using a stainless steel riffle splitter if the material was dry and using a rotary splitter situated below the cyclone if the material was wet. The cyclone and splitters were cleaned off between samples and, in the case of wet samples, the cyclone and splitters were blown out using compressed air and also washed out between each sample using clean water. Using a 12.5 cm drill bit and a sample length of 1.52 m, it is estimated that the original sample weighed 48.3 kg prior to recovery. It is estimated that the average recovery was between 90% and 95%, which would indicate that the mass of the recovered sample varied between 42 and 45 kg.

The method of splitting the samples derived from the reverse circulation drilling was as follows:

1) If the sample was dry, the entire sample interval was collected in a bucket and then passed through the riffle splitter twice before the final sample of 21- to 23-kg was collected with the remaining 21- to 23-kg rejected. The 21- to 23-kg sample was subjected to a second split to obtain two samples of 10- to 12-kg (an original and a witness sample). The geologist or one of his helpers (under supervision) had previously marked the drill hole number and sample number on the plastic sample bags and inserted the sample tag in the sample bag for the original sample. Both bags were closed and sealed at the drill with plastic tie wraps and transported to the camp facilities.

2) If the sample was wet, it was discharged to a cyclone and then passed through a rotary cone splitter to divide the sample into two equal portions, one of which was automatically rejected and the other collected and simultaneously split into two equal halves by means of a mechanism designed for this purpose and installed in the lower portion of the rotary splitter (See Figures 13.1 and 13.2). The two samples were collected in fabrine (micropore) sample bags to allow retention of the solids and the slow dissipation of the drilling water through the pores in the sample bags without sample loss. In all cases, a flocculent was used to settle the solids including the fine portion prior to tying the fabrine bag. The outside of each sample bag was marked with the sample's individual number which corresponded to the number on the sample tag which was inserted into the sample bag.

Figure 13.1
Rotary Cone Splitter Set-up on Drill HoleTF-20 R

Photograph provided by Timmins

Figure 13.2
Close-up of the Final Discharge Splitter Set-up for a Wet Sample on Drill HoleTF-20 R

Photograph provided by Timmins

All samples from the reverse circulation drilling were prepared at the drill site by the TMM/Timmins staff geologists and their assistants. Several times per day a truck was dispatched to take the samples to the secure camp storage facilities which were located in the Geomaque assay laboratory. These laboratory facilities are currently inactive.

A truck was dispatched twice a week to deliver the samples to the Hermosillo assay preparation laboratory of ALS Chemex for the first five reverse circulation drill holes while the remaining samples for the drilling program were shipped to the SSP facility, also in Hermosillo.

Sample bags containing the blank and standard samples were added into the sequential numbering system prior to being shipped to the assay preparation facility in Hermosillo. Samples selected as duplicates were split into two separate sequentially numbered samples during the sampling process at the drill.

13.1.2 DIAMOND (CORE) DRILLING

For the core drilling, a truck collected the core boxes from each drill site at regular intervals during the day. The boxes were loaded into the truck and placed in a crisscross pattern and then secured to the truck by ropes to prevent movement on the short drive back to the core logging facilities at the camp.

Once the core boxes arrived at the logging facility, the boxes were laid out in order, the lids were removed and the core was washed to remove any grease and dirt which may have entered the boxes. The depth markers were checked by the geologist and the depth “from” and “to” for each box was noted for both, the top and the bottom covers of each core box to ensure that the boxes were correctly recorded.

For the 2006 drilling program, the entire length of the drill hole was sampled from where the hole first intersected bedrock to the toe of the hole. The standard sample intervals varied from 1 to 2 m in conjunction with the geological and mineral features observed during the logging. Some samples were limited to geological boundaries that were less than 1 m, but this was only an occasional occurrence. The geologist who was logging the core would begin by examining the core to ensure it was intact. During the core logging process the geologist would define the sample contacts and designate the axis along which to cut the core with special attention placed on the mineralized zones to ensure representative splits. The sample limits were marked on the core and also on the side of the box, including the sample number. The sample limits were imported into an Excel spreadsheet, which defined the sample number and intervals. The sample numbers were marked on the box next to the sample limits at the beginning and end of each sample interval or at the centre. See Figure 13.3 for a photograph of the geologist logging the core at the San Francisco project.

Figure 13.3
Core Logging at the San Francisco Project

Photograph provided by Timmins

Once the core was logged and the samples marked, the core boxes were brought to area where a diamond saw had been set up to cut the samples. Two core splitters and their helpers processed the samples by using the diamond saw to cut the core in half. Once the core was sawn in half, one half was placed into a plastic sample bag and the other half was returned to the core box. The geologist or his assistant had previously marked the sample bags with the sample number and inserted the individually numbered sample tag in the plastic bag. A geologist supervised the sawing of core to ensure that the quality of the core sample remained high and that no mistakes were introduced into the system due to sloppy practices. The boxes containing the remaining half core were stacked, with lower numbers at the bottom and the higher numbers at the top, and stored on site in a secure core storage facility. See Figure 13.4 for a photograph of the core sawing area.

Figure 13.4
Core Sawing at the San Francisco Project

Photograph provided by Timmins

The sample bags were placed into large canvas sacks with generally 7 to 10 plastic sample bags per sack. These sacks were secured and then shipped to the laboratory. A truck was dispatched twice a week to deliver the samples to the ALS Chemex facility (the first three diamond drill holes) and later to the SSP facility for sample preparation.

13.1.3 GENERAL PROCEDURES

As part of TMM/Timmins' QA/QC procedures, a set of samples comprised of a blank sample, a standard reference sample and a field-duplicate sample were inserted randomly into the sample sequence. The insertion rate for the blanks, standards and duplicate samples was one in every 25 samples.

13.1.3.1 Blank Samples

The blank sample used for the San Francisco drilling program was obtained from a tonalite dyke that crops out at the southwestern extent of the existing San Francisco pit. Geologically, the rock unit is younger than both the host rock of the gold mineralization and the mineralizing events in the region. A geologist currently working for Timmins and previously for Geomaque considered the material in the dyke as barren rock and this was verified by assaying.

A 400 kg sample was obtained from the tonalite dyke in the outcrop, and washed thoroughly with fresh water to remove any dust that could have contaminated the rock because of its proximity to the San Francisco pit. Once the rock was washed, it was dried and crushed to obtain material down to minus 1 inch in size. The material was homogenized in a gyratory tank for 12 hours and finally passed through a splitter to be distributed in 1-kg sample bags which were sealed with tie wraps.

Five of the 1-kg samples were randomly selected and sent to different laboratories to be assayed for gold using the fire assay method and multi-element analysis using multi-acid digestion. All five laboratories were given a sample which corresponded to number 636741. All samples were prepared using the same procedure and the selected laboratories to which the samples were sent for assaying were ALS Chemex, ACME, Accurassay Laboratories (Accurassay), TSL Laboratories Inc. (TSL), and SGS Mineral Services (SGS), all of which are certified laboratories based in Canada.

Table 13.1 shows the gold assay results for the five samples including the assay results for the group of elements of interest. From the assay results obtained from five separate independent laboratories, which show a very low to negligible content for gold, it is apparent that the material from the tonalite dyke can be used as a local blank for the drilling program.

Table 13.1
Assays for the Blank Sample Used For the 2006 San Francisco Project Exploration Drilling Program

Laboratory	Gold (ppb)	Silver (ppb)	Arsenic (ppm)	Copper (ppm)	Molybdenum (ppm)	Lead (ppm)	Tungsten (ppm)	Zinc (ppm)
ALS Chemex	6,	0.75	1.6	84	0.75	31.2	2	122
ACME	10	<2	<0.02%	<0.00004	<0.001%	<0.02%	<0.01%	<0.01%
SGS	7	<2	<3	26	1	75	<10	148
TSL	<5	0.3	2	43	0.7	32.2	1.5	144
Accurassay	11	<1	12	81	14	32	<10	123

Table provided by Timmins

13.1.3.2 Standard Samples

For preparation of the In House Standard Reference Material (IHSRM) sample, the retained mineralized samples generated during the 2005 drilling program and assayed by ALS Chemex were used. Timmins engaged six different qualified laboratories to assay the prepared samples to determine the average grade of the sample and a total of 18 randomly selected samples were sent to the six laboratories.

The preparation of the IHSRM was contracted to the SSP preparation facility based in Hermosillo, Sonora, which supplies preparation services to ACME, IPL and Jacobs Laboratories. All samples were prepared using the same procedure as is shown in Figure 13.5, and the selected laboratories to which the samples were sent for assaying were ALS Chemex, ACME, International Plasma Labs Ltd. (IPL), Accurassay, TSL, and SGS, all of which are certified laboratories based in Canada. The samples from the 2005 drilling program which were used to generate the IHSRM sample are recorded in Table 13.2.

Table 13.2
2005 Drilling Samples used to Generate the 2006 In House Standard Reference Material Sample

Drill Hole Number	Sample Number	ALS Chemex (g/t gold)	Sample Weight (kg)
TF-01	561195	0.563	6.8
TF-11	636077	0.386	7.6
TF-12	561086	0.483	10.4

Table provided by Timmins

Because of the close relationship between the gold grades of samples 561195 and 561086 it was decided to mix these samples in order to obtain one sample to use as the IHSRM sample. To conduct the assays, two samples of the combined sample material, with different sample numbers, were sent to each of the six laboratories for a total of 12 samples. The two original sample numbers that were to be used were 561195 and 561093. However, in the case of sample 561093, prior to shipping the sample was considered to be unreliable to be used as a standard since the bag was found broken and probably contaminated. As a result, material from a secondary bag labelled 561086 was shipped to the laboratories as the second sample.

The assay results for the two samples generated from the combined IHSRM sample are shown in Table 13.3 and graphically shown in Figure 13.6.

Figure 13.5
Preparation Procedures used to Generate the In House Standard Reference Standard Sample

TIMMINS GOLD CORP.
SAN FRANCISCO PROJECT
SCHEDULE FOR STANDARDS MATERIAL PREPARATION
INTERNAL PROCEDURE

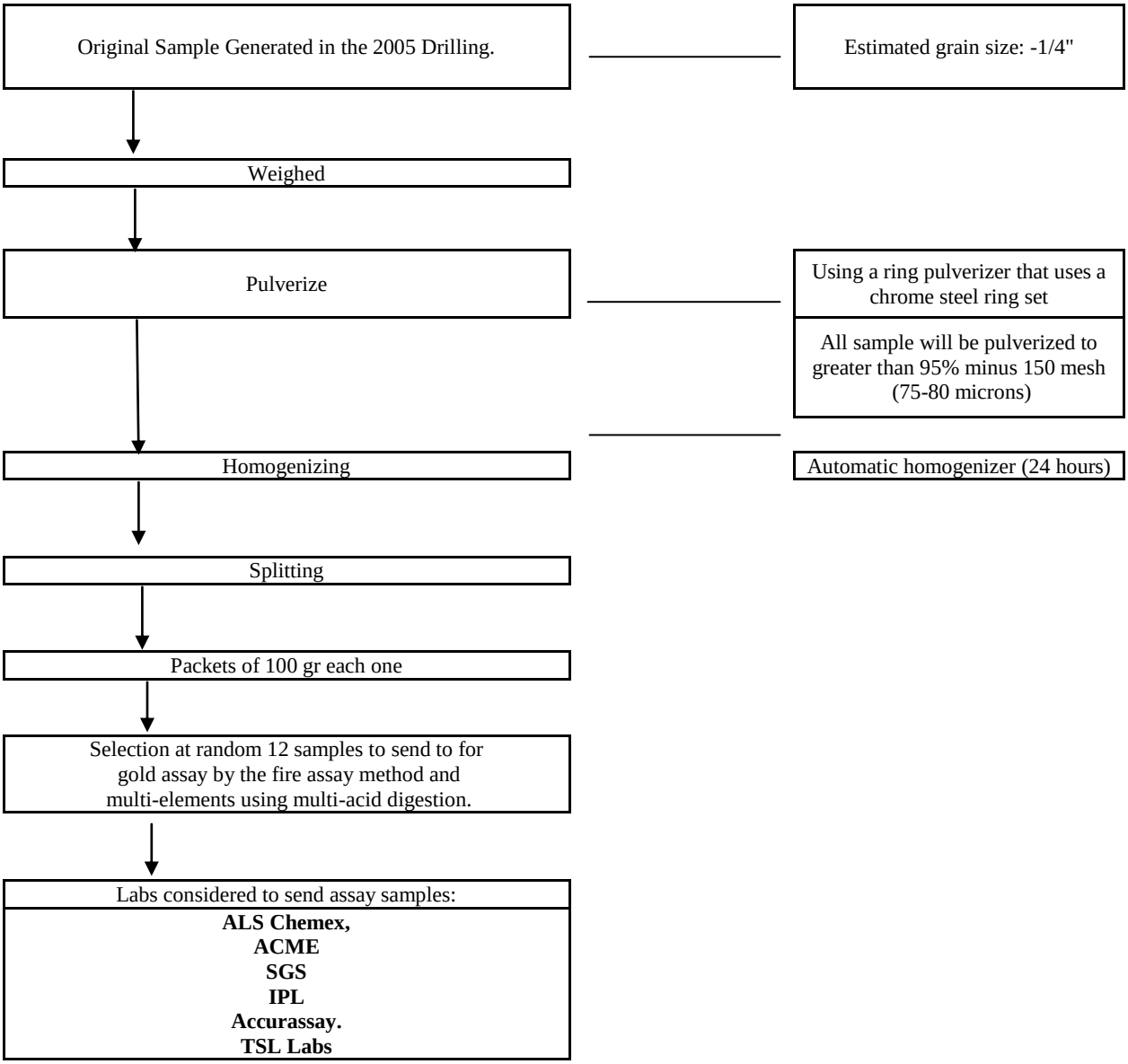


Figure provided by Timmins

Table 13.3
Assay Results for Combined 2006 In House Standard Reference Material Sample

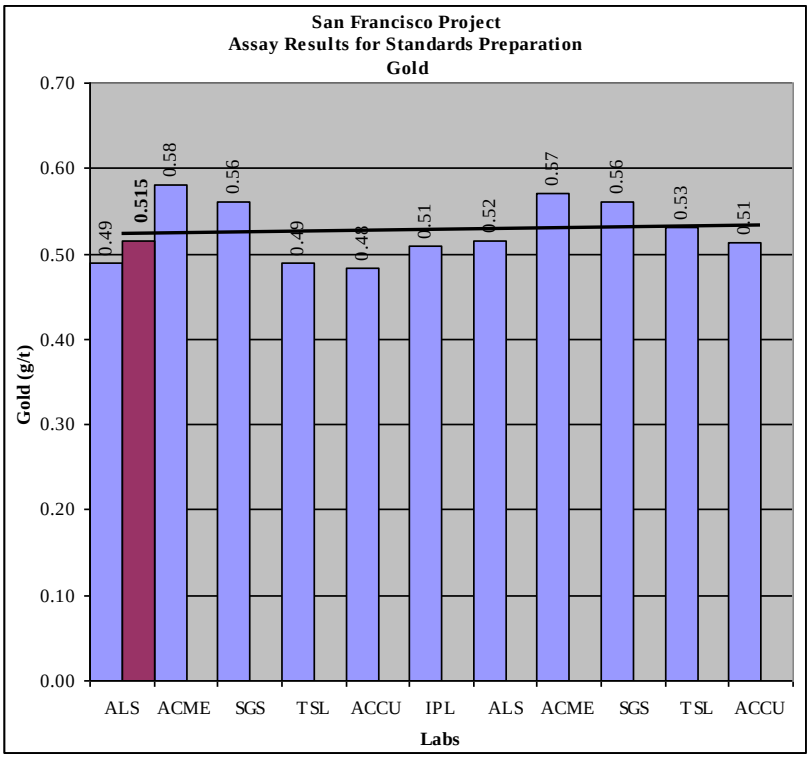
Laboratories	Original Sample Data + ALS Chemex Assays			Assay Results for Duplicate Samples							
	Sample Number	Weight (kg)	Gold (g/t)	Gold (g/t)	Varation (g/t)	Variation (%)	Silver (g/t)	Arsenic (ppm)	Copper (ppm)	Lead (ppm)	Zinc (ppm)
ALS	561195	10.40	0.483	0.489	0.026	4.98	0.830	1.2	12.6	14.1	33.0
ACME				0.580	-0.065	-12.70	<2	<0.02	0.0	<0.02	<0.01
SGS				0.560	-0.045	-8.82	<2	<3	8.6	23.0	35.0
TSL				0.490	0.025	4.79	0.800	2.0	13.5	15.8	39.0
Accurassay				0.483	0.032	6.15					
IPL				0.510	0.005	0.90	1.0	<5	15.0	30.0	30.0
ALS	561086	6.80	0.563	0.516	-0.001	-0.27	0.390	1.0	12.0	10.8	34.0
ACME				0.570	-0.055	-10.76	<2	<0.02	0.0	<0.02	<0.01
SGS				0.560	-0.045	-8.82	<2	<3	8.8	14.0	35.6
TSL				0.530	-0.015	-2.99	0.400	2.0	13.2	11.5	38.0
Accurassay				0.514	0.001	0.12					
IPL				0.440	0.075	14.50	<0.5	<5	16.0	24.0	30.0
Estimated Gold Grade			**0.515**	**0.520**	**-0.006**	**-1.08%**					
Statistic Parameters											
Samples				12							
Maximum				0.580							
Minimum				0.483							
Average				0.520							
Mean				0.515							
Mode				0.560							
Standard Deviation				0.042							

Table provided by Timmins

As can be observed in Table 13.3 and Figure 13.6, most of the assays are in agreement with the estimated grade of 0.515 g/t gold for the mixture of the two samples. In two cases, the variance between the original sample and the mixed sample to be used as the IHSRM sample returned positive and negative variances of 13%. As the variance is not for the same laboratory it is believed that this bias may have been produced during preparation of the sample by SSP in Sonora.

In the case of the lower gold sample selected to be used as an IHSRM sample (Sample number 636077 at 0.386 g/t gold), the variance is stronger than the combined sample used for the higher grade. All of the assay laboratories returned lower assays for this sample than the original ALS Chemex assay which may indicate that this sample contains a higher nugget effect than some of the other samples which may have been selected to act as the standard reference sample. Micon recommends that as a check, this sample be assayed by the screen metallics procedure.

Figure 13.6
Assay Results for the Combined In House Standard Reference Standard Sample



Graph provided by Timmins

Table 13.4 and Figure 13.7 show the assay results from the six laboratories for sample 636077 which is the second IHSRM sample prepared by Timmins.

A Standard Reference sample, with a grade comparable to the higher grade material within the San Francisco deposit, was purchased from Proveedora de Laboratorios del Norestes, S.A. de C.V., distributors of Standard Reference Material samples prepared by Rocklabs Ltd. of Auckland, New Zealand. The Standard Reference sample purchased was Standard OxG46, and a copy of the label as it appears on the jar containing the Standard Reference sample is shown in Figure 13.8.

Table 13.4
Assay Results for the 2006 In House Standard Reference Material (Sample 636077)

Laboratories	Original Sample Data + ALS Chemex Assays			Assay Results for Duplicate Samples							
	Sample Number	Weight (kg)	Gold (g/t)	Gold (g/t)	Varation (g/t)	Variation (%)	Silver (g/t)	Arsenic (ppm)	Copper (ppm)	Lead (ppm)	Zinc (ppm)
ALS	636077	7.6	0.386	0.237	0.149	38.60	0.58	3.3	35.1	17.6	41.0
ACME				0.280	0.106	27.49	<2	<0.02	0.0	<0.02	<0.01
SGS				0.256	0.130	33.68	<2	<3	34.9	23.0	42.0
TSL				0.210	0.176	45.60	0.600	2.0	39.9	19.5	45.0
Accurassay				0.351	0.035	9.07					
IPL				0.166	0.166	43.01	<0.50	<5	40.0	35.0	36.0
Estimated Gold Grade			**0.386**	**0.259**	**0.127**	**32.90%**					
Statistics Parameters											
Samples				6							
Maximum				0.351							
Minimum				0.210							
Average				0.259							
Mean				0.256							
Mode				-----							
Standard Deviation				0.056							

Table provided by Timmins

Figure 13.7
Assay Results for the In House Standard Reference Standard (Sample 636077)

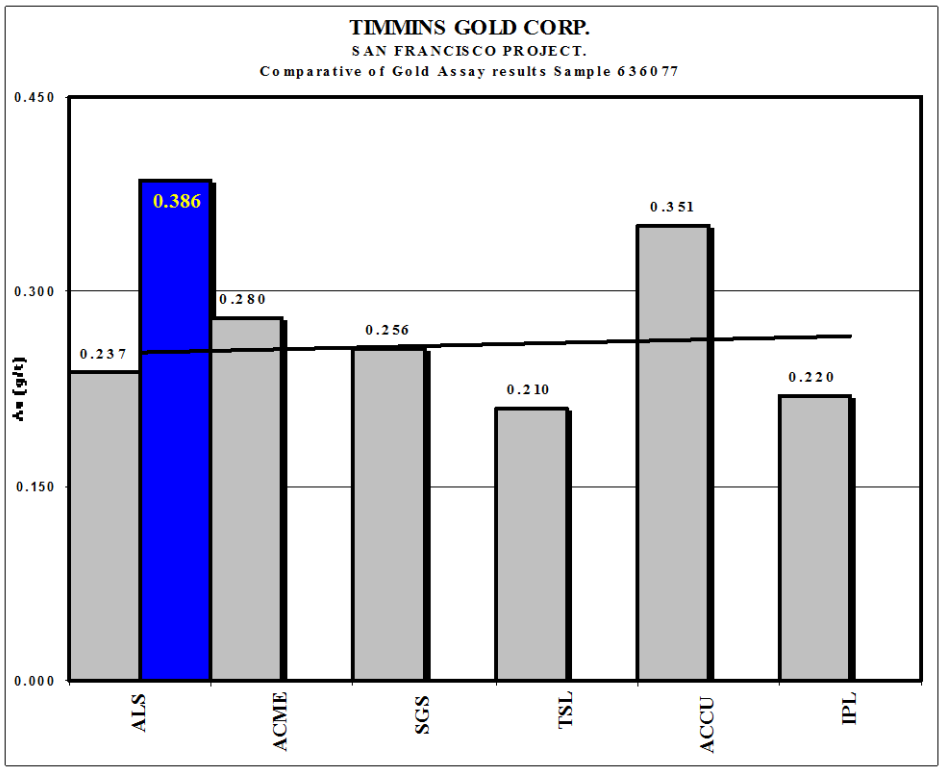


Figure provided by Timmins

Figure 13.8
Label for Jar Containing Standard Reference Sample OxG46

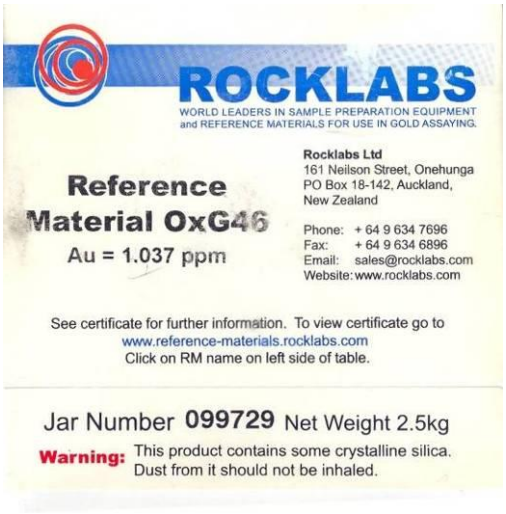


Figure provided by Timmins

13.1.3.3 Duplicate Samples

For the reverse circulation drilling, samples which were identified for duplication (i.e., field-duplicates) were processed and split in the same way as the regular samples taken on either side of them. However, the final 21- to 23-kg sample was subjected to a further split in the field which yielded two 10.5- to 11.5-kg samples in the case of dry samples. In the case of wet samples, they were dried and then passed through the riffle splitter to obtain a second (duplicate) sample of approximately the same mass as the original. The duplicate samples were given sequential numbers and submitted as two separate samples for the purpose of assaying so that the receiving laboratory did not know it was receiving duplicate samples.

For the diamond drilling, samples which were identified as the duplicate samples were sawn in half, as were the regular samples, and then the half which was identified as the portion to be sent for assaying was further sawn in order to obtain two quarter samples. The quarter samples were each individually placed in separate plastic bags. One of the quarter samples was identified as the original sample and a tag with the sample number was placed in the bag with the sample. The second quarter core sample was identified as the duplicate and a sample tag with the next consecutive sample number was placed in the sample bag.

13.1.3.4 Preparation Laboratories

For the first part of the drilling program, Timmins sent the samples to be prepared at the ALS Chemex preparation laboratory in Hermosillo. The samples for diamond drill holes TFD-01, TFD-02, and TFD-03 as well as the samples for reverse circulation holes TF-15, TF-16, TF-18 and TF-20 R were sent directly to the ALS Chemex facility. However due to the extended turn around time for the assays, which was in excess of 50 days, ACME was chosen because of its shorter turn around time. ACME's preparation facility is SSP and the remaining

diamond and reverse circulation drilling samples were prepared at this facility prior to shipping the prepared samples to the ACME laboratory in Vancouver, British Columbia for final analysis. The sample preparation procedures for both ALS Chemex and SSP are outlined below.

ALS Chemex Preparation Facilities

The ALS Chemex preparation laboratory in Hermosillo is ISO 9001:2000 registered. ALS Chemex has attained ISO 9001:2000 registration at all of its North American and Peruvian laboratories. The ALS Chemex Vancouver assay laboratory has obtained its ISO 17025:1999 accreditation.

Once the sample arrives at ALS Chemex's preparation laboratory in Hermosillo it undergoes the following process:

1) A bar code is attached to the sample. The bar code is scanned and the weight of the sample along with the date, time, equipment used, and operator name is recorded. Sample integrity is guaranteed by scanning the label at every stage in the process.

2) All samples are sent to the drying oven. For rocks, rock chips, core and other coarse samples a drying temperature within the range of 110˚C to 120˚C (230˚ to 250˚F) is used. With soils, silts, sediments and other fine samples the drying temperature is limited to 60˚C (140˚F). All samples are logged as to their positions on the moveable drying racks prior to being rolled into the drying oven and once dry they are returned to their original sample bags.

3) After drying, the samples are sent to the crushing area for preparation. The crushing area is comprised of three oscillating jaw crushers, which reduce the sample to a fineness of 70% passing a 10 mesh screen. Because the oscillating jaw crusher provides enhanced crushing by ensuring that the sample receives continuous grinding as it passes between the plates, the sample usually needs only one pass through the crusher to reduce it to the required fineness. The sample is split to 250 g, using a stainless steel riffle splitter prior to it being shipped to the next step in the process.

4) Once the crushing has been completed the sample is passed on to the pulverizing area. ALS Chemex uses a ring and puck pulverizer (low chrome steel ring set) to grind the sample further and achieve a very fine grind. During this stage of the process the 250-g crushed sample is reduced to a fineness of greater than 85% passing a 200 mesh (75 microns) screen.

5) Once the sample has been pulverized it is split into two portions using a stainless steel splitter. The split portion which is forwarded to Vancouver for assaying is placed in a yellow packet and the portion retained as a master sample is placed in a brown packet which is placed in storage.

6) Once the packet containing the sample is ready for shipment it is scanned using the bar code assigned to the sample and placed into a box. Once the box is full, it is labelled with another bar code and the box is scanned into the ALS Chemex computer network in order for the laboratory in Vancouver to receive the details concerning the contents of the box and assay procedures to be conducted on the samples. The retained portion of the sample goes through the same scanning process and is packed and stored for future use by the client.

7) From Hermosillo the portion of the sample sent to Vancouver, British Columbia, Canada, for assaying is shipped by United Parcel Service (UPS). UPS usually trucks the sample shipment to the border town of Nogales, where the shipment enters the United States (USA). Once the shipment arrives in the USA, it is shipped by air to Vancouver, arriving within 24 hours of having left Hermosillo 85% of the time.

Assaying of the San Francisco drilling samples was undertaken using using a combination of fire assay and atomic absorption, specifically ALS Chemex method Au-AA23. In the fire assay procedure, typically the samples are mixed with fluxing agents including lead oxide, and fused at high temperature. The lead oxide is reduced to lead, which collects the precious metals and, when the fused mixture is cooled, the lead remains at the bottom, while a glass-like slag forms at the top. The precious metals are separated from the lead in a secondary procedure called cupellation. The final technique used to determine the gold and other precious metals contents of the residue can range from the use of a balance, in the case of very high grade assays, to atomic absorption. ALS Chemex recommends a maximum charge weight of 30 g for optimum gold recoveries for most sample matrices, according to the published information in its schedule of Services and Fees. The lower and upper limits for gold detection using ALS Chemex method Au-AA23 are 0.005 and 10 ppm gold. The samples above the upper limit are re-assayed in the final step by the ALS Chemex gravimetric method (Au-GRA21) with a detection range in the order of 0.05 to 1,000 ppm gold.

ACME SSP Preparation Facilities

ACME conducts its sample preparation through the SSP preparation laboratory facilities in Hermosillo and, while SSP does not have its own ISO certification, it operates under ACME's ISO certification. The ACME Vancouver assay laboratory was the first assaying laboratory in North America to be accredited under ISO 9002 and ACME is currently registered with ISO 9001:2000 accreditation.

To conduct the assay analysis on the San Francisco drilling samples, ACME used its Group 3B Au method, i.e., 30 g/t Ag inquart fire assay fusion with Inductively Coupled Plasma Emission Spectroscopy (ICP-ES). In the fire assay procedure, typically the samples are mixed with fluxing agents including lead oxide, and fused at high temperature. The lead oxide is reduced to lead, which collects the precious metals and when the fused mixture is cooled, the lead remains at the bottom, while a glass-like slag forms at the top. The precious metals are separated from the lead in a secondary procedure called cupellation. The final

technique used to determine the gold and other precious metals contents of the residue can range from the use of a balance, in the case of very high grade assays, to atomic absorption. For its Group 3B assays, ACME uses a maximum charge weight of 30 g for optimum gold recoveries. The lower and upper limits for gold detection using the ACME method Group 3B are 2 ppb and 1,000 ppb gold. The samples above the upper limit (1,000 ppb) are re-assayed in the final step by the ACME method Group 6B (Fire Assay) with a lower detection range in the order of 0.01 g/t gold. See Figure 13.9 for the ACME preparation methods and specifications for the Group 3B sample analysis.

Figure 13.9
ACME Methods and Specifications for the Group 3B Sample Analysis

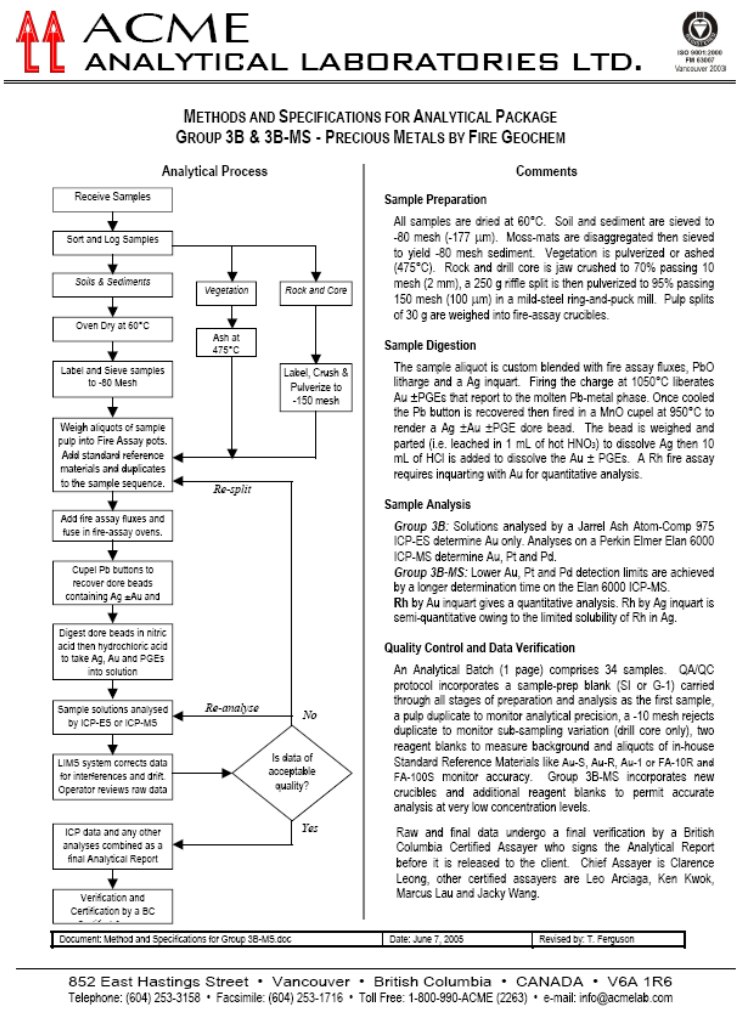


Figure provided by Timmins

13.2 RESULTS OF THE 2006 QA/QC PROGRAM

13.2.1 CHECK ASSAYING

A total of 289 samples were chosen from the reverse circulation (102 samples) and diamond drilling (186 samples) and sent to a second laboratory in order to check the assays against the ones obtained from ACME. Three of the samples were possibly contaminated and the results were not used in the interpretation of the check assaying. The laboratory chosen to conduct the check assaying was ALS Chemex in Vancouver, B.C. The samples were comprised of mineralized material as well as samples which were derived from the zone of influence surrounding the mineralized zone either above or below. Most of the randomly chosen samples were low grade material, with 7% of the ACME assays beneath the lower detection limit of 0.002 g/t and 33% of the ALS Chemex assays below the lower detection limit of 0.005 g/t.

Table 13.5 shows the correlation between the mean grade for ALS Chemex assays and ACME assays for the duplicate pulps for both the reverse circulation and the diamond drilling. Check assay results for the individual samples are given in Appendix I.

Table 13.5
Check Assaying Results for the 2006 San Francisco Drilling

Number of Samples	286
Acme Analytical Mean Grade	0.062 g/t gold
ALS Chemex Mean Grade	0.060 g/t gold
Difference Between Means	0.002
Mean Difference %	3.83%
Correlation Factor	0.9943

Table provided by Timmins

Figure 13.10 is a scatter plot of the correlation between ACME and ALS Chemex's assays for the same pulp samples. Samples 619194 (TF-26), 619285 (TF-28) and 643787 (TFD-09) were removed from the interpretation of the check assay data base due to possible contamination of the sample either during one or both assay procedures, or to nuggety gold in the sample. These samples need to be reassayed to determine the reason for the wide differences in the assays. The removal of these samples does not significantly impact the data used in the interpretation of the check assays as the 3 samples represent only 1.01% of the total samples (289) used for this analysis. Once the 3 samples were removed the regression line drawn between the gold check assays shows a correlation coefficient of 0.994 (Figure 13.10) which could be considered as indicating that there is no significant bias between the grades of ACME and ALS Chemex. A difference of 3.83% between the average grades of the 286 gold assays appears to indicate that there is probably a very low bias in the sampling preparation or internal ACME laboratory procedures. This is probably not significant as it is more than likely accounted for by the differences in the assaying procedures at the laboratories and the differences in the precision of the instrumentation which was used. Any appearance of a bias may be also aggravated due the high number of assays below the detection limits of the two assay laboratories.

Figure 13.10
Scatter Plot for Gold Acme Analytical Check Assays vs ALS Chemex Assays

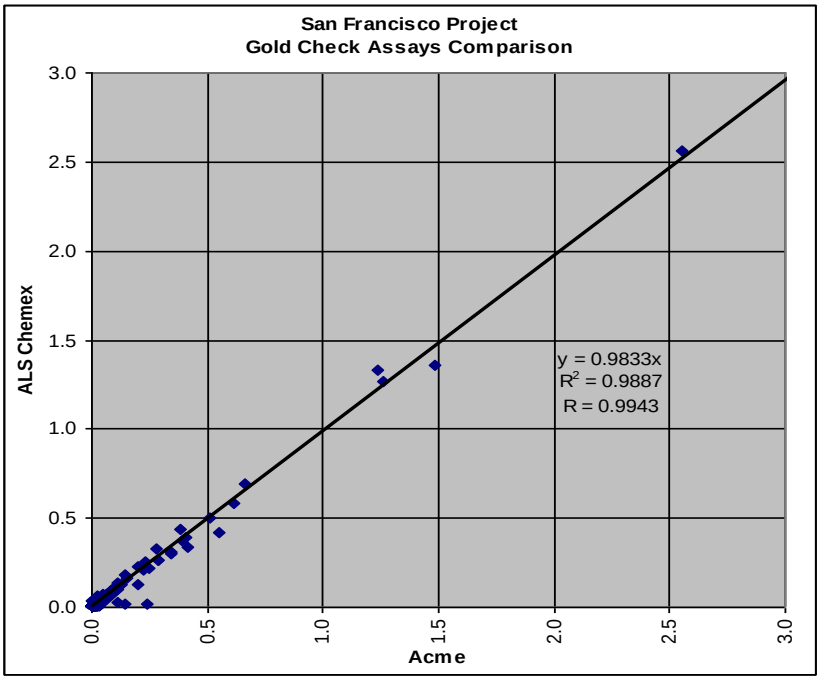


Figure provided by Timmins

13.2.2 STANDARDS

As noted earlier, in addition to purchasing a commercial Standard Reference sample to add to the sample stream Timmins created its own In House Standard Reference samples. These standards were added to the sample stream as a check of the quality of the assay preparation and assaying techniques of the laboratories to which the samples were sent.

Table 13.6 shows the results of all the gold assaying conducted by ACME on the standard reference samples used for the 2006 drilling program on the San Francisco project. Standard Reference Sample assays for the individual samples sent to the assay laboratory are given in Appendix II.

Table 13.6
Results of ACME Assaying on all of the Reference Standards

Standard Id	636077	561093 + 561195	OxG46	Total
Number of Samples	68	129	30	227
Standard Grade (g/t)	0.259	0.520	1.037	0.510
Average Grade (g/t)	0.228	0.527	0.994	0.499
Absolute Difference	-0.031	0.007	-0.43	-0.011
% Difference	-13.79	1.24	-4.38	-2.30
Correlation Factor				0.9628

Table provided by Timmins

Figure 13.11 is the scatter plot which shows the best fit regression line drawn for the gold assaying conducted by ACME on the standard reference samples.

Figure 13.11
Scatter Plot for Gold Reference Standard Assays vs ACME Assays

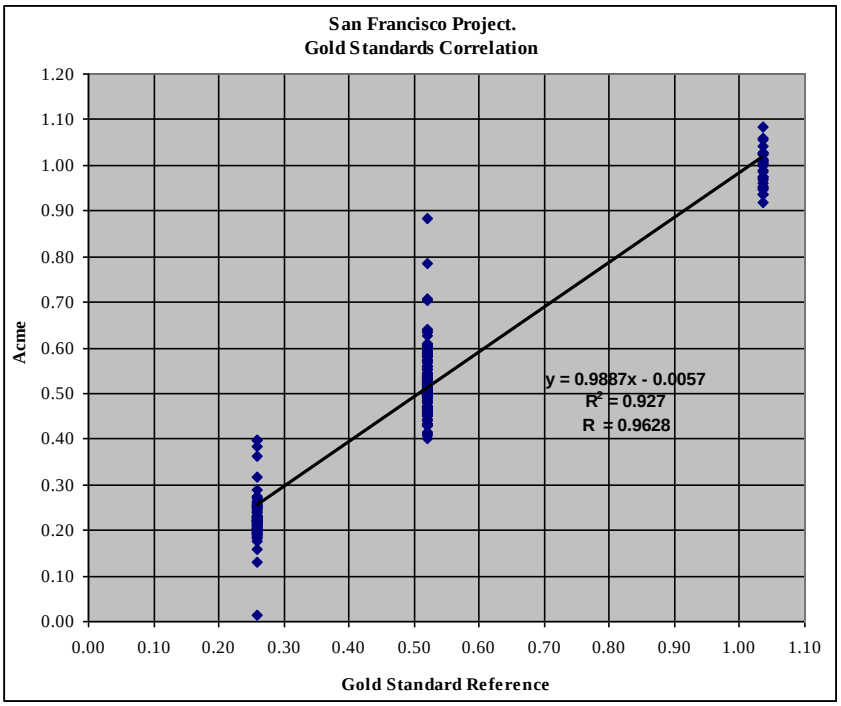


Figure provided by Timmins

If all the sampling is accounted for, the average ACME standard reference sample assaying shows a correlation factor of 0.963 (Table 13.8 and Figure 13.11) and a percentage difference between the ACME assay and the individual reference standard assays of between minus 13.79% for in-house standard reference sample 636077, 1.24% for the combined in-house reference sample (561093 + 561195) and minus 4.38% for the purchased standard reference sample OxG46. In general, and with the exception of the combined in-house standard reference sample, the ACME sampling averaged below the standard reference samples assay results. With the exception of the in-house standard reference sample 636077, the results of ACME assaying of the standard reference samples, suggest that there is no significant bias in the sample preparation or internal ACME laboratory procedures. In the case of standard reference 866077, the number of samples assayed was low and, as pointed out earlier, this sample may contain a pre-existing bias due to a nugget effect. Micon recommends discontinuing the use of this sample until further assaying using the screen metallics procedure indicates that it is not affected by coarse grained gold.

13.2.3 BLANKS

During the 2006 drilling program, a total of 224 blanks were added into the sample sequence generated at the San Francisco project. Of the total of 224 blanks, 104 were generated for the

diamond drilling portion of the program and 120 were generated for the reverse circulation drilling.

Table 13.7 summarizes the overall assay results for the blank assays. Assay results for each of the individual blank samples used in the drilling program are given in Appendix III. It should be noted that one of the blank samples (643461 for drill hole TFD-06) was lost and one blank sample (728399 for drill hole TFD-28) may have been mislabelled or contaminated during the sampling or preparation procedures.

Table 13.7
Summary of Blank Assay Data for the 2006 San Francisco Project Drilling

Samples	224
Mean Grade (g/t gold)	0.008
Maximum Grade (g/t gold)	0.272
Minimum Grade (g/t gold)	<0.002

Table provided by Timmins

13.2.4 DUPLICATES

Duplicate samples were taken during the drill program at the rate of one duplicate for every 25 samples. For the reverse circulation drilling the duplicate samples were taken after the initial drill sample (41 to 45 kg) had been split in the field to obtain a 21- to 23- kg sample. In the case of a dry sample it was split in the field again to form a duplicate weighing approximately 10.5 to 11.5 kg. In the case of a wet sample, it was dried first. In the case of the diamond drilling the sawn half core samples were sawn a second time to produce quarter core samples which then acted as the duplicate samples.

The duplicate samples were assigned consecutive numbers in the sample numbering sequence so that the laboratory did not know it was receiving duplicate samples. These samples were submitted in the same shipment as their matching original samples but were not necessarily placed in the same furnace load as the matching original sample.

The total number of duplicates assayed for gold was 238 pairs of samples of which 116 pairs were generated from the reverse circulation drilling and 122 pairs were generated from the diamond drilling portions of the drilling program. All of the samples were assayed for gold by fire assay.

The results of the 238 duplicate samples submitted by Timmins were compiled and plotted on scatter diagrams. Table 13.8 summarizes the results of the comparison between the duplicate sample assays for the reverse circulation and diamond drilling, as well as for the entire drilling program. Results for individual duplicate samples are given in Appendix IV.

Table 13.8
Summary of the Results for the Duplicate Samples, 2006 Drilling Program

	Reverse Circulation Drilling		Diamond Drilling		Entire Drilling Program	
	Gold (g/t)		Gold (g/t)		Gold (g/t)	
	Original	Duplicate	Original	Duplicate	Original	Duplicate
Number of Pairs	116	116	122	122	238	238
Average Grade	0.114	0.123	0.196	0.160	0.156	0.142
Maximum	2.590	3.870	12.250	9.230	12.250	9.230
Minimum	0.002	0.002	0.001	0.001	0.001	0.001
Absolute Difference between Avg Grades.		0.009		-0.036		-0.014
Difference%		7.476%		-18.54%		-9.06%
Correlation Coefficient		0.9383		0.9773		0.9584

Table provided by Timmins

Figure 13.12 shows the best fit regression line drawn through all of the duplicate gold assays, while Figures 13.13 and 13.14 show separately those for the diamond drilling and reverse circulation drilling, respectively.

Figure 13.12
Scatter Plot for Gold for All Duplicate Assays

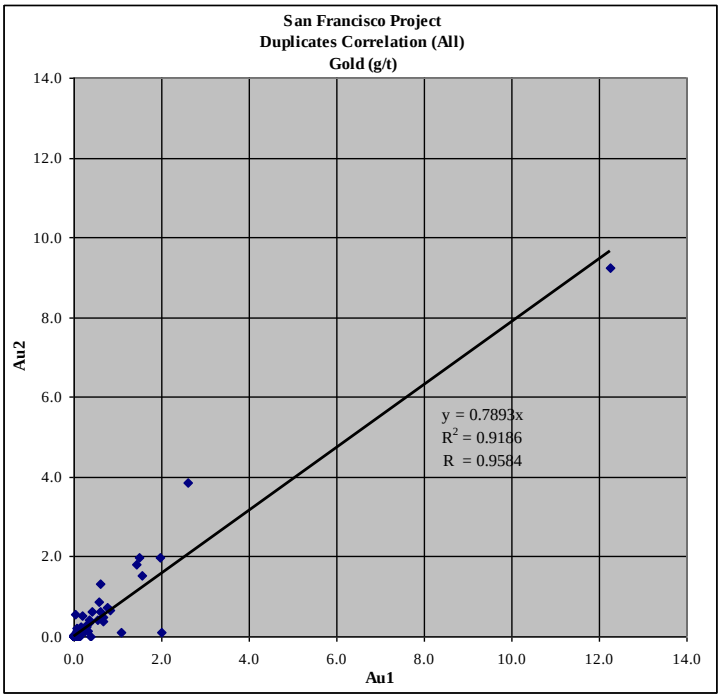


Figure provided by Timmins

Figure 13.13
Scatter Plot for Gold for just the Reverse Circulation Drilling Duplicate Assays

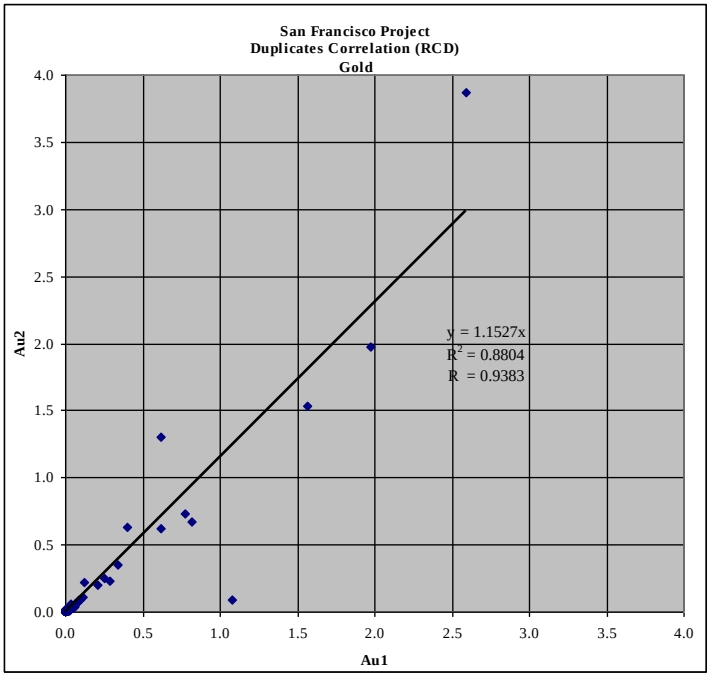


Figure provided by Timmins

Figure 13.14
Scatter Plot for Gold for just the Diamond Drilling Duplicate Assays

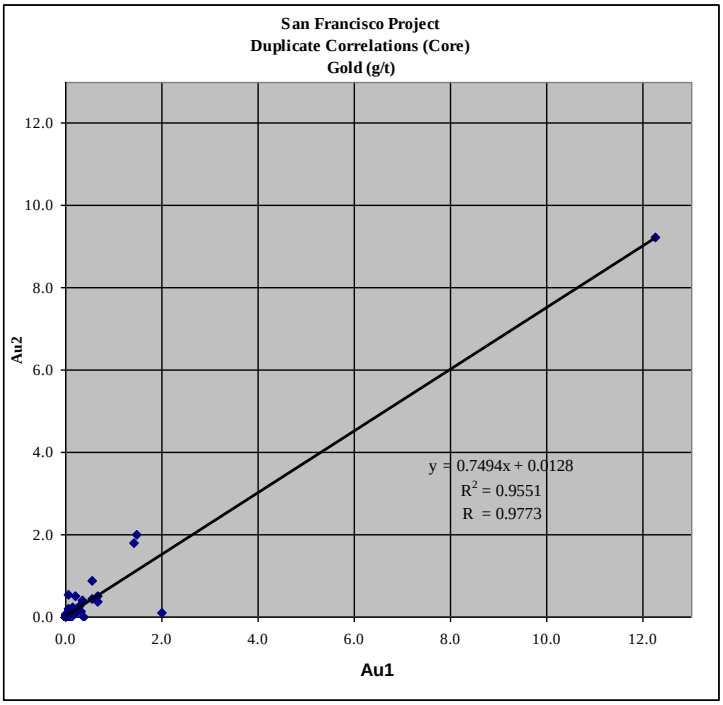


Figure provided by Timmins

When the entire 2006 program of diamond and reverse circulation drilling is considered, the duplicate gold assays show a correlation coefficient of 0.958 and a difference of -9.06% between the average grades of the 238 gold assays (Figure 13.12). The gold samples show an indication of bias. However, it is important to note that there are a large number of low grade samples and a greater variability between the original and the duplicate assays for some of these samples, especially for the very low values and nominally for some of the higher assays as well. This bias may be as a result of sampling and this should be investigated as well as the possibility that the bias may result from coarse gold within the sample which is producing a nugget effect in the sampling. The bias may also be an artefact of combining the results from two different drilling types into one graph as there are different sampling techniques and procedures used during reverse circulation and diamond drilling.

The results of the duplicate reverse circulation assays show a number of higher grade outliers in the data set which results in a correlation coefficient of 0.938 (Figure 13.13) and a difference of 7.48% between the average grades. Thus the duplicate reverse circulation samples show an indication of bias. However, a number of the gold samples were below the detection limits of the assaying methods used and these samples were given a constant value, which in itself contributes to the overall bias and, in this case, may contribute a larger than normal bias. Also, some of the individual pairs show a greater variability between the original sample and the duplicate especially in the very low grade assays, as well as for some of the higher assays. The bias may be the as a result of sampling but it is far more likely to result from the different lower detection limits used for assaying due to the large number of low grade assays near the detection limits of the assaying methods. A review of the available assaying methods should be conducted and the method with the lowest detection limit chosen for each laboratory that the duplicates are sent to. However, coarse gold within the sample may be producing a nugget effect and samples which exhibit a greater than 10% spread should be reassayed using the screen metallics method.

The duplicate gold assays for the diamond drilling exhibit a correlation coefficient of 0.977 and a difference of -18.54% between the average grades (Figure 13.14). While the scatter plot appears to be less biased and the correlation coefficient is higher for the diamond drilling assays, there is a much higher variation between the average grades. Part of the greater difference between the average grades may lie in the sampling method of quartering the core for the duplicate samples. This may result in coarse gold within the sample that is producing a nugget effect which is then aggravated by the fact that the sample is only half of a normal sample.

The correlation coefficients (0.938 to 0.977) and the percentage differences (-18.54% to 7.48%) for each of the drilling types plus the combined drilling program indicate that there appear to be some possible biases in the sampling preparation or internal laboratory procedures. However, before making the assumption that it is a preparation or laboratory problem all other possibilities need to be ruled out, especially in light of the assaying for all three of the standard reference samples which produced a combined correlation coefficient of 0.963. First, a representative number of the reverse circulation and core duplicate assays

should be reassayed at a third laboratory as a check of the procedures. In the case of the core duplicates, IMC has recommended that 15 or so core duplicates should be re-assayed where either the original or duplicate assay exceeds 0.1 g/t. The procedure would be to combine the coarse rejects from the original and duplicate sample, homogenize the material and split into halves and send one of the halves to the laboratory as an "original" and the other half as the "duplicate". IMC suspects the procedure actually used for core duplicates, which consisted of using only a quarter core for the original and duplicate sample, introduced more sampling error into the duplicate program than actually exists in the non-duplicated core samples. Micon also recommends that in future exploration drilling, all the gold assays which show a range of assays greater than 10% be assayed by the screen metallics procedure to examine the samples for a nugget effect. In the case of duplicate samples, Timmins' database should use the average of the two duplicate samples instead of the original assay so as not to introduce any undue biases into future resource estimates.

It is Micon's opinion based on a general and specific assessment of the drill sample data that the quality for the drilling samples meets accepted industry standards. In general, the sampling is believed to be representative of the areas examined. However, in the case of diamond drilling a re-evaluation of the use of quarter core for the duplicate samples should be undertaken and IMC's procedure adopted. Also a re-examination of the lower detection limits of the assaying procedures used should be undertaken and procedures checked to ensure that the methods used at the various laboratories have the same lower detection limits.

14.0 DATA VERIFICATION

Micon's Mr. Lewis visited the San Francisco project on September 10, 2006. Micon previously conducted two site visits in 2005 during which both the San Francisco and La Chicharra open pits were inspected and it was noted that the geology matched the previous descriptions. During the 2005 site visits, several of the old rotary drill sites to the southeast of the San Francisco pit were identified and a number of grab and drill samples were taken to independently verify the mineralization on the property. The results of Micon's sampling were discussed in the previous NI 43-101 Technical Report placed on SEDAR by TMM on April 28, 2006.

During the September, 2006 site visit Micon discussed the drilling program objectives and layout with Miguel Angel Soto y Bedolla and Daniel Maya. Micon observed the reverse circulation drilling near the La Chicharra pit and the core logging and sampling procedures for both the diamond and reverse circulation drilling. The reverse circulation drilling procedures observed were standard for the industry, where the drill crew chief determined the drill advance metreage, under the oversight of the drill geologist. Sample collection was conducted by a two-man crew under the direction of the drill geologist. No abnormal drilling conditions were encountered during the time Micon was on site. The logging and sampling procedures for both the diamond and reverse circulation drilling as observed by Micon, were consistent with the normal logging practices in the industry at this time and are described in Section 12 of this report.

Micon's review of the QA/QC procedures undertaken by TMM/Timmins at the San Francisco project has found that the exploration procedures and QA/QC procedures are consistent with the CIM Best Practices Guidelines of August 20, 2000.

IMC's data verification work consisted of the following. IMC conducted mineral resource and mineral reserve studies for Geomaque during 1997, while the property was in commercial operation. As a result of these studies, IMC had an archive copy of the drill hole database as of mid-1997. IMC compared the drilling database provided by Timmins to this historic database. The comparison verified that there has not been any tampering with, or inadvertent errors introduced into, the data.

IMC plotted cross-sections of the drilling data and compared the Geomaque data from prior to 1997 with the newer data. The trends of mineralization in the newer data correspond well with what would be expected from the older data. IMC also proposed that the general integrity of the database is verified by historical production from 1996 to 2000. As Section 17.2.3 shows, it is relatively easy to take the existing data and develop a resource model that is a close analog of actual mining results.

IMC conducted the most recent site visit to the property on February 14, 2007. The main purpose of the visit was to examine the condition of the San Francisco and La Chicharra pits, and also the condition of the San Francisco waste dumps since they will be expanded if the operation is re-started.

A multi-disciplinary team from Micon comprising Mr. R.J. Leader, P.Eng., Senior Mining Engineer, Mr. Victor Bryant, IEng, AIMMM, Vice President and Senior Metallurgist, Ms. Jenifer Hill, R.P.Bio., Environmental Scientist, and Mr. C.A. Jacobs, CEng MIMMM, Senior Consultant mineral economist and project manager, visited the project site during the period September 25-28, 2007.

Mr. Leader has reviewed the mine planning and production scheduling carried out by Timmins, and also the terms of the proposed mining contract and related cost estimates.

Ms. Hill reviewed the social and environmental studies prepared by others, and compiled the material presented in Section 18.4 of this report.

Mr. Jacobs reviewed the capital and operating cost estimates for the project and compiled the discounted cash flow analysis and sensitivity studies used to evaluate project economics. Mr. Jacobs also supervised the work of Ms. Hill, and has taken responsibility for that work.

Mr. Ian R. Ward, P.Eng., President of Micon, has reviewed and taken responsibility for the work of Mr. Bryant, who undertook the site visit and has compiled descriptions of the metallurgical testwork and the crushing, heap leaching and adsorption/desorption/recovery (ADR) processes and plant.

15.0 ADJACENT PROPERTIES

The San Francisco property exists within the Sierra Madre Occidental metallogenic province and is known to host a number of separate zones of anomalous gold mineralization. There are other metallic mineral deposits in the area, but very little information is available on those properties. There are no immediately adjacent properties which directly affect the interpretation and evaluation of the mineralization or anomalies found at San Francisco. However, the 1995 San Francisco Property Reserve and Resource document by Mine Development Associates of Reno, Nevada, listed a number of exploration possibilities in the immediate area of the mine, including La Chicharra which was mined during the last two years of production.

The targets which remain are the bedrock area surrounding the Arroyo La Perra, a placer deposit located approximately 2 km northwest of the San Francisco pit. The 1995 report mentions that seven drill holes had been drilled in bedrock to that point and one of the holes intercepted 8 m of 1.6 g/t gold at 42.5 m down hole, while another intercepted 18 m of 0.422 g/t gold at 4 m down hole. According to the report, other targets with fair to good exploration potential for the discovery of significant gold deposits were La Desconocida, Casa de Piedras Oeste, and La Trinchera, all of which are located between 2 km to 5 km northwest of the San Francisco pit.

Micon considers that the previous mining history of the San Francisco and La Chicharra deposits and the stated exploration potential of the area as contained in previous reports, positively affect the prospectivity of the ground.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 INTRODUCTION

The San Francisco mine operated during the period 1995-2000, when approximately 13.5 million tonnes of ore at a grade of 1.13 g/t gold was treated by heap leaching, and 300,834 oz of gold were recovered. Metal recovery from the pregnant solution was effected through carbon columns. Loaded carbon from the columns was transferred to a Zadra elution plant for precious metal extraction and the production of Doré bullion.

Average gold recovery over the mine life was about 63%. Mining operations ceased in 2001 as a result of low gold prices, although leaching and rinsing of the heap continued for approximately one year after this.

Toward the end of the mining operations, testwork was initiated to determine the cause of lower recoveries which were being experienced at that time.

In 1997, METCON Research Inc. (METCON) of Tucson, Arizona was commissioned by Geomaque de Mexico, to determine optimum crush size and other parameters for leaching. The work was performed on two ore types, schist and granitic. For each ore type, samples were crushed to a P_{80} of 1.5, 0.75-, 0.5-, and 0.375-in, respectively. Recoveries after 74 day leach cycles varied from 53% for the coarsest material to 72% for the finer crushed sizes. Interestingly, though, the finest crushed size did not give the best results. The schist samples produced marginally higher recoveries than the granitic ore.

A subsequent project review was carried out in October 1998 by The Winters Company (TWC) on behalf of the Republic National Bank of New York. The technical review included information from the 1997 METCON testwork and previous site visits by TWC. TWC examined three scenarios using updated costs, varying mine cut-off grades and metal prices and projected realistic recoveries, which were higher than those actually being achieved in the plant. TWC indicated that in its opinion the maximum likely heap recovery possible, would be 68% and in what they called their adverse case, used a 62% recovery. It was TWC's opinion that the lower plant recoveries experienced in the latter years of the operation were a result of :

(i) low solution coverage, evidenced, according to TWC, during the latter years of operation when an increase in coverage produced an obvious increase in recovery, and

(ii) coarsely crushed ore.

With regard to the latter, during its visit to the site Micon observed coarse particle sizes on the heap. This issue is further discussed in Section 18.2.2.

16.2 PREVIOUS TESTWORK

The following information is based upon a review carried out for Timmins by Sol & Adobe Ingenieros Asociados S.A. de C.V. (Sol & Adobe). The report, entitled "San Francisco Scoping Study" while not NI 43-101 compliant, provides a compilation of work carried out on the project up to that time.

In Section 9 of its 2006 San Francisco Scoping Study, Sol & Adobe noted that:

- Norberg Inc. Minerals Research and Test Centre conducted tests to evaluate impact work indices and paddle abrasion index and that the results indicated that the material was characterized as very hard and abrasive. Sol & Adobe also noted that as a result of the tests the selection of crushing equipment would be affected.

- Metallurgical testwork was performed on the San Francisco project by four separate organizations, the first being Servicios Industriales Peñoles, S.A. de C.V. (Peñoles) of Monterrey, Mexico. Additional work was performed by McCelland Laboratories Inc. (McCelland), Kappes, Cassiday, both of Sparks, USA and METCON.

- The testwork performed by Peñoles and McCelland included both roll and column leach tests. Work completed at Kappes Cassiday included bottle roll tests on drill cutting samples and column tests on bulk material crushed to minus 5/8 in. Work by METCON included column tests on samples designated as schist and granite and representing four different crush sizes for each type of material. Tables 16.1, 16.2 and 16.3 contain summaries of the bottle roll tests and column leach tests by Peñoles, McCelland, Kappes Cassiday and METCON.

- Table 16.4 summarizes the Geomaque operational data from 1996 to 2000 which indicate that the average gold recovery was 61%, with the highest 66% and the lowest, 49%.

- A review of the available information indicated that, from 1998 onwards, the crush size changed significantly, as shown in Table 16.5 which indicates the crush size for 1997 and 1998. The importance of the effect that particle size plays in gold recovery has been recognized at the San Francisco project and the gold recovery "is expected to be in the order of 64% as long as the particle size is kept within the range recommended by the different laboratories, i.e. -3/4 inch to -1/2 inch." (Sol & Adobe, 2006). The 2006 Sol & Adobe report also mentions that "Column data indicate there is no correlation between ore type and recovery."

Table 16.1
McClelland and Peñoles Laboratories Summary of Bottle Roll Leach Test

Sample ID	Crush Size	Laboratory	Calculated Head (g/t gold)	Gold % Extraction	Hours Leach	Chemical Consumption (kg/t)	
						NaCN	$Ca(OH)_2$
BR-1	3/8 inch[1]	McClelland	0.79	34.8	96	0.15	2.3
BR-1	200#[2]	McClelland	0.55	87.5	96	0.08	2.6
BR-2	3/8 inch[1]	McClelland	0.86	72	96	0.05	2.3
BR-2	200#[2]	McClelland	0.96	89.3	96	0.08	2.6
BR-3	3/8 inch[1]	McClelland	1.06	25.8	96	0.06	2.3
BR-3	200#[2]	McClelland	0.51	80.0	96	0.05	2.6
BR-4	3/8 inch[1]	McClelland	0.41	50.0	96	0.08	2.3
BR-4	200#[2]	McClelland	0.69	70.0	96	0.08	2.6
BR-5	3/8 inch[1]	McClelland	0.72	52.4	96	0.08	2.3
BR-5	200#[2]	McClelland	0.65	84.2	96	0.08	2.6
CC-1	3/8 inch[1]	McClelland	3.05	61.8	96	0.12	1.5
CC-1	200#[2]	McClelland	3.43	92.0	96	0.16	1.8
CC-2	3/8 inch[1]	McClelland	2.26	60.6	96	0.11	1.6
CC-2	200#[2]	McClelland	2.50	90.4	96	0.05	2.0
CC-3	3/8 inch[1]	McClelland	2.57	58.7	96	0.05	1.5
CC-3	200#[2]	McClelland	2.54	91.9	96	0.21	1.8
CC-4	3/8 inch[1]	McClelland	2.98	50.6	96	0.08	4.4
CC-4	200#[2]	McClelland	2.30	88.1	96	0.08	4.4
UC-1	3/4 inch[1]	McClelland	1.03	63.3	96	0/08	1.7
UC-1	3/8 inch[1]	McClelland	1.51	65.9	96	0.08	1.5
UC-2	3/4 inch[1]	McClelland	1.95	52.6	96	0.08	1.5
UC-2	3/8 inch[1]	McClelland	2.91	75.3	96	0.08	1.8
UC-2	6#[2]	McClelland	3.70	63.9	96	0.08	1.9
UC-2	200#[2]	McClelland	4.53	93.9	96	0.05	2.0
UC-3	3/4 inch[1]	McClelland	1.92	53.6	96	0.08	1.4
UC-3	3/8 inch[1]	McClelland	1.47	65.1	96	0.07	1.5
BULK	15.45% 200#	Peñoles	1.58	79.1	72	0.06	1 $Ca(OH)_2$
BULK	43.13% 200#	Peñoles	1.98	86.8	72	0.07	1 $Ca(OH)_2$
BULK	67.95% 200#	Peñoles	1.95	94.4	72	0.05	1 $Ca(OH)_2$
BULK	89.25% 200#	Peñoles	1.96	94.4	72	0.07	1 $Ca(OH)_2$

[1] 90% minus
[2] 100% minus
Table from the Sol & Adobe 2006 report..

Table 16.2
Kappes, Cassiday Laboratories Summary of Bottle Roll Leach Test

Rock Type	Zone	Number of Tests	Average Calculated Head (g/t gold)	Average Gold % Extraction	Hours Leach	Chemical Consumption (kg/t)	
						NaCN	$Ca(OH)_2$
Granite-Gneiss	San Francisco	15	2.74	75.45	24	0.15	0.50
Diorite	San Francisco	1	3.34	45.51	24	0.11	0.70
Alluvium	San Francisco	1	1.34	94.78	24	0.11	0.80
Andesite-Gneiss	San Francisco	1	1.72	66.28	24	0.11	0.50
Gneiss	En Medio	15	1.75	68.76	24	0.24	0.73
Diorite-Gneiss	En Medio	3	2.79	81.19	24	0.24	0.50
Andesite-Gneiss	En Medio	1	2.55	76.86	24	0.49	0.50
Diorite	En Medio	1	0.29	44.83	24	<0.01	0.50
Gneiss	El Manto	18	2.25	62.52	24	0.25	0.76
Migmatite	El Manto	3	2.47	64.12	24	0.03	0.63
Diorite-Gneiss	El Manto	1	2.07	59.90	24	0.34	1.20

Table from the Sol & Adobe 2006 report

Table 16.3
San Francisco Project Summary of Column Leach Tests

Sample ID	Crush Size	Laboratory	Calculated Head (g/t gold)	Gold % Extraction	Hours Leach	Chemical Consumption (kg/t)		
						NaCN	$Ca(OH)_2$	Cement
CC-1	3/8 inch[1]	McClelland	3.50	64.7	68	0.37	-----	5
CC-2	3/8 inch[1]	McClelland	2.30	71.6	68	0.47	-----	5
CC-3	3/8 inch[1]	McClelland	2.71	58.2	68	0.34	-----	5
CC-4	3/8 inch[1]	McClelland	2.61	68.4	68	0.35	-----	5
UC-1	3/4 inch[1]	McClelland	1.30	71.1	68	0.52		2.5
UC-1	3/8 inch[1]	McClelland	1.44	73.8	68	0.68		2.5
UC-2	3/4 inch[1]	McClelland	1.85	64.8	68	0.54		2.5
UC-2	3/8 inch[1]	McClelland	2.33	82.4	68	0.75		2.5
UC-3	3/4 inch[1]	McClelland	2.02	67.8	68	0.70		2.5
UC-3	3/8 inch[12]	McClelland	1.95	77.2	68	0.55		2.5
UC-1	3/8 inch[1]	McClelland	1.44	75.5	68	0.61		2.5
UC-1	3/8 inch[1]	McClelland	1.75	74.1	68	0.62		2.5 CaO
BULK	2 inch[2]	Peñoles	1.71	54.5	53	0.33	4	-----
BULK	1 inch[2]	Peñoles	1.66	67.5	53	0.58	4	-----
BULK	1/2 inch[2]	Peñoles	1.71	78.7	53	0.70	4	-----
BULK	3/8 inch[2]	Peñoles	1.72	77.3	53	0.41	4	-----
BULK	1/4 inch[2]	Peñoles	1.62	74.0	53	0.68	4	-----
18428[3]	16 mm	Kappes Cassiday	1.10	73.63	60	0.86	0.68	-----
18429[4]	16 mm	Kappes Cassiday	1.85	75.67	60	0.87	0.68	-----
18430[5]	16 mm	Kappes Cassiday	0.91	60.44	60	1.22	0.72	-----
CL-07[6]	1 inch	METCON	1.275	62.31	54	0.0272	0.59	-----
CL-08[7]	1 inch	METCON	1.222	57.15	54	0.031	0.66	-----
CL-09[8]	1 inch	METCON	2.249	56.05	75	0.039	nil	-----
CL-19[9]	1 1/2 inch	METCON	0.684	56.03	74	0.105	0.92	-----
CL-20[9]	3/4 inch	METCON	0.632	68.34	74	0.096	0.92	-----
CL-21[9]	1/2 inch	METCON	0.673	70.29	74	0.085	0.93	-----
CL-22[9]	3/8 inch	METCON	0.777	61.28	74	0.087	0.93	-----
CL-23[10]	1 1/2 inch	METCON	1.593	62.53	70	0.074	0.79	-----
CL-24[10]	3/4 inch	METCON	1.430	60.11	70	0.060	0.74	-----
CL-25[10]	1/2 inch	METCON	1.549	61.33	70	0.055	0.74	-----
CL-26[10]	3/8 inch	METCON	1.268	60.48	70	0.061	0.80	-----

[1] 90% minus
[2] 100% minus
[3] El Manto zone, medium grade
[4] El Manto zone, high grade composite
[5] En Medio zone
[6] June-96, monthly composite
[7] July-96, monthly composite
[8] Tahoe Norte, monthly composite
[9] Schist
[10] Granite
Table from the Sol & Adobe 2006 report

Table 16.4
San Francisco Project Geomaque Annual Production 1996 to 2000

Year	Dry Crush on Pads (t)	Grade (g/t)	Ounces on Pad	Gold/Silver Ounces Doré	Gold Ounces Doré	Gold Recovered (%)
1996	1,735,550	1.32	73,655	46,787	36,127	49.0
1997	2,288,662	1.12	82,412	75,847	54,519	66.2
1998	3.074,902	1.05	103,803	86,940	58,808	56.7
1999	3,010,639	1.14	110,345	98,726	64,371	58.3
2000	3,380,431	1.09	118,465	104,953	69,100	58.3
2001					17,092	
2002					264	
Total	13,490,184	1.13	488,680		300,281	61.4

Note: 301,893 tonnes of mineral and 975,900 tonnes of waste rock were mined in 1995.
Table from the Sol & Adobe 2006 report.

Table 16.5
San Francisco Project Crush Size Change Operation Data 1997 and 1998

Size Distribution Average for Year	1997	1998
Percent passing 1 in	99.8	86.8
Percent passing 3/4 in	94.2	73.2
Percent passing 5/8 in	83.4	64.7

Table from the Sol & Adobe 2006 report

In the summary for the mineral processing and metallurgical testwork section of its report, Sol & Adobe noted the following: "Independently of ore type, schist or granite, gold extraction averaged 57% for a leach cycle of 72 days. Cyanide consumption ranged from 0.0554 to 0.085 kg/t the lowest to 0.105 kg/t, the highest. Lime consumption averaged from 0.77 to 0.93 kg/t." In plant, reported cyanide consumption averaged 0.11 kg/t, whereas lime gave 0.96 kg/t, values considered very close to the ones reported in several metallurgical tests."

16.3 RECENT TESTWORK

16.3.1 OFF-SITE LABORATORY TESTWORK

Four samples, representing the major ore types from the San Fransisco mine were shipped to PRA, in Richmond, B.C. Details of the PRA test procedure are contained in their report dated November 13th 2007, appended. The ore types represented, were identified as; Granite; Gneiss; Pegmatite and Gabbro. Each of the samples were treated as follows, the ''as received'', size of approximately minus ¾ in, was treated directly. Representative portions of each type were then crushed to achieve a P_{80} of ½ in and ¼ in respectively. Head grades varied from 0.19 to 1.57 g/t. When assayed for metallic gold it was found that all samples contained relatively high proportions although probably fairly fine as it did not appear to affect recoveries to a great extent. The samples also contained organic carbon in approximately 0.1 – 0.2%. Again this did not appear to impact cyanidation results.

Bottle roll testing was carried out on the "as received" nominally ¾ in and ½ in material, after 144 hours of leaching recoveries remained low and it was decided to crush to a P_{80} of ¼ in. This size was considered to be the finest that could practically be used for heap leaching and would require agglomeration with cement. As expected, the finer crush increased extractions and it was decided that the column testing would be carried out at this size.

Four columns were used, one for each ore type. The columns used for testing were filled to 2.44 m and had an internal diameter of 10.2 cm. Each column contained 25-28 kg of material. The material was agglomerated with 0.05 kg/t of hydrated lime, 5 kg/t of cement and approximately 5% moisture. Samples of pregnant solution were taken regularly and the gold extracted through a small carbon column, cyanide concentration and solution flows were maintained and adjusted daily, as required.

The leaching was stopped on the granite and gneiss columns after 75 days and gold extractions of 79.6% and 85.6% were achieved. Cyanide consumptions of 1.12 and 0.95 kg/t and 0.12 and 0.11 kg/t of lime, respectively. The gabbro and pegmatite ore leaching was continued to 97 days and extractions of 65.8% and 90.2% were achieved for each, respectively. Reagent consumptions were again low with the exception of the pegmatite test in which the lime doubled to 0.21 kg/t. It is understood that the pegmatite represents a relatively small fraction of the overall deposit and may not be mined for treatment.

Standard acid base accounting on four waste rock samples indicated that they could be classified as non-acid generating. This is consistent with the observation that the edges of the heap leach dumps exhibit no acid run-off and that the water in the open pit is alkaline.

Having established basic criteria and data at the PRA laboratory it was decided that full scale testing would be carried out at the Timmins gold facilities at the San Francisco minesite in Mexico.

16.3.2 TESTWORK CONDUCTED AT SITE

Prior to commencing the testwork at site, the initial PRA testwork was reviewed. It was agreed that, in addition to testing the exposed pit ore, samples would be taken from drill core corresponding to the previously unmined areas in the proposed pit expansion. Testing of material from these domains would establish whether there existed any metallurgical variability between the principal domains and also substantiate the PRA test results.

To determine locations for the samples, the pit was divided into the western and central sectors and further dividing the western portion into shallow (0-50m) and deeper (50-100m) sub-sectors. Initial testwork was based on bottle roll tests, using channel samples from within the existing open pit (for the central sector) and quarter-core from Timmins drill holes in the western sector. Following the bottle roll tests, a limited number of tests were carried out in 2.3-m and 6.0-m columns, the latter representing the actual lift height to be used for the operation.

16.3.2.1 Bottle Roll Tests

Bottle roll tests were conducted on all four major ore types, Granite/Gneiss; Gneiss; Pegmatite, and Gabbro, plus a sample of schistose material. The samples were crushed to 100% minus 3/8 in to ensure optimum leach exposure. Sample size was 1,000 g, and sufficient lime added to ensure that the leach pH was maintained above 10.5. Sodium cyanide concentrations were held in excess and were generally between 500 to 1,000 g/t equivalent.

The results, presented in Table 16.6, were generally lower than those obtained during the PRA testwork and also exhibited considerable variation. It was believed that the small sample size and free gold may have had an impact on the variability, so it was decided to proceed with tests using the 2.3 m columns.

Table 16.6
Bottle Roll Test Summary

Composite Sample No.	Mineral Type	Head Assay (g/t Au)	Calc'd Head (g/t Au)	% Recov. per Assays	% Recov Calc'd
5	Gra. Gneiss	0.50	0.56	72.10	64.32
10	Gra. Gneiss	0.03	0.02	68.00	100.00
11	Gra. Gneiss	0.85	0.59	27.35	39.57
12	Gra. Gneiss	1.58	1.16	14.53	19.79
15	Gra. Gneiss	2.68	1.29	28.2	40.65
18	Gra. Gneiss	2.40	1.66	26.35	38.04
19	Gra. Gneiss	2.17	1.46	25.98	38.52
21	Gra. Gneiss	11.53	19.05	104	63.26
22	Gra. Gneiss	1.10	0.95	38.88	44.66
3	Gneiss	12.94	11.03	18.58	21.79
4	Gneiss	1.37	0.62	11.20	24.62
7	Gneiss	5.39	3.60	39.66	58.61
8	Gneiss	7.27	5.30	23.16	46.78
16	Gneiss	1.67	1.73	27.99	26.9
17	Gneiss	3.13	2.08	21.74	32.7
20	Gneiss	2.90	2.08	13.32	18.5
17	Gneiss	3.13	2.08	21.74	32.7
1	Pegmatita	6.99	5.30	24.70	16.22
2	Pegmatita	5.74	4.30	30.85	41.80
6	Pegmatita	7.19	5.61	34.75	44.47
13	Pegmatita	12.08	9.25	29.41	27.57
14	Gabbro	0.84	1.51	28.11	20.49
9	Schist	4.98	1.95	9.79	24.90

16.3.2.2 Two-Metre Column Tests

A total of six tests were conducted, three on granitic material, and one each on partly altered granite, gabbro, and gneiss. All the tests were conducted on material crushed to 100% minus ½ in with the exception of test 146809-810 which was crushed to 100% minus ¼ in. Leach

times varied from 44 to 66 days as the tests were stopped once the extraction kinetics had slowed and the curves flattened.

The granitic material crushed to minus ¼ in produced a lower extraction rate (56%), compared to the two tests at 100% minus half inch, which averaged approximately 75%. The partly altered granite produced an extraction of 55%, the gabbro and the gneiss both approximately 47%. Although recoveries were improved over the bottle roll tests, they were still considerably less than those achieved during the PRA testwork. Figures 16.1 to 16.6 show the results of these column tests.

It should be noted that in all the leaching testwork, recoveries are calculated by two methods. The first, during the actual test, is based on solution assays and volumes; these were used to plot the recovery curves shown below. The second method, considered to be the more accurate, is calculated after the test, based on actual mineral head and residue assays and recovered gold.

Figure 16.1
Granite Crushed to 100% minus ¼ in (146809-10)

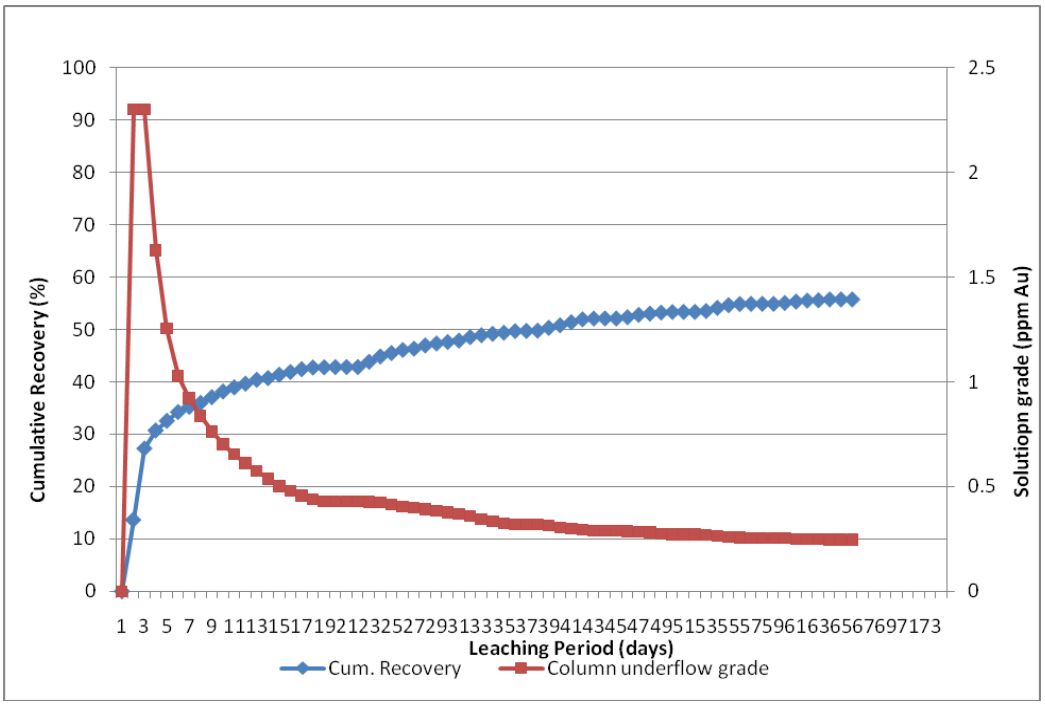


Figure 16.2
Granite (146826-33)

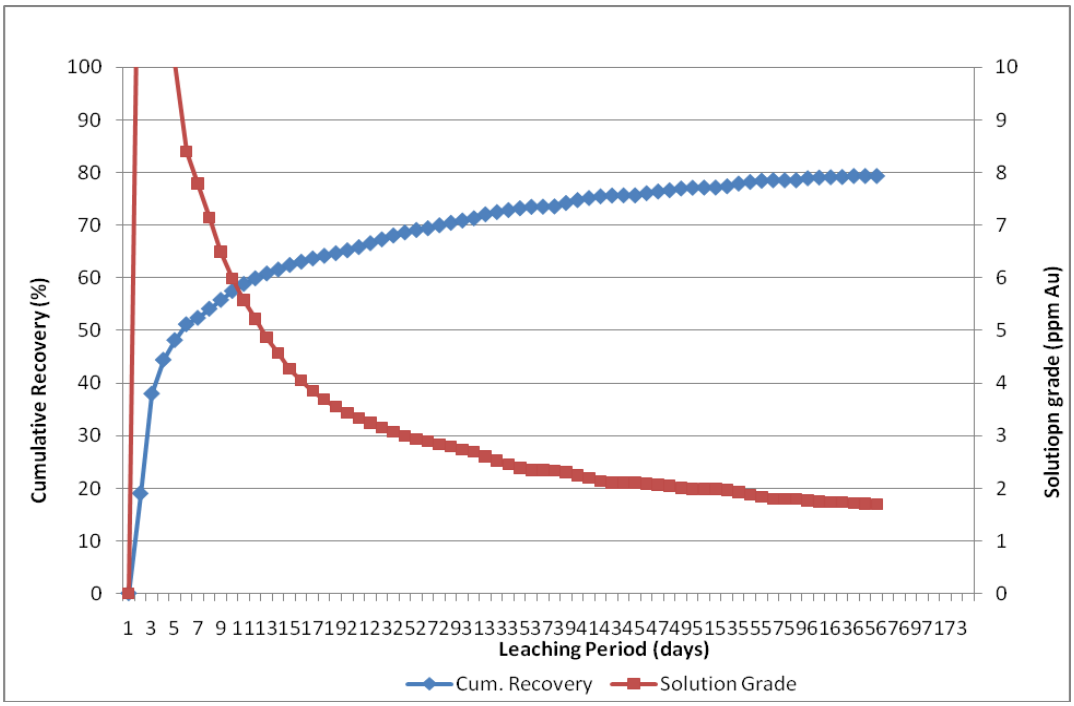


Figure 16.3
Granite (146 818-25)

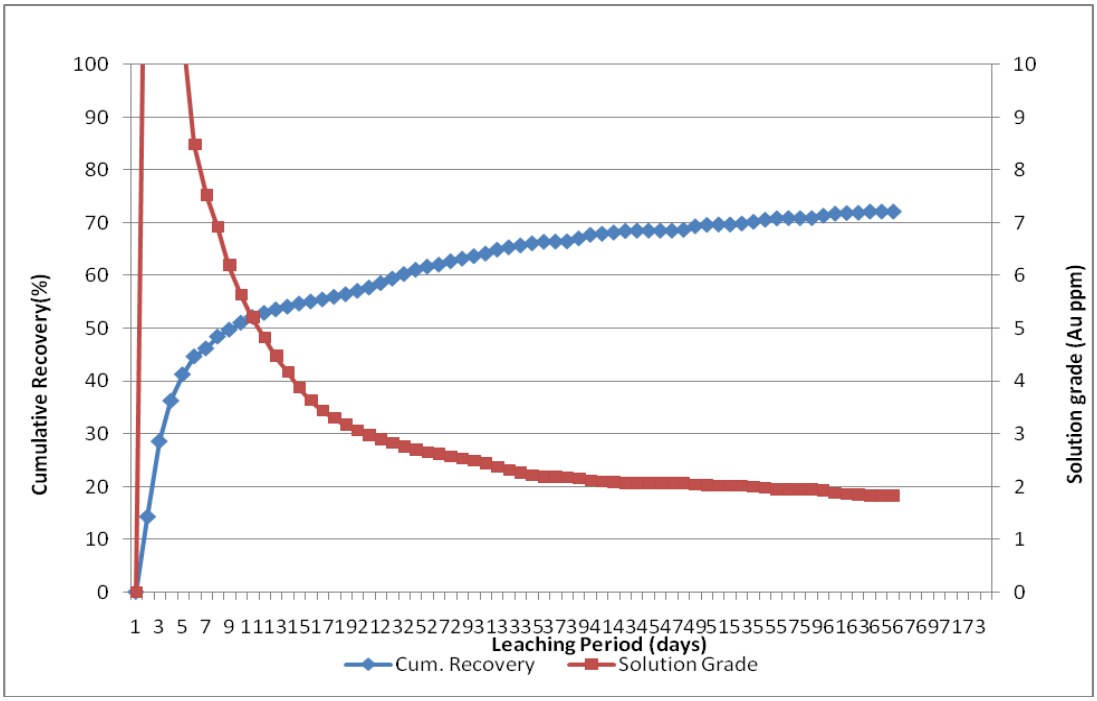


Figure 16.4
Partly Altered Granite/Gneiss

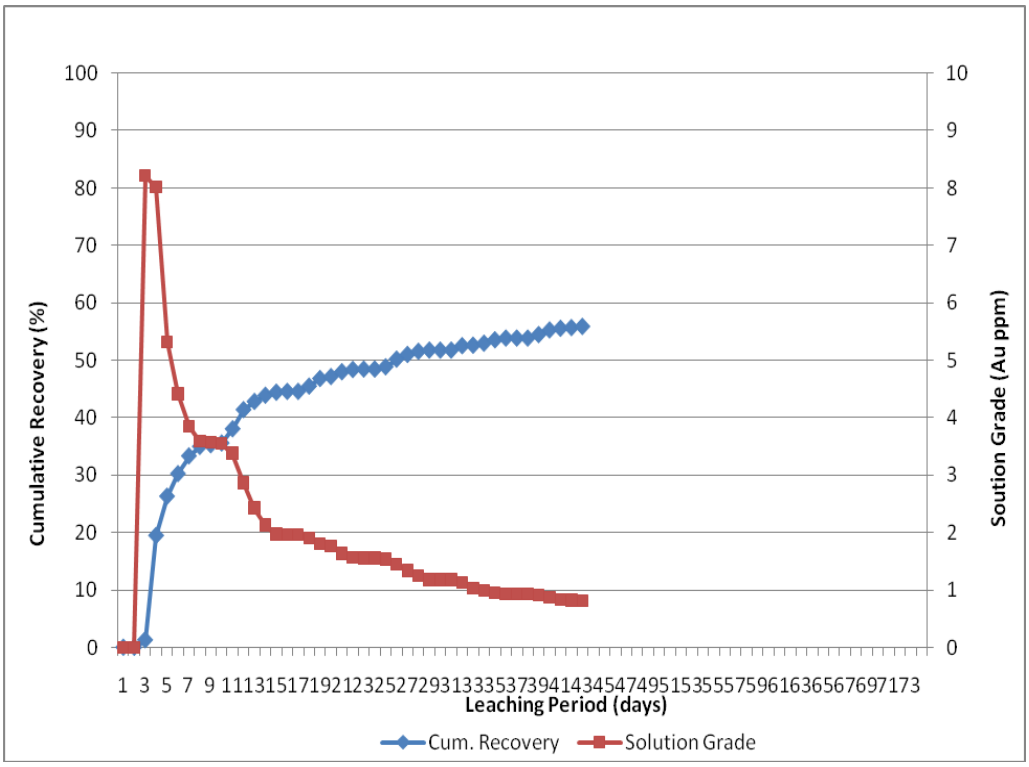


Figure 16.5
Gabbro

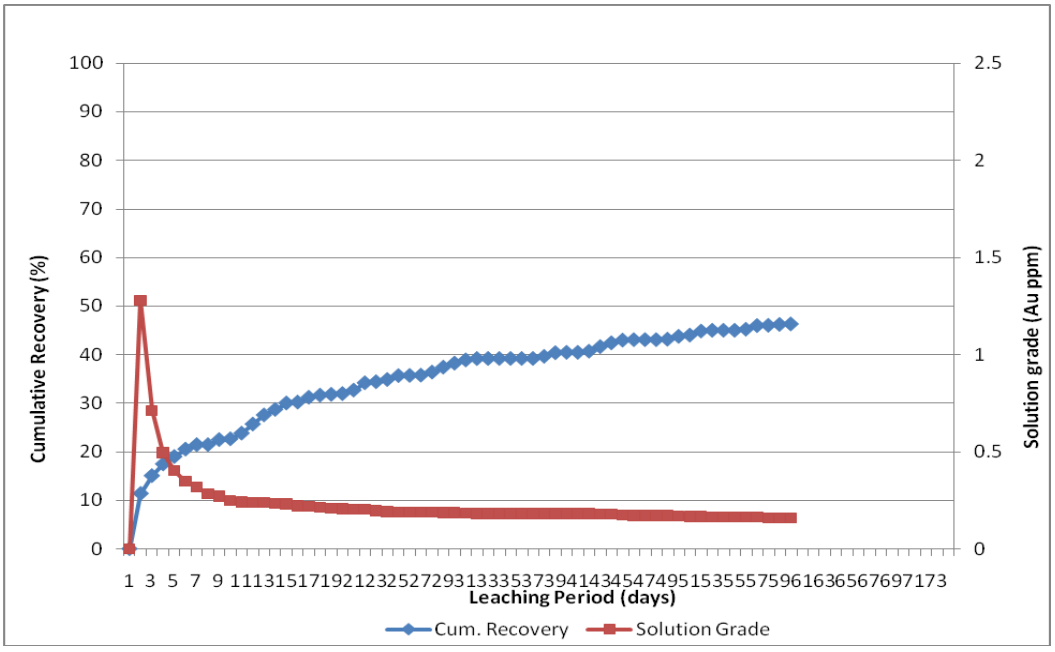
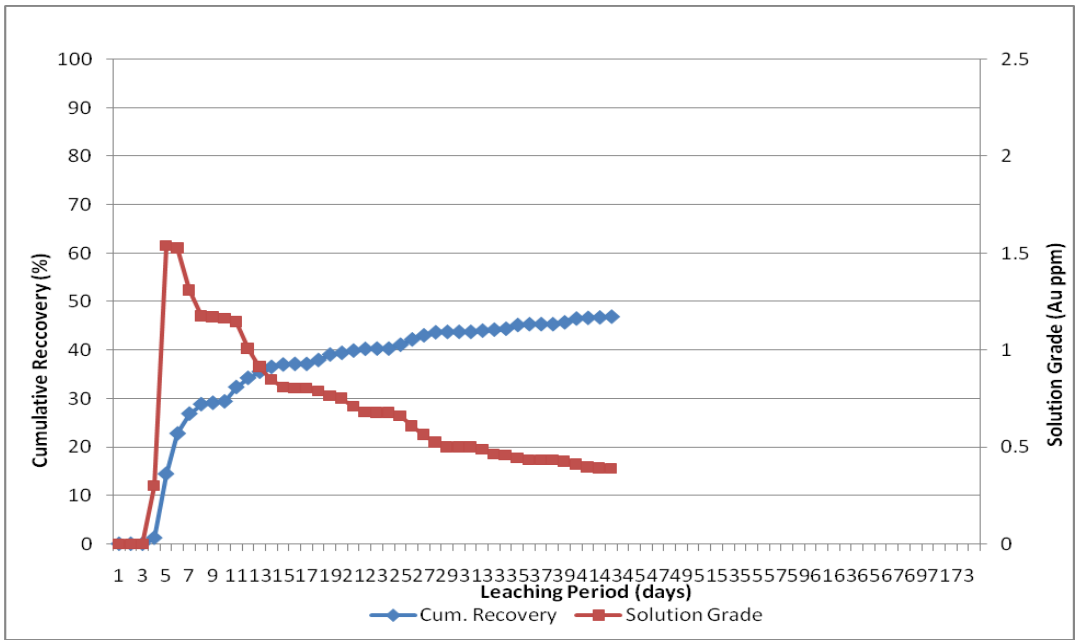


Figure 16.6
Gneiss



16.3.2.3 Six-Metre Column Tests

The columns used for the testwork were 140 mm ID by 6.0 m tall. Each column contained approximately 175 kg of mineral and all tests were conducted on material crushed to 100% minus ½ in. Leaching was conducted for a maximum of 47 days only, due to time constraints, but as can be seen from the leach kinetic curves, all samples were still leaching when the columns were shutdown for washing and residue assaying. It is intended that the normal leach cycle on the heap will be 90 days.

If the pregnant solution tenor dropped below 0.5 g/t then leaching would be stopped for 2 to 3 days to allow the solution grades to recover, i.e., "resting" the leaching process, which is standard practice for heap leach operations. After leaching was stopped, the columns were washed for 3 days, the residue removed and oven dried. Once dried, the material was thoroughly mixed coned and quartered and a portion screened and assayed by mesh size. Depending on the leach characteristics of the ore types, the procedure was varied slightly. This variation in leach characteristics, however, does not preclude co-leaching of these ore types on the heap.

The results from each ore type are shown in Tables 16.7 to 16.11 and corresponding Figures 16.8 to 16.12, respectively, showing recovery by mesh size and leaching rates. Percentage recoveries on the charts are measured against the back-calculated head grades. As might be expected given the truncated leach time, recoveries were lowest in the coarser size fractions.

Near-surface Granite

As found in the previous tests, this material gave the best leach response and overall extraction (see Table 16.7 and Figure 16.8). No rest periods were required during the 44 days of leaching. Gold recovery was 83% with a low sodium cyanide consumption of 0.075 kg/t, no additional lime was added for pH control so consumption remained as per the initial addition of 0.5 kg/t.

Gneiss

The column was leached continuously for 44 days and washed for 3 days. No rest period was required as the solution tenor remained above 0.24 g/t. Sodium cyanide consumption was low at 0.119 kg/t and no additional lime was required to maintain pH control. Recovery was lower at 62.2% which was experienced in previous tests and was also a result of the relatively low head grade. See Table 16.8 and Figure 16.9.

Granite Gneiss (Altered)

The column was leached in total for 40 days. During this period, two rest periods, each of 4 days were effected; after each, leaching was continued which resulted in an increase of 4% and approximately 2% in extraction, respectively. A recovery of 66.44% was calculated. It is significant to note that the solution tenors were still high when the test was terminated indicating the potential for a higher recovery which would be expected when compared with the granite results but dependent on the degree of alteration. Sodium cyanide consumption was low at 0.119 kg/t, no additional lime was required for pH control. See Table 16.9 and Figure 16.10.

Pegmatite

The material was leached for 17 days and then rested for 4 days. After the rest period, extraction improved by approximately 5%. A second rest period was carried out after a further 9 days and also resulted in a small extraction increase. Total leach time was 35 days and a recovery of 65.4% was achieved. Cyanide consumption was 0.074 kg/t, no additional lime was required. See Table 16.10 and Figure 16.11.

Gabbro

The gabbro was leached for a total of 32 days excluding 2 rest periods totalling 8 days. Extraction increases were noted after the rest periods but the overall recovery was 52.9% which is better than expectations for this material and it may, therefore, be considered economic to treat. Cyanide consumption was low at 0.110 kg/t. See Table 16.11 and Figure 16.12.

Table 16.7
Near-Surface Granite

Size (in)	Weight (kg)	Weight (%)	Au (ppm)	Contained Au (g)	Distribution (%)	
+3/8	2.3	23.15	3	6.9	16.44	
+1/4	2.49	25.07	2.3	5.73	13.65	
+10	3.27	32.92	2.6	8.50	20.26	
+60	1.25	12.58	9.63	12.04	28.69	
+100	0.18	1.81	10.07	1.81	4.32	
+150	0.074	0.74	8.33	0.62	1.47	
-150	0.37	3.72	17.2	6.36	15.17	
Total	**9.934**	**100**	**4.22**	**41.96**	**100.00**	
Size (in)	**Weight (kg)**	**Weight (%)**	**Au (ppm)**	**Contained Au (g)**	**Distribution (%)**	**Rec. (%)**
+3/8	3.78	25.19	1.10	4.16	29.65	63.33
+1/4	3.825	25.49	1.17	4.46	31.82	49.28
+10	4.565	30.42	0.73	3.35	23.87	71.79
+60	1.875	12.50	0.87	1.63	11.59	91.00
+100	0.295	1.97	0.40	0.12	0.84	96.03
+150	0.15	1.00	0.27	0.04	0.29	96.80
-150	0.515	3.43	0.53	0.27	1.96	96.90
Total	**15.005**	**100.00**	**0.93**	**14.03**	**100.00**	**77.96**

Figure 16.8
Near-Surface Granite

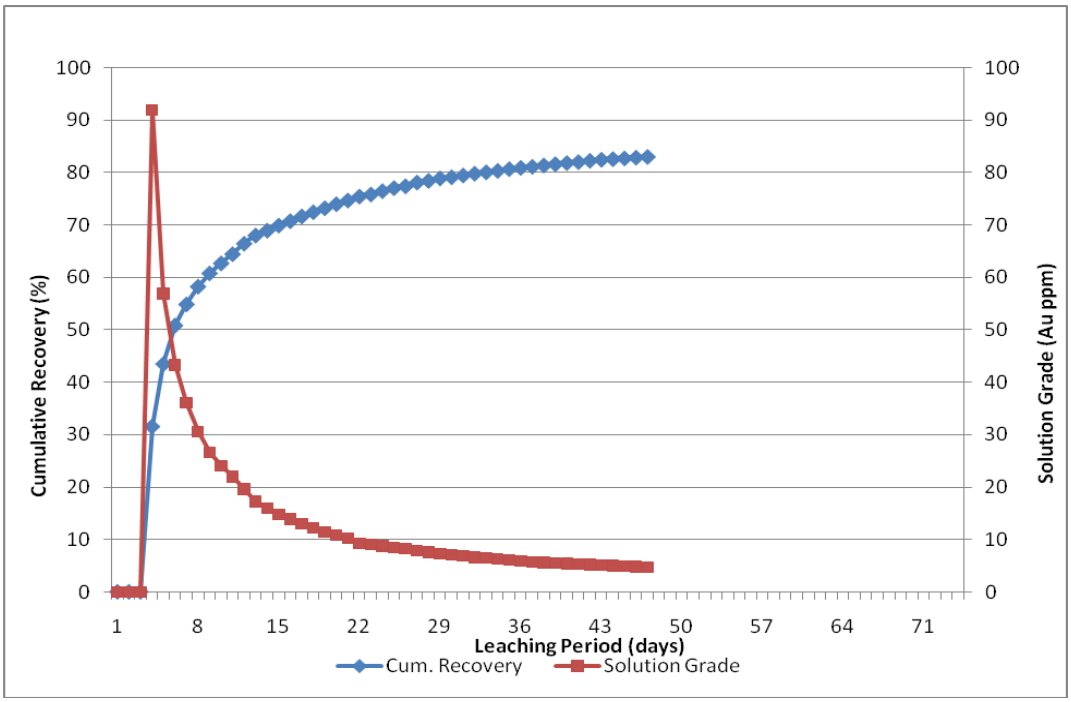


Table 16.8
Gneiss

Size (in)	Weight kg	Weight %	Au (ppm)	Contained Au (g)	Distribution %	
+3/8	2.19	22.28	0.6	1.314	8.38	
+1/4	2.57	26.14	0.93	2.3901	15.24	
+10	3.24	32.96	1.77	5.7348	36.56	
+60	1.26	12.82	3.3	4.158	26.51	
+100	0.19	1.93	2.33	0.4427	2.82	
+150	0.07	0.71	4.63	0.3241	2.07	
-150	0.31	3.15	4.27	1.3237	8.44	
Total	**9.83**	**100.00**	**1.60**	**15.6874**	**100.00**	
Size (in)	**Weight kg**	**Weight %**	**Au (ppm)**	**Contained Au (g)**	**Distribution %**	**Rec. %**
+3/8	4.425	29.58	0.53	2.36	25.38	63.33
+1/4	3.51	23.46	0.60	2.11	22.65	49.28
+10	4.27	28.54	0.90	3.84	41.33	71.79
+60	1.92	12.83	0.37	0.70	7.57	91.00
+100	0.29	1.94	0.27	0.08	0.83	96.03
+150	0.14	0.94	0.23	0.03	0.35	96.80
-150	0.405	2.71	0.43	0.18	1.89	96.90
Total	**14.96**	**100.00**	**0.62**	**9.30**	**100.00**	**62.20**

Figure 16.9
Gneiss

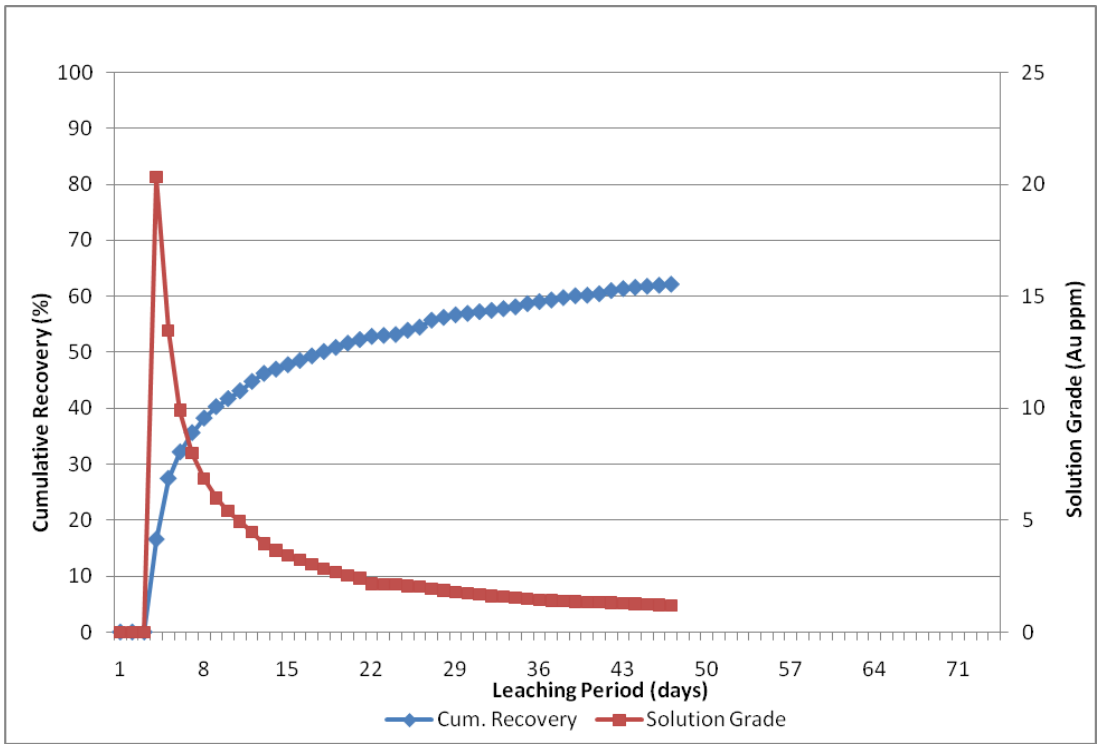


Table 16.9
Granite Gneiss (Altered)

Size (in)	Weight kg	Weight %	Au (ppm)	Contained Au (g)	Distribution %	
+3/8	1.65	16.57	0.4	0.66	15.88	
+1/4	2.7	27.11	0.35	0.945	22.73	
+10	3.5	35.14	0.35	1.225	29.47	
+60	1.43	14.36	0.37	0.5291	12.73	
+100	0.25	2.51	1.2	0.3	7.22	
+150	0.11	1.10	1.53	0.1683	4.05	
-150	0.32	3.21	1.03	0.3296	7.93	
TOTAL	**9.96**	**100.00**	**0.42**	**4.16**	**100.00**	
Size (in)	**Weight kg**	**Weight %**	**Au (ppm)**	**Contained Au (g)**	**Distribution %**	**Rec. %**
+3/8	2.74	20.35	0.23	0.6302	34.23	42.50
+1/4	3.58	26.59	0.17	0.6086	33.06	51.43
+10	4.01	29.78	0.07	0.2807	15.25	80.00
+60	2.31	17.16	0.1	0.231	12.55	72.97
+100	0.295	2.19	0.1	0.0295	1.60	91.67
+150	0.13	0.97	0.07	0.0091	0.49	95.42
-150	0.4	2.97	0.13	0.052	2.82	87.38
TOTAL	**13.465**	**100.00**	**0.14**	**1.84**	**100.00**	**66..64**

Figure 16.10
Granite Gneiss (Altered)

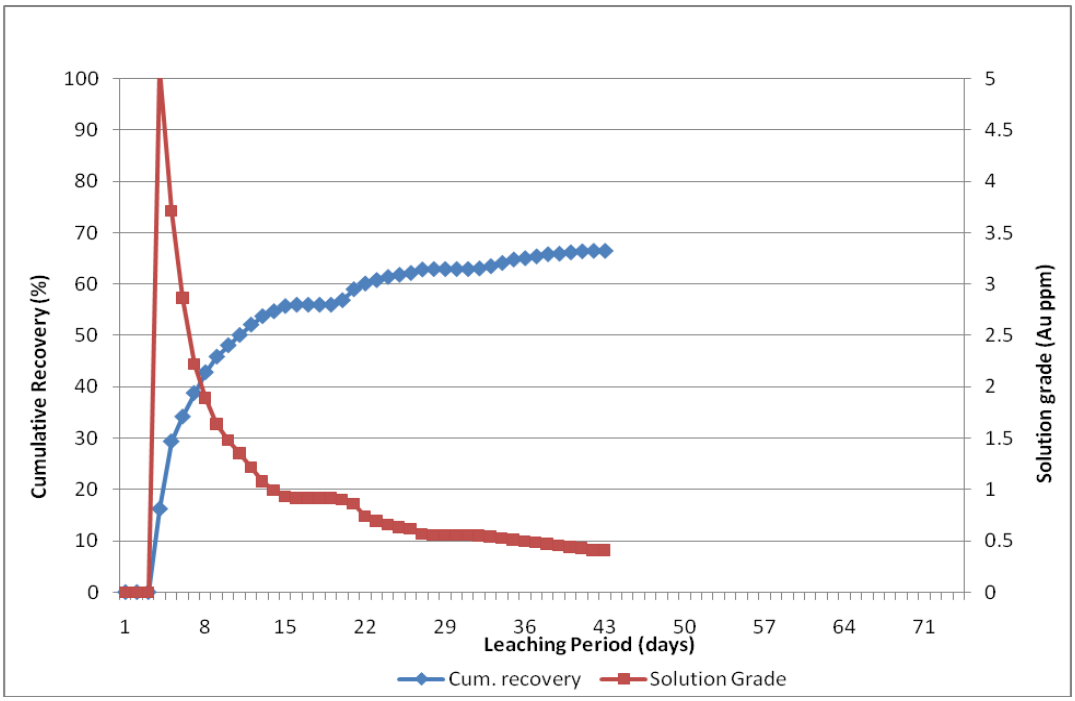


Table 16.10
Pegmatite

Size (in)	Weight kg	Weight %	Au (ppm)	Contained Au (g)	Distribution %	
+3/8	2.09	21.18	0.8	1.672	22.81	
+1/4	2.42	24.52	0.8	1.94	26.41	
+10	3.28	33.23	0.6	1.97	26.84	
+60	1.47	14.89	0.53	0.78	10.63	
+100	0.21	2.13	1.4	0.29	4.01	
+150	0.11	1.11	1.27	0.14	1.91	
-150	0.29	2.94	1.87	0.54	7.40	
TOTAL	**9.87**	**100.00**	**0.74**	**7.33**	**100.00**	
Size (in)	**Weight kg**	**Weight %**	**Au (ppm)**	**Contained Au (g)**	**Distribution %**	**Rec. %**
+3/8	4.285	28.63	0.30	1.29	33.21	62.50
+1/4	3.7	24.72	0.30	1.11	28.67	62.50
+10	4.475	29.90	0.27	1.21	31.21	55.00
+60	1.845	12.33	0.10	0.18	4.77	81.13
+100	0.255	1.70	0.13	0.03	0.86	90.71
+150	0.095	0.63	0.20	0.02	0.49	84.25
-150	0.31	2.07	0.10	0.03	0.80	94.65
TOTAL	**14.965**	**100.00**	**0.26**	**3.87**	**100.00**	**65.40**

Figure 16.11
Pegmatite

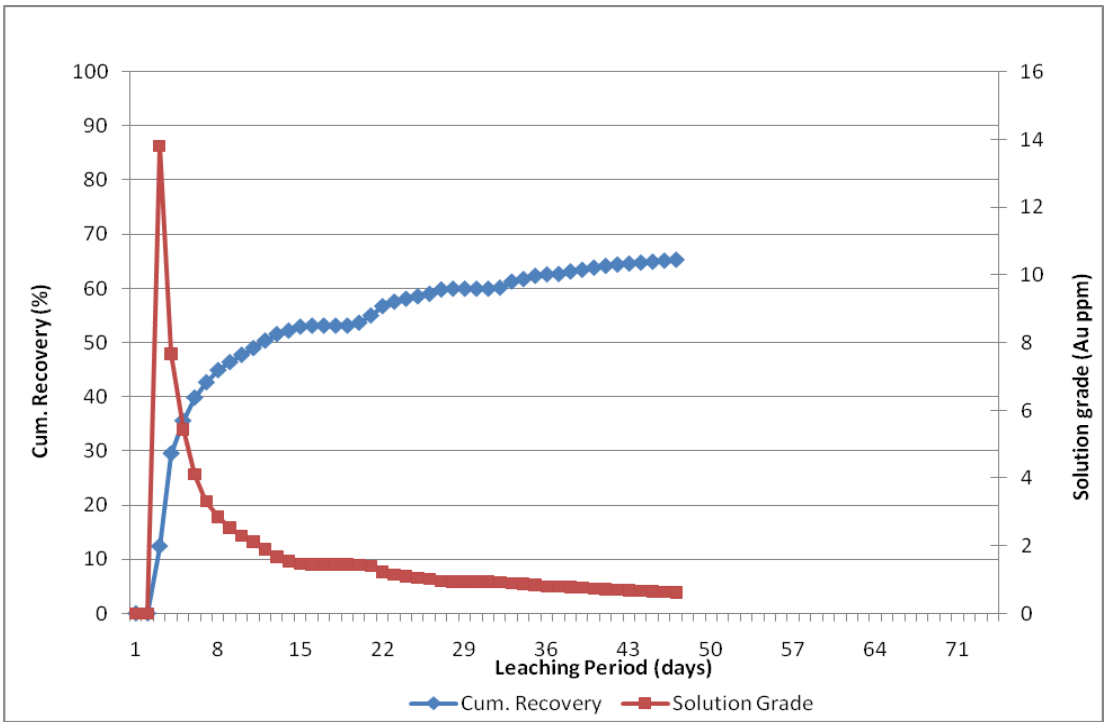
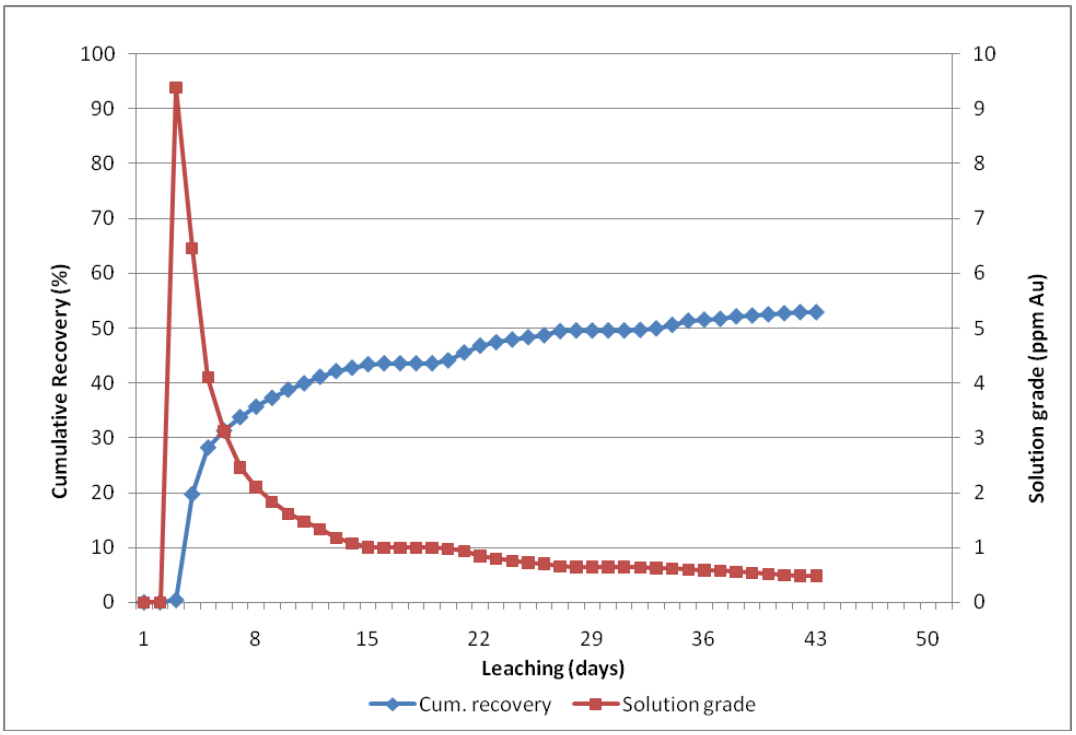


Table 16.11
Gabbro

Size (in)	Weight kg	Weight %	Au (ppm)	Contained Au (g)	Distribution %	
+3/8	2.75	27.76	0.47	1.2925	22.00	
+1/4	2.7	27.26	0.7	1.89	32.17	
+10	2.9	29.28	0.53	1.537	26.16	
+60	1.04	10.50	0.73	0.7592	12.92	
+100	0.14	1.41	0.8	0.112	1.91	
+150	0.046	0.46	0.23	0.01058	0.18	
-150	0.33	3.33	0.83	0.2739	4.66	
TOTAL	**9.906**	**100.00**	**0.59**	**5.8752**	**100.00**	
Size (in)	**Weight kg**	**Weight %**	**Au (ppm)**	**Contained Au (g)**	**Distribution %**	**Rec. %**
+3/8	4.525	30.36	0.27	1.22	29.17	42.55
+1/4	3.615	24.25	0.27	0.98	23.30	61.43
+10	3.73	25.03	0.3	1.12	26.72	43.40
+60	2.135	14.32	0.33	0.70	16.82	54.79
+100	0.26	1.74	0.27	0.07	1.68	66.25
+150	0.135	0.91	0.08	0.01	0.26	65.22
-150	0.505	3.39	0.17	0.09	2.05	79.52
TOTAL	**14.905**	**100.00**	**0.28**	**4.1882**	**100.00**	**52.90**

Figure 16.12
Gabbro



16.4 GEOLOGICAL DOMAINS

The difference in recovery observed between the various ore types made it necessary to estimate the proportion of each geological domain within the resource model. A sectional estimate was prepared by Timmins, which indicates the distribution of mineralised domains that will be encountered during mining of the pit shown in Table 16.12. From the testwork, observed and anticipated recoveries have been assigned for each domain, to arrive at an overall forecast of gold recovery from the mineral resource. It should be noted that the expected recoveries were obtained from material crushed to 100% minus ½ in.

Table 16.12
Actual and Forecast Leach Recovery

Domain name	Weight %	Actual Column Recovery after 47 Days (%)	Forecast Column Recovery after 90 Days (%)
Pegmatite	11.4	65.2	71.7
Gabbro	7.7	52.5	57.8
Gneiss	63.5	61.1	67.2
Granite	17.1	77.9	85.7
Granite/Gneiss (altered)	0.2	67.3	74.0
Overall weighted recovery		**63.7**	**70.1**

Whilst the same recoveries could possibly be obtained from coarser material, the leach time required would be longer, and would therefore impact on the planned production rate.

For financial modelling an overall recovery of 70% has been used for crushed material, which is consistent with the forecast 90-day recovery given above. For low-grade material heaped as uncrushed ROM ore, a recovery of 40% has been assumed over the same leaching period.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

As discussed in Section 6.2, the historical resource estimates were discussed in a previous NI 43-101 report, dated December 20, 2005 and entitled "Technical Report on the San Francisco Mine property, Estación Llano, Sonora, Mexico." The technical report was posted to the SEDAR website on April 28, 2006.

In 2006, IMC was engaged by Timmins to estimate the mineral resources for the San Francisco mine, using the historical Geomaque data along with the results of Timmins' 2005 and 2006 exploration drilling programs. IMC developed a three-dimensional (3D) block model and used floating cone techniques to develop a mineral resource within a constrained pit outline.

The resource estimate completed by IMC in January, 2007 is compliant with the current CIM standards and definitions specified by NI 43-101, and supersedes the historical resource estimate for the San Francisco mine.

Timmins also completed a drill program in 2007, the results of which have been publicly disseminated but which have not yet been incorporated into the present resource estimate. Micon understands a revised resource estimate is being compiled but, at the time of writing, this was not available for inclusion in the preliminary feasibility study.

17.1 MINERAL RESOURCE ESTIMATE

The mineral resource, as estimated by IMC, is presented on Table 17.1.

Table 17.1
IMC Mineral Resource Estimate for the San Francisco Project
(0.23 g/t Gold Cut-off Grade)

Category	Tonnes (000 t)	Grade (g/t Au)	Contained Gold (oz)
Measured Mineral Resource	5,352	0.912	156,930
Indicated Mineral Resource	22,296	0.781	559,860
Total Measured + Indicated Resources	**27,648**	**0.806**	**716,790**
Inferred Mineral Resource	2,506	0.788	63,490

Table provided by Independent Mining Consultants, Inc.

Both Canadian NI 43-101 and the Australasian Joint Ore Reserves Committee (JORC) code state that mineral resources must meet the condition of "a reasonable prospect for eventual economic extraction." For open pit material, IMC recognizes a floating cone geometry at reasonable long term prices, and reasonable costs and recovery assumptions, as meeting this condition for mineral resources. The resource presented in Table 17.1 is based on a floating cone pit shell at a gold price of USD 500 per ounce and additional cost and recovery

parameters developed by Timmins and IMC which meet the conditions for classification of the material as a mineral resource.

IMC does not know of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which would adversely affect the mineral resources estimated above. However, the reader should be cautioned that mineral resources that are not mineral reserves do not have demonstrated economic viability.

17.2 RESOURCE ESTIMATION PROCEDURES

This section provides a summary of the data, resource modelling methods, and other procedures used to develop this estimate.

17.2.1 DATABASE

The drilling database provided to IMC by Timmins consists of 1,133 drill holes amounting to 116,000 m of drilling. There are 62,137 sample intervals of which 61,346 were assayed for gold. The sampling interval is predominantly 2 m (86% of the intervals), though about 7% of the intervals are 1.5 m in length, and about 3% of the intervals are 1 m in length. In the case of the duplicate samples the original sample was used in the database.

17.2.2 RESOURCE MODELLING

IMC was also provided with a block model for the deposit. The exact source of this model is uncertain, although it could be the model IMC developed as part of its 1997 work for Geomaque. There has been a substantial amount of new drilling by both Geomaque and Timmins since the 1997 model was developed.

The provided model is based on 10 m by 10 m by 6 m high blocks and the coordinate limits and block size structure of the model were retained for this current work. The topography surface was updated to reflect the surface at the termination of Geomaque's mining activity in 2002. Pre-mining and current topographic surfaces are available in the model.

The 1997 block model was based on a significant amount of manual interpretation. Grade zones were interpreted on cross-sections and transferred to plan maps on a bench-by-bench basis. The plan maps were adjusted and digitized to incorporate the ore zones into the model. This method is quite time intensive, and difficult to maintain in situations where significant amounts of new drilling are being added.

For this study, IMC has implemented an indicator kriging estimation method to eliminate the manual interpretation and allow quicker incorporation of new data. Overall, the method is similar to the 1997 method, except that the grade zones are computer designed.

The estimation method is as follows:

1) The model is based on regular blocks of size 10 m by 10 m by 6 m high, as described above.

2) Block grade estimations are based on 3 m regular bench composites, even though the block height is 6 m. Assays were length weighted for each composite. The reason for the relatively short composite is to unsmooth the resultant block grade distribution to try to better match the likely distribution of mined blocks.

3) Assays were capped at 30 g/t prior to compositing. Assays were capped at 20 g/t for the 1997 work. Model validation work, to be presented later, indicates the 20 g/t cap grade understates the grade compared to historic production.

4) Ore zones were established in the model by indicator kriging, in which a discriminator of 0.125 g/t was used. Composites greater than 0.125 g/t were assigned a value of 1 and composites less than 0.125 g/t were assigned a value of 0. The ones and zeros were kriged to obtain a value between 0 and 1 for each block that may be interpreted as the probability that the block is above 0.125 g/t gold. Blocks with a probability over 0.5 were assigned a code to designate them inside the ore zone, i.e. an ore block.

5) Composites were assigned an ore/waste code as follows. Composites greater than 0.125 g/t gold were marked as ore. Composites below 0.125 g/t gold, but located within ore blocks defined as 0.75 or more probability of being ore were also assigned a code to mark them as in the ore zone. This accounts for internal waste composites.

6) Gold grades were estimated by ordinary kriging. Only ore blocks were estimated and only the ore zone composites were used for the estimation. The search radius for the estimation was 50 m along strike (N65°W), 60 m down dip (30°NE) and 20 m in the perpendicular (near vertical) direction. A maximum of 10 and a minimum of 1 composite were used to assign grade with a limit of 2 composites per hole. The indicator kriging to develop the ore zone was based on a maximum of 10 and a minimum of 2 composites, again with a maximum of 2 per hole. The relatively small maximum number of composites was, again, to limit over-smoothing the grade distribution. The search radii used represent 100% of the estimated variogram ranges of the gold variograms.

7) Based on historic data, an in-situ block density of 2.66 t/m^3 was assigned for the ore blocks and 2.77 t/m^3 for the waste blocks.

8) A resource classification code (measured, indicated, and inferred resources) was also assigned to the model. More details on this assignment are included below.

17.2.3 MODEL VALIDATION

Timmins personnel provided IMC with a table of historic production from the San Francisco pit between 1996 and the mine closure in 2002. IMC also has the topography surfaces to reflect the volume of material mined during the same period. Table 17.2 compares the historic production with the new IMC model for this material. The IMC model is tabulated at a cut-off grade of 0.4 g/t gold. This was the cut-off grade that was used for mining when IMC was last involved with the project in 1997.

Table 17.2
Comparison of the IMC Model with Historic Production for the San Francisco Project

Description	Tonnes (000 t)	Grade (g/t gold)	Gold (000 oz)
Historic Production	13,490	1.127	488.7
IMC Model at 0.4 g/t cut-off grade	13,707	1.113	490.5
% Difference	+1.6	-1.2	+0.037

Table provided by Independent Mining Consultants, Inc.

Table 17.1.2 represents a good comparison between the new model and reported historic production. This is also the basis for IMC increasing the historic cap grade from 20 g/t to 30 g/t. It appears that the 20 g/t cap grade would result in too low a gold head grade compared to the historic results. The 0.125 g/t gold grade IMC used for the ore/waste discriminator was also selected to get a good comparison of the model with historic results. A lower discriminator, such as 0.1 g/t gold, would result in slightly more ore at a slightly lower grade, resulting in a slightly less favourable comparison with actual results.

17.2.4 RESOURCE CLASSIFICATION

The resource classification is described as follows for the San Francisco deposit. A special kriging was carried out exclusively for the purpose of resource classification. The number of samples and kriging standard deviation from this exercise were used for the resource classification.

1) The maximum search radii for the special kriging were set to 38 m along strike, 45 m down dip, and 20 m perpendicular. This represents about 75% of the variogram range; IMC generally assumes that measured/indicated resources should be defined within 67% to 75% of the variogram range. The variogram was also normalized to a sill of 1 and a nugget of 10% of the sill. A maximum of one composite per drill hole was allowed in the kriging and the ore zones used in the grade kriging were respected. This kriging procedure provides a count of the number of holes within 75% of the maximum search radius and also calculates a kriging standard deviation based on these data. The number of holes and kriging standard deviation were stored in the model.

2) Probability plots of block kriging standard deviations by the number of holes were constructed. Figure 17.1 shows this plot.

Figure 17.1
Resource Classification Method

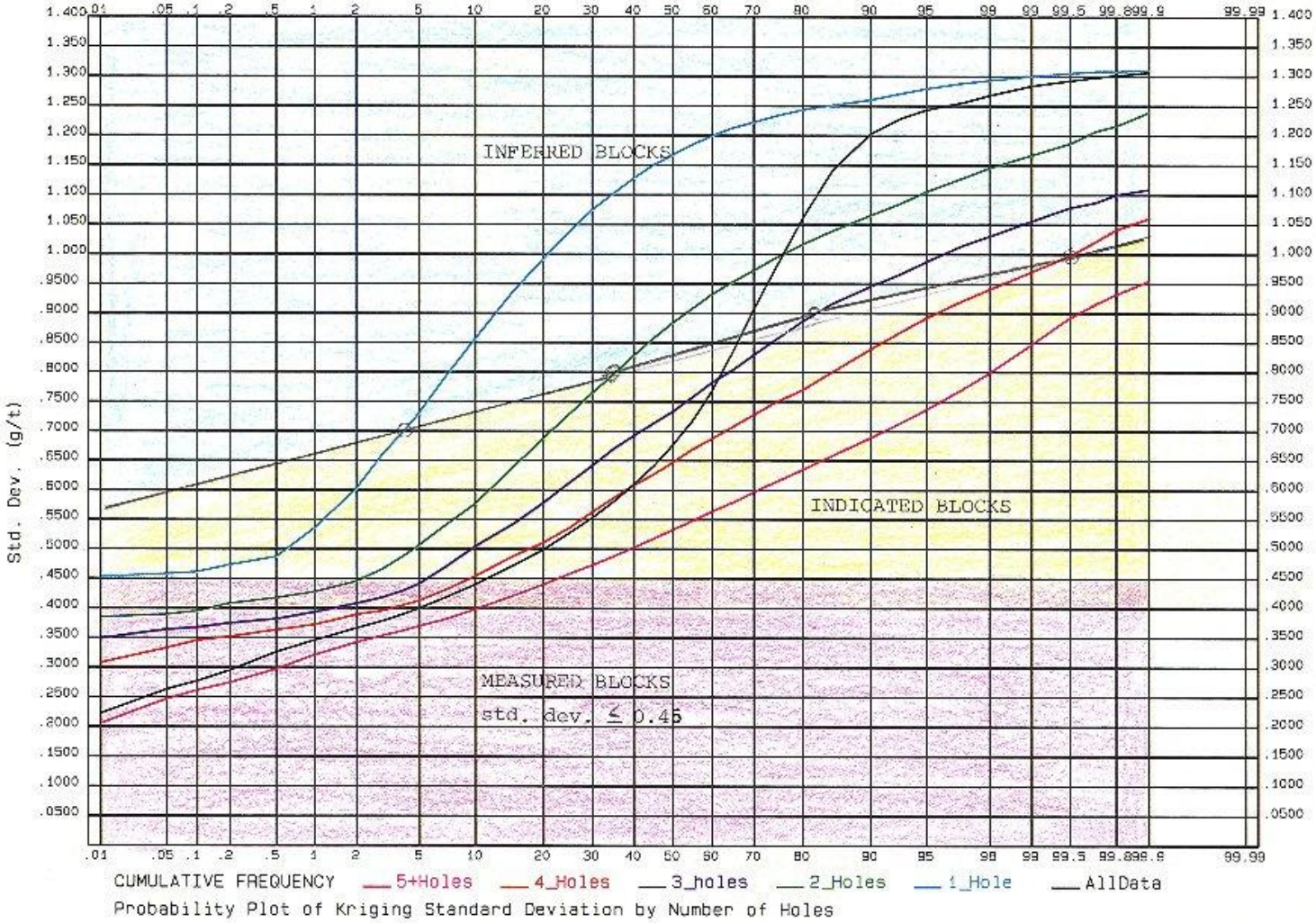


FIGURE 1: RESOURCE CLASSIFICATION METHOD

3) First, all blocks in the ore zone were set to a default of inferred resource. For blocks with the closest composite outside the 38 m by 45 m search, that is their final classification. They were not examined in the special kriging.

4) The plots of kriging standard deviations indicate that blocks estimated with five or more holes generally have a standard deviation less than 1.0. These were classified as indicated resource.

5) Blocks kriged with four holes and with a kriging standard deviation less than 1 were classified as indicated resource. This is about 99.5% of the blocks kriged with four holes. Blocks kriged with three holes and with a kriging standard deviation less then 0.9 were classified as indicated resource. This is about 82% of the blocks kriged with three holes. Blocks kriged with two holes and with a kriging standard deviation less than 0.8 were also classified as indicated resource. This is about 35% of those blocks. Blocks kriged with one hole and a kriging standard deviation less than 0.7 (about 4% of those blocks) were also classified as indicated resource.

6) All blocks with a kriging standard deviation less than 0.45 were then re-classified as measured resource. This is about 10% of all blocks kriged with the special kriging.

Visually, the described method appears to give good results. Indicated resources are not extrapolated far outside of the drilling data and measured resources are developed only in well-drilled areas. Blocks kriged with one or two holes can generate indicated resources only very close to the holes.

17.2.5 FLOATING CONE EVALUATION

17.2.5.1 Economic Parameters

To constrain the reported mineral resource within an economic pit outline, IMC performed a floating cone analysis based on the updated block model. Table 17.3 shows the economic parameters used for the analysis. The parameters are a combination of IMC and Timmins inputs.

Table 17.3
Floating Cone Parameters for the San Francisco Project

Parameters	Costs, Etc.
Mining Cost per Total Tonne	USD 1.00
Processing Cost Per Ore Tonne	USD 2.19
Crushing	USD 0.86
Leaching/Plant	USD 1.11
Maintenance	USD 0.12
Sales	USD 0.10
G&A Cost Per Ore Tonne	USD 0.20

Parameters	Costs, Etc.
Gold Recovery	64%
Gold Price Per Troy Ounce	USD 500
Cut-off Grades	
Internal Cut-off (g/t)	0.23
Breakeven Cut-off (g/t)	0.33
Slope angles based on Golder 1996 report	

Table provided by Independent Mining Consultants, Inc.

Two sets of optimizations were carried out: the first considered all resource material (including Inferred Resources) which was appropriate for the preparation of the resource statement; and the second considered only Measured and Indicated Resource material, which was required in order to generate a pit desing for the preliminary feasibility study and the conversion of those mineral resources to mineral reserves.

Note that the cost estimates included above were subsequently updated once more detailed engineering work had been completed and reviewed by Micon, for the purposes of preparing a cash flow forecast and economic evaluation of the project (see Section 17.3.1). Nevertheless, the optimized pit shells generated using only the Measured and Indicated resources are considered to have remained appropriate for use in the current preliminary feasibility study.

The USD 1.00 per total tonne owner mining cost was estimated by IMC. The processing and G&A costs were provided by Timmins. The G&A cost was based on a fixed cost of USD 600,000 per year and an ore production rate of 3 million tonnes per year.

Pit bench heights were set at 6.0 m (the block height of the 3D block model) and base-case slope angles used for the pit optimization were based on interramp angles recommended by Golder Associates in its December, 1996 report and adjusted by IMC to allow for haul roads of 20m width. Densities used to convert volumes to tonnage were 2.66 t/m^3 for ore and 2.77 t/m^3 for waste. The gold recovery of 64% was a provisional Timmins estimate.

These parameters resulted in a pit shell that contained the mineral resource estimate given above. IMC then analyzed the sensitivity of mineral resources to variations in gold price, process recovery, changes in wall slope angle and mine operating costs. Two sets of sensitivity analyses were carried out, the first being inclusive of measured, indicated and inferred resources, and the second including only measured and indicated The results of IMC's analysis can be found in Tables 17.2.3 and 17.2.4 and are described below.

17.2.5.2 Sensitivity Analysis Based on Measured, Indicated, and Inferred Resource

IMC ran a set of floating cones to test the sensitivity of the results to the cost and recovery parameters. Table 17.4 presents the results of this study. Note that in each case measured, indicated and inferred resources were allowed to contribute to cone economics. The base case (Case 1) is based on a gold price of USD 500 per ounce. This resulted in the resource tonnages shown in Table 17.1.1, above. Total in-pit material was 81.7 Mt. These results are tabulated at a 0.23 g/t cut-off grade, i.e. internal cut-off grade as calculated on Table 17.1.3.

Cases 2 and 3 show the sensitivity of results to gold price, and it can be seen that the results are quite sensitive to price. A 10% increase in price increased ore tonnes, contained ounces, and total material 12.9%, 9.4%, and 17.6% respectively compared to the base case. A 10% decrease in price reduced ore tonnes, contained ounces, and total tonnes 9.3%, 6.9%, and 11.6% respectively compared to the base case.

Cases 4 and 5 show sensitivity to process recovery, which, as expected, gives approximately the same results as the gold price analysis.

Cases 6 and 7 show sensitivity to the process and G&A costs. It can be seen that a 10% increase in costs results in a reduction of ore tonnes, contained ounces, and total tonnes by a modest 4.5%, 3.1%, and 5.2% respectively, while a 10% decrease in costs increased ore tonnes, contained ounces, and total tonnes 5.6%, 3.3% and 5.5%, respectively. This indicates only a moderate sensitivity of results to process costs.

Cases 8 and 9 show sensitivity to slope angles. Note that 10% steeper angles increased ore tonnes and contained ounces 3.9% and 3.6% respectively, while reducing total tonnes 0.4%. A 10% flattening of the angles resulted in 4.5% more ore, 2.7% more contained ounces, and a 9.2% increase in total material. This indicates resource results are only moderately sensitive to slope angles also.

Cases 10 and 11 show the impact of mining cost. A 10% reduction in mining cost results in a 6.9% increase in ore tonnes, a 4.9% increase in contained ounces, and a 9.3% increase in total tonnes. A 10% increase in mining cost results in 7.6% less ore tonnes, 6.0% less contained ounces, and 10.5% less total tonnes. Sensitivity to mining cost can also be described as moderate.

17.2.5.3 Sensitivity Analysis Based on Measured and Indicated Resource Only

Table 17.5 shows a similar sensitivity analysis, but with only measured and indicated resource allowed to contribute to cone economics. This scenario was selected as the basis of the preliminary feasibility study. Any inferred resource material contained within the pit is treated as waste rock. The base case cone contains 26.0 Mt of measured and indicated resource at 0.823 g/t gold for 689,000 contained ounces. Total material is 70.2 Mt.

Cases 2 and 3 show the sensitivity of this pit shell to gold price. A 10% price increase results in a 3.4% increase in ore tonnes, a 2.1% increase in contained ounces, and a 3.9% increase in total material compared to the base case. A 10% decrease in price results in a decrease of 8.3% in ore tonnes, a 6.5% decrease in contained ounces, and an 11.7% decrease in total tonnes compared to the base case.

As before, Cases 4 and 5 show that the sensitivity to recovery is the same as for gold price.

Cases 6 and 7 show a moderate sensitivity to process and G&A costs. A 10% increase in these costs reduces ore tonnes, contained ounces, and total tonnes by 4.8%, 3.7%, and 6.9% respectively. A 10% decrease in costs increases ore tonnes, contained ounces, and total tonnes 1.6%, 0.8%, and 1.4% respectively.

Cases 8 and 9 indicate that results are not very sensitive to slope angle when only measured and indicated resource is used. 10% steeper angles result in a 1.1% increase in ore tonnes, a 1.3% increase in contained ounces, and a 3.3% decrease in total tonnes. At slope angles 10% flatter, there is a decrease of 2.3% in ore tonnes, a 2.1% decrease in contained ounces, and a 0.2% decrease in total tonnes.

Cases 10 and 11 also show moderate to low sensitivity to mining costs. A 10% decrease in mining costs results in a 2.9% increase in ore tonnes, a 2.0% increase in contained ounces, and a 3.9% increase in total tonnes. A 10% increase in mining costs results in a 5.3% decrease in ore tonnes, a 4.3% decrease in contained ounces, and an 8.0% decrease in total tonnes.

It can be seen that for most parameters the sensitivity results are not very symmetric. There is more down side than up side in the results. The base case parameters have resulted in the extraction of a significant portion of the available measured and indicated resource, so there is little additional mineralized material in the block model available to increase the mineral resource.

Table 17.4
Sensitivity Analysis for Floating Cones Based on Measured, Indicated, and Inferred (MII) Mineral Resources

Case	Case	Resource Classes	Gold Cut-off Grade (g/t)	Tonnes (000)	Grade (g/t)	Gold (000 oz)	Waste (000 t)	Total (000 t)	Strip Ratio
1	Base Case – USD 500 Gold	MII	0.23	30,154	0.805	780.4	51,541	81,695	1.71
2	Gold Price – USD 550 Gold % Change from Base Case	MII	0.23	34,039 12.9	0.780 -3.1	853.6 9.4	62,044 20.4	96,083 17.6	1.82 6.6
3	Gold Price – USD 450 Gold % Change from Base Case	MII	0.23	27,358 -9.3	0.826 2.6	726.5 -6.9	44,891 -12.9	72,249 -11.6	1.64 -4.0
4	Recovery of 70.4% % Change from Base Case	MII	0.23	34,197 13.4	0.779 -3.2	856.5 9.7	62,736 21.7	96,933 18.7	1.83 7.3
5	Recovery of 57.6% % Change from Base Case	MII	0.23	27,358 -9.3	0.826 2.6	726.5 -6.9	44,891 -12.9	72,249 -11.6	1.64 -4.0
6	Process/G&A of USD 2.63 % Change from Base Case	MII	0.23	28,790 -4.5	0.817 1.5	756.2 -3.1	48,695 -5.5	77,485 -5.2	1.69 -1.0
7	Process/G&A of USD 2.15 % Change from Base Case	MII	0.23	31,835 5.6	0.788 -2.1	806.5 3.3	54,367 5.5	86,202 5.5	1.71 -0.1
8	Slope Angles 10% Steeper % Change from Base Case	MII	0.23	31,344 3.9	0.802 -0.4	808.2 3.6	50,040 -2.9	81,384 -0.4	1.60 -6.6
9	Slope Angles 10% Flatter % Change from Base Case	MII	0.23	31,523 4.5	0.791 -1.7	801.7 2.7	57,685 11.9	89,208 9.2	1.83 7.1
10	Mining Cost of USD 0.90/t % Change from Base Case	MII	0.23	32,243 6.9	0.790 -1.9	819.0 4.9	57,018 10.6	89,261 9.3	1.77 3.5
11	Mining Cost of USD 1.10/t % Change from Base Case	MII	0.23	27,867 -7.6	0.819 1.7	733.8 -6.0	45,223 -12.3	73,090 -10.5	1.62 -5.1

Table provided by Independent Mining Consultants, Inc.

Table 17.5
Sensitivity Analysis for Floating Cones Based on Measured and Indicated (MI) Mineral Resources Only

Case	Case	Resource Classes	Grade Cut-off (g/t)	Tonnes (000 t)	Grade (g/t)	Gold (000 oz)	Waste (000 t)	Total (000 t)	Strip Ratio
1	**Base Case – USD 500 Gold**	**MI**	**0.23**	**26,039**	**0.823**	**689.0**	**44,146**	**70,185**	**1.70**
2	Gold Price – USD 550 Gold	MI	0.23	26,919	0.813	703.6	46,021	72,940	1.71
	% Change from Base Case			3.4	-1.2	2.1	4.2	3.9	0.8
3	Gold Price – USD 450 Gold	MI	0.23	23,882	0.839	644.2	38,097	61,979	1.60
	% Change from Base Case			-8.3	1.9	-6.5	-13.7	-11.7	-5.9
4	Recovery of 70.4%	MI	0.23	26,917	0.813	703.6	46,017	72,934	1.71
	% Change from Base Case			3.4	-1.2	2.1	4.2	3.9	0.8
5	Recovery of 57.6%	MI	0.23	23,882	0.839	644.2	38,097	61,979	1.60
	% Change from Base Case			-8.3	1.9	-6.5	-13.7	-11.7	-5.9
6	Process/G&A of USD 2.63	MI	0.23	24,798	0.832	663.3	40,539	65,337	1.63
	% Change from Base Case			-4.8	1.1	-3.7	-8.2	-6.9	-3.6
7	Process/G&A of USD 2.15	MI	0.23	26,443	0.817	694.6	44,710	71,153	1.69
	% Change from Base Case			1.6	-0.7	0.8	1.3	1.4	-0.3
8	Slope Angles 10% Steeper	MI	0.23	26,337	0.824	697.7	41,553	67,890	1.58
	% Change from Base Case			1.1	0.1	1.3	-5.9	-3.3	-6.9
9	Slope Angles 10% Flatter	MI	0.23	25,452	0.824	674.3	44,610	70,062	1.75
	% Change from Base Case			-2.3	0.1	-2.1	1.1	-0.2	3.4
10	Mining Cost of USD 0.90/t	MI	0.23	26,788	0.816	702.8	46,146	72,934	1.72
	% Change from Base Case			2.9	-0.9	2.0	4.5	3.9	1.6
11	Mining Cost of USD 1.10/t	MI	0.23	24,662	0.832	659.7	39,937	64,599	1.62
	% Change from Base Case			-5.3	1.1	-4.3	-9.5	-8.0	-4.5

Table provided by Independent Mining Consultants, Inc.

17.2.6 MINERAL RESOURCE SENSITIVTY TO GOLD PRICE

Both Canadian National Instrument 43-101 and the Australian JORC code state that mineral resources must meet the condition of "a reasonable prospect for eventual economic extraction". For open pit material, IMC recognizes a floating cone geometry at reasonable long term prices, and reasonable cost and recovery assumptions, as meeting this condition for mineral resources.

Table 17.6 shows additional floating cone results at several gold prices from USD 700 to USD 400 per ounce. Measured, indicated, and inferred resources were allowed to contribute to cone economics. IMC suggests that the floating cone at USD 500 gold provides a reasonable estimate of the mineral resource. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Table 17.6 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.

Table 17.6
Floating Cones for Resource Definition – Measured, Indicated and Inferred (MII) Mineral Resources

Case	Gold Price	Resource Classes	Gold Cut-off (g/t)	Ore (000 t)	Gold (g/t)	Gold (000 oz)	Waste (000 t)	Total (000 t)	Strip Ratio
1	700	MII	0.23	41,803	0.738	991.9	89,083	130.886	2.13
2	675	MII	0.23	40.507	0,742	966.3	82,645	123,152	2.04
3	650	MII	0.23	39,381	0,747	945.8	77,940	117,321	1.98
4	625	MII	0.23	38,490	0.749	926.9	73,911	112,401	1.92
5	600	MII	0.23	36,378	0.762	891.2	67,871	104,249	1.87
6	575	MII	0.23	35,096	0.772	871.1	64,904	100,000	1.85
7	550	MII	0.23	34,039	0.780	853.6	62,044	96,083	1.82
8	525	MIL	0.23	32,045	0.790	813.9	55,935	87,980	1.75
9	**500**	**MII**	**0.23**	**30,154**	**0.805**	**780.4**	**51,541**	**81,695**	**1.71**
10	475	MII	0.23	27,967	0.821	738.2	46,138	74,105	1.65
11	450	MII	0.23	27,358	0.826	726.5	44,891	72,249	1.64
12	425	MII	0.23	26,071	0.833	698.2	41,527	67,598	1.59
13	400	MII	0.23	23,522	0.852	644.3	35,930	59,452	1.53

Table provided by Independent Mining Consultants, Inc.

The additional floating cone results in Table 17.6 are approximate as normally the cut-off grade would change with the gold price. For the purposes of this report, however, use of the constant cut-off grade demonstrates the approximate changes in tonnage and grade of the deposit as the gold price fluctuates.

17.3 MINERAL RESERVES

17.3.1 COMPARATIVE CUT-OFF GRADES

Having established a Measured and Indicated mineral resource estimate from the floating cone analysis presented in Section 17.2.5 (see Table 17.5, above), during 2007 IMC then

designed an open pit with haul roads capable of accepting trucks in the 91-t class, and prepared a preliminary production schedule for the extraction of the measured and indicated mineral resources from that pit. Details of the open pit design and the scheduling of production are given in Section 18.1 of this report.

During the course of its work towards the preparation of this study, Micon has identified increases in some of the operating costs forecast for the project since the time at which IMC conducted its open pit optimization, design and production scheduling in 2006 and 2007.

Most notable amongst these is the increase in mining costs per tonne, which have now been priced to include the intended mining contractor's cost of ownership as a cash cost, rather than maintaining the previous assumption of an owner-operated fleet where cost of ownership would have been considered as a capital expense. The revised average mining cost is USD 1.72 per tonne of material based on a mining contractor operation.

The revised costs are summarized in section 18.7.2. However, the rise in forecast gold recoveries and, most significantly, the higher forecast gold price, have off-set such changes, so that the cut-off grades used in the pit design are now seen to be conservative. Micon therefore considers that IMC's resource estimate remains valid. Table 17.7 compares the parameters used in pit optimization with those resulting from the completed preliminary feasibility study.

Table 17.7
Comparative Parameters for the San Francisco Project

Parameters	Resource Estimate Jan-2007	Preliminary Feasibility Study	Preliminary Feasibility Study
Processing Method	**Crusher Feed**	**Crusher Feed**	**Low Grade (ROM)**
Mining Cost per Total Tonne	USD 1.00	USD 1.72	USD 1.72
Processing Cost Per Ore Tonne:			
Crushing	USD 0.86	USD 0.86	nil
Leaching/Plant	USD 1.11	USD 0.87	USD 0.87
Maintenance	USD 0.12	USD 0.12	USD 0.12
Sales	USD 0.10	USD 0.23	USD 0.23
TOTAL	USD 2.19	USD 3.80	USD 3.80
G&A Cost Per Ore Tonne	USD 0.20	USD 0.30	USD 0.30
Gold Recovery	64%	70%	40%
Gold Price Per Troy Ounce	USD 500	USD 686	USD 686
Cut-off Grades			
Internal Cut-off (g/t)	0.23	0.15	0.10
Breakeven Cut-off (g/t)	0.33	0.27	0.21

Table compiled by Micon. Data for Resource Estimate column provided by Independent Mining Consultants, Inc.

17.3.2 MINERAL RESERVE STATEMENT

On the basis that the information presented in Section 18 of this report, the economic viability of the proposed extraction and treatment of the portion of the measured and indicated mineral resource found within the designed open pit has been demonstrated.

Modelling of the San Francisco deposit has been undertaken by IMC in a manner such that the resource tonnage and grade estimates already provide sufficient allowance for grade dilution and losses in mining recovery that no further modifying factors are required before reporting these as a mineral reserve.

Micon therefore considers the measured and indicated mineral resource within the open pit design to be a mineral reserve in terms of the CIM definitions.

Notwithstanding the fact that engineering of certain aspects of the project has reached an advanced stage, Micon considers that, overall, the basic project engineering is at the level normally associated with a preliminary feasibility study. In this study, no break-down of the resources by rock type and recovery on a bench-by-bench or annual basis was available. Only a manual, sectional estimate was available, providing a life-of-mine average. This resulted in "*uncertainties associated with the modifying factors*" in terms of NI 43-101 and, hence Micon has classified both the Measured and Indicated Mineral Resources within the pit as a Probable Mineral Reserve.

Table 17.8 sets out the Mineral Reserves of the San Francisco Project. These reserves are valid as of February 29, 2008.

Table 17.8
Mineral Reserve Estimate

Case	Reserve Class	Gold Cut-off (g/t)	Reserve (000 t)	Grade (g/t)	Gold (000 oz)
High Grade Crusher feed	Probable	0.50	12,000	1.05	403.7
Low Grade Crusher feed	Probable	0.23	4,653	0.88	132.0
Sub-total Crusher feed	Probable		16,653	1.01	535.7
Low Grade ROM leach	Probable	0.28	5,981	0.39	75.3
Grand Total	**Probable**		**22,634**	**0.84**	**611.0**

Compiled by Micon from Schedules provided by Independent Mining Consultants, Inc.

In addition to the Reserve tonnage given above, total waste rock within the final pit outline is estimated to be 46.0 Mt, giving a waste: ore stripping ratio of 2.0:1.

18.0 OTHER RELEVANT DATA AND INFORMATION

The following Section describes the results of the technical work undertaken by Micon which, together with the foregoing mineral resource estimate, comprises a Preliminary Feasibility Study of the re-activation of the San Francisco mine.

18.1 MINING

18.1.1 OPEN PIT MINE DESIGN

18.1.1.1 Geotechnical Studies and Pit Design Criteria

The most recent geotechnical study carried out on the San Francisco pit was conducted by Golder Associates in December, 1996 for the previous owners of the property, Geomaque de Mexico. Golder's scope of work was to carry out site investigations, testing and analysis to develp final slope angle recommendations for the final pit plan.

The recommended overall slope angles ranged from 37° for single 6m benches along the northeast facing slopes to a maximum slope of 56° for double benching in schist units. In its report Golder presented a table of recommended inter-ramp slope angles and catch bench widths to achieve the recommended overall slope angles.

IMC used this information when carrying out the pit optimization analysis and included an allowance for ramp widths in the overall slope angles.

18.1.1.2 Hydrological Considerations

Micon has no information on hydrology in the pit area. However, at the time of Micon's visit, the pit floor was under water and this was understood to represent accumulated precipitation rather than groundwater. The existing pit walls were generally dry, with a few minor seepages along shear zones.

18.1.1.3 Phased Pit Designs

The current emphasis is to extend the old main pit outline towards the newly discovered resources to the northwest. In December, 2006 IMC developed a two-phase pit design. The initial phase was based on a pit optimiation run using a gold price of USD 300/oz and the second (final) phase was an optimization run using a USD 500/oz gold price. Both final and initial pit designs were based on the topography using the existing pit outline. These phased outlines were smoothed to create "mineable" pit outlines and included ramps for access and ore haulage by 100 ton trucks. No revisions have been made to these pit designs since the IMC work. Given the gold prices used by IMC for the pit optimizations, it is Micon's opinion that these outlines would represent a conservative pit design.

Table 18.1 gives the in-pit material on a bench-by-bench basis.

Table 18.1
Mineral Reserve (Measured and Indicated) Bench-by-Bench

Bench	High Grade Ore Crusher Feed (000 t)	Grade (g/t Au)	Low Grade Ore ROM Leach Feed (000 t)	Grade (g/t Au)	Waste Rock (000 t)
728	1	0.510	8	0.314	122
722	49	0.751	113	0.386	777
716	112	0.887	178	0.380	1,581
710	140	0.905	185	0.377	1,886
704	187	0.931	165	0.381	1,904
698	231	0.862	179	0.399	1,930
692	354	1.101	248	0.377	2,340
686	437	1.023	285	0.400	2,528
680	502	0.939	306	0.400	2,509
674	519	0.988	331	0.380	2,414
668	489	1.068	291	0.385	2,304
662	434	1.158	272	0.382	2,154
656	469	1.124	277	0.389	2,058
650	596	0.911	264	0.386	1,765
644	689	0.928	277	0.400	1,595
638	705	1.063	312	0.403	1,553
632	678	1.108	339	0.393	1,573
626	567	1.206	321	0.392	1,648
620	524	1.093	269	0.397	1,661
614	554	1.027	241	0.399	1,542
608	674	1.062	167	0.393	1,424
602	736	1.112	150	0.413	1,317
596	728	1.242	154	0.404	1,248
590	753	1.314	145	0.412	1,144
584	742	1.337	113	0.401	1,067
578	749	1.323	100	0.398	928
572	689	1.220	67	0.395	838
566	654	0.984	59	0.390	684
560	594	0.912	65	0.416	504
554	478	0.936	60	0.410	363
548	404	0.920	30	0.395	230
542	344	0.804	10	0.420	172
536	279	0.674			140
530	232	0.649			91
524	167	0.714			43
518	120	0.834			17
512	72	0.848			7
Total	**16,652**	**1.00**	**5,981**	**0.392**	**46,059**

Figures 18.1 and 18.2 show the existing San Francisco open pit and waste rock dumps, respectively, while Figures 18.3 to 18.7 show the proposed annual pit outlines, with the access ramp included.

Figure 18.1
Existing San Francisco Open Pit

Figure 18.2
Existing Waste Rock Dumps South of San Francisco Open Pit

Figure 18.3
Open Pit Layout After Pre-Production

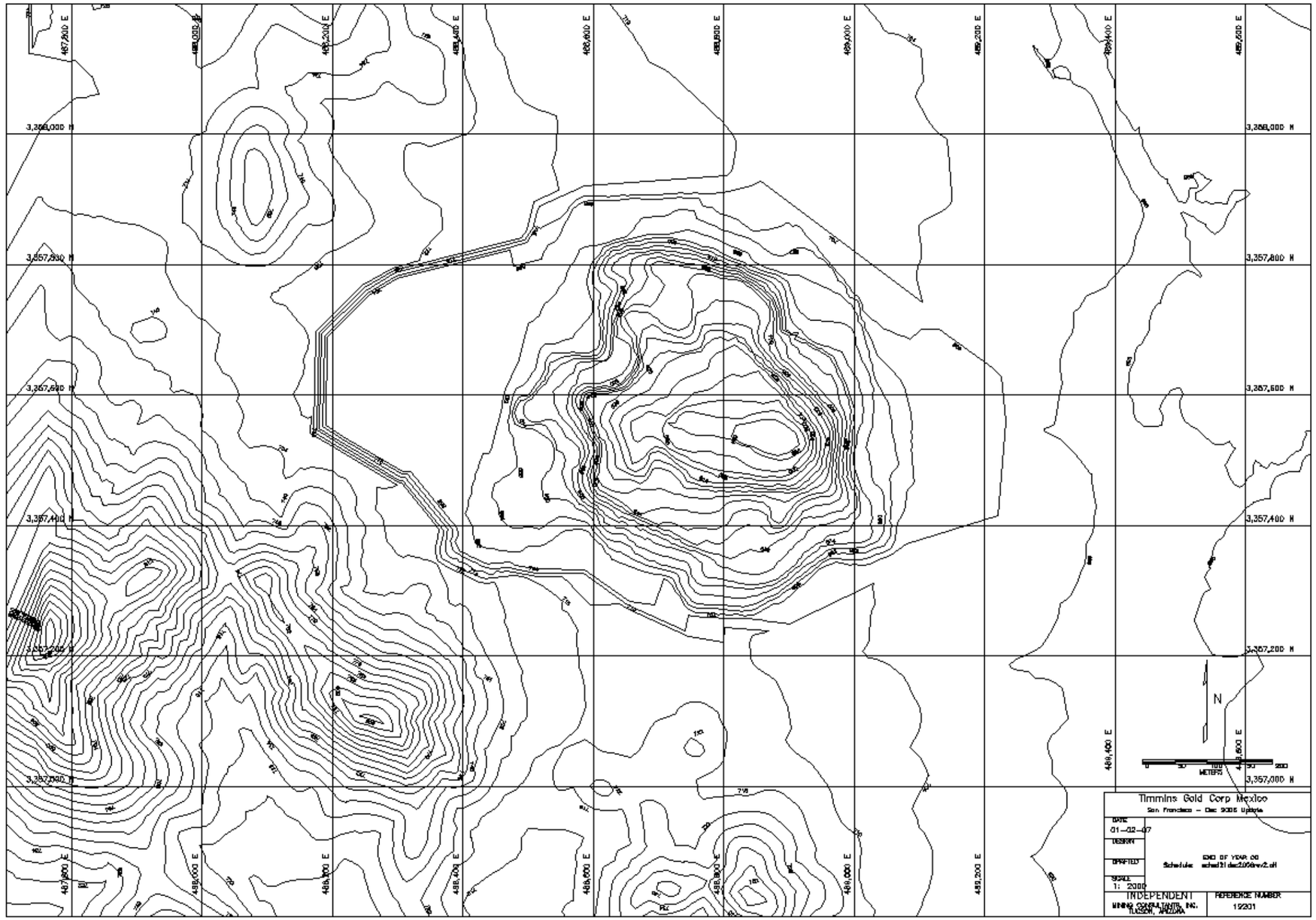


Figure 18.4
Open Pit Layout - Year 1

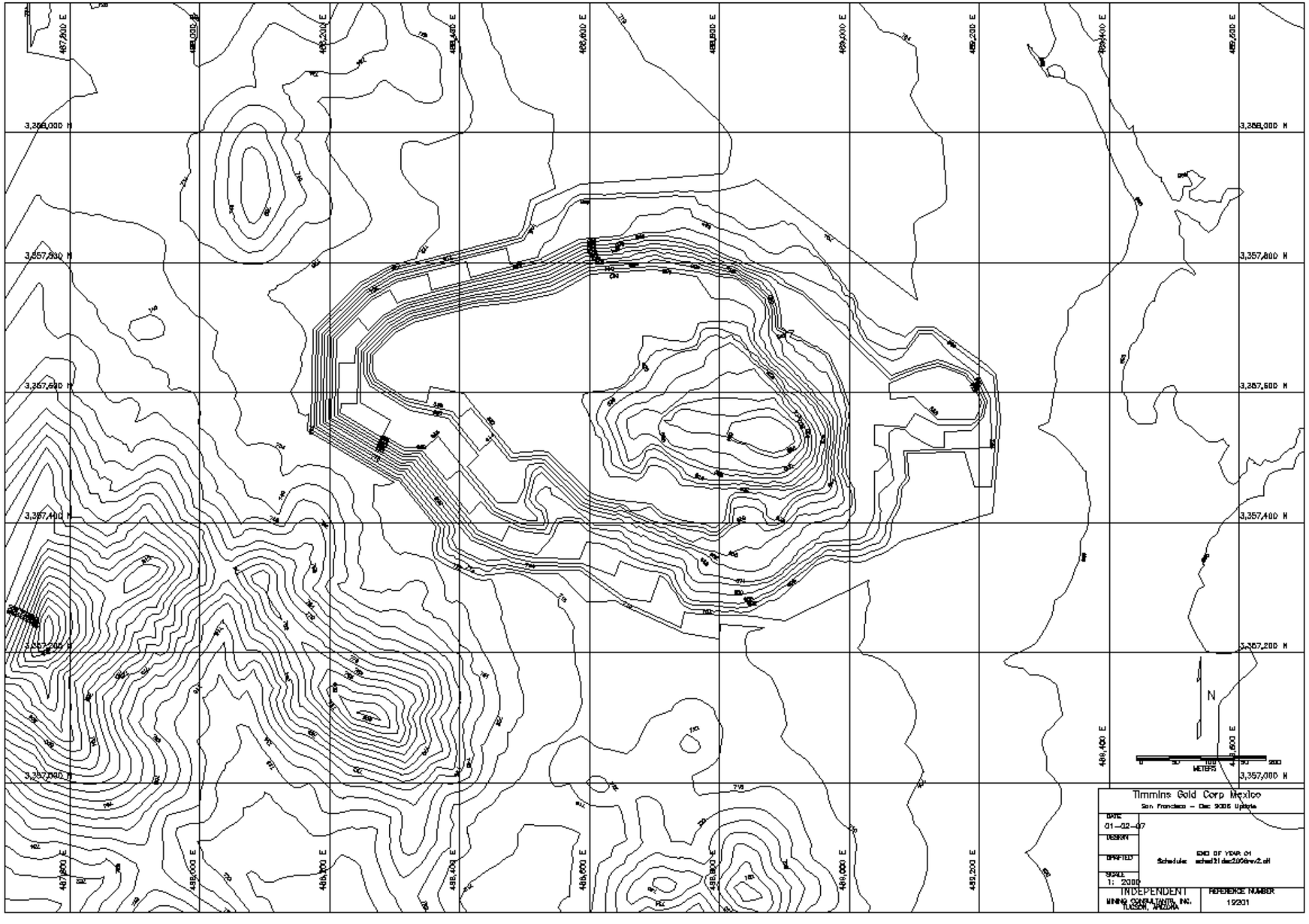


Figure 18.5
Open Pit Layout - Year 2

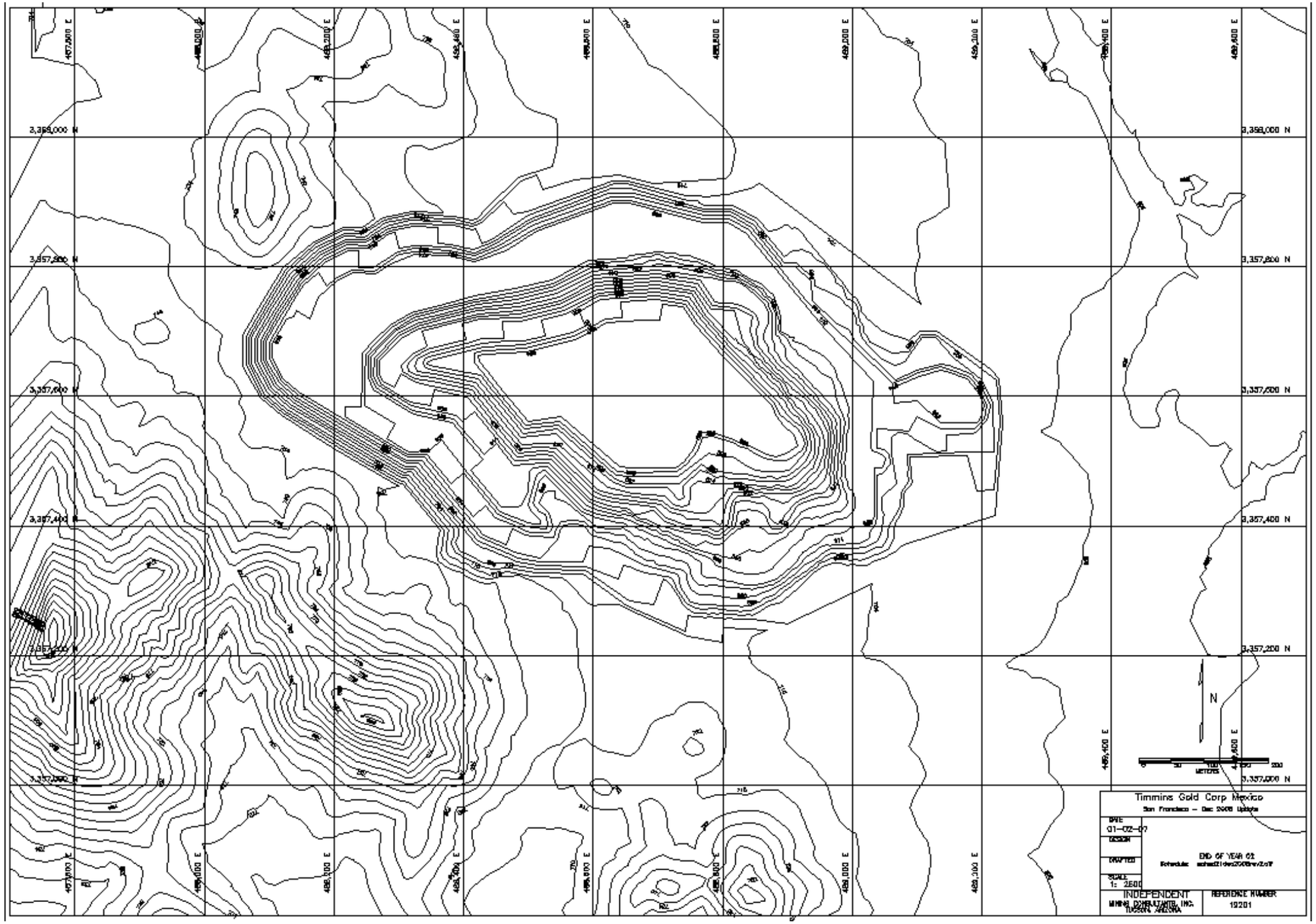


Figure 18.6
Open Pit Layout - Year 3

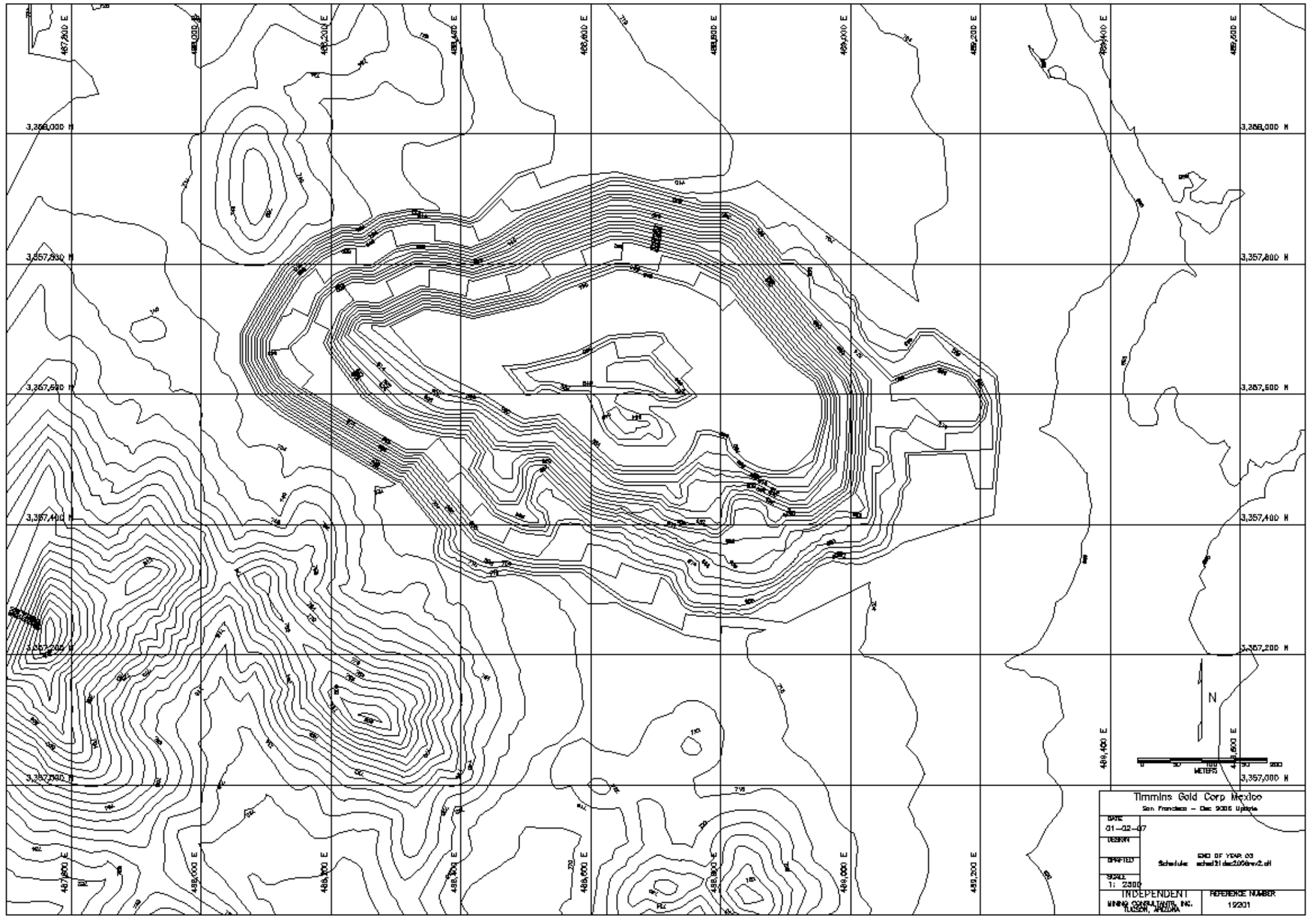
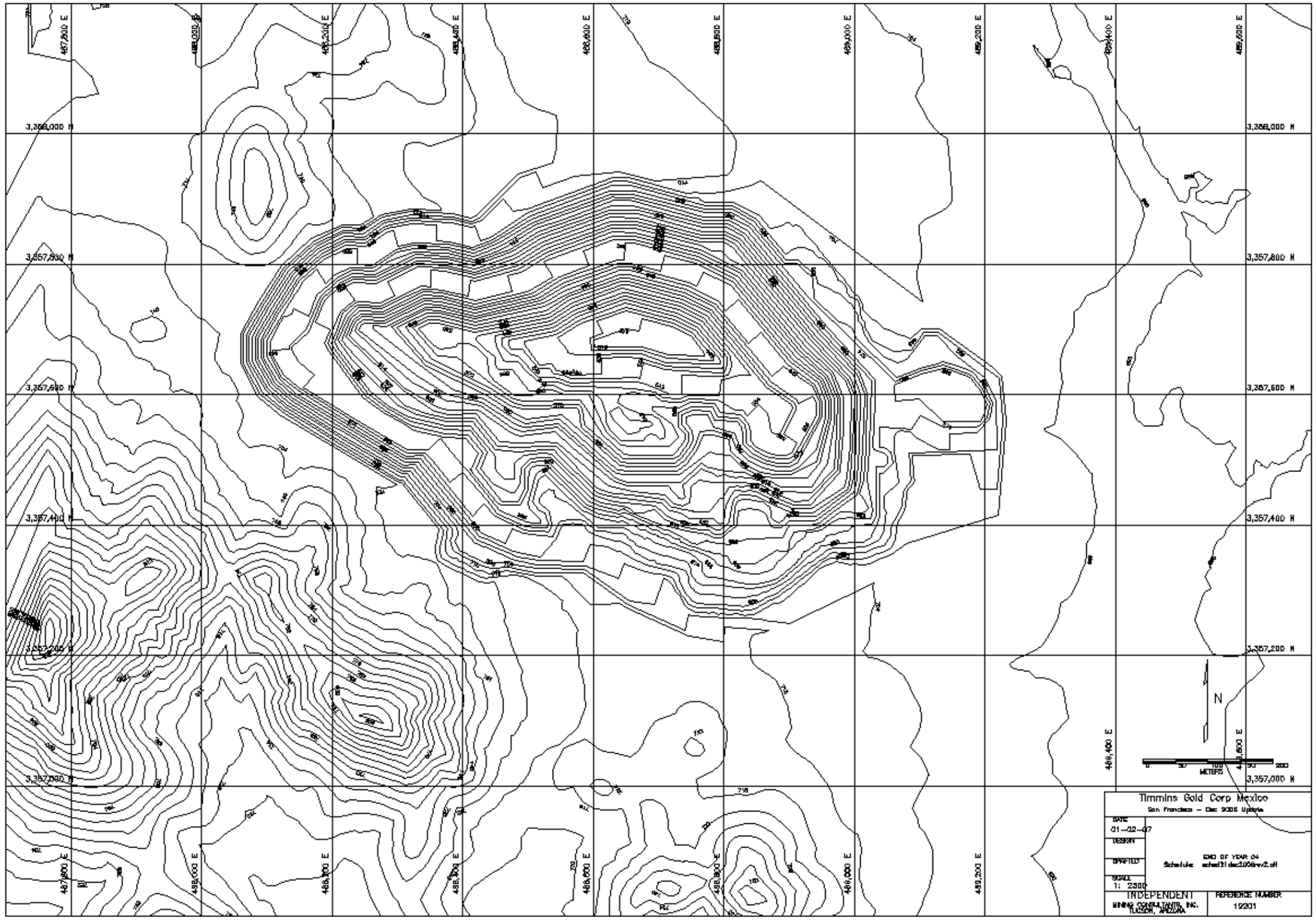


Figure 18.7
Open Pit Layout - Year 4



18.1.1.4 Waste Rock Management

Existing waste rock dumps are located on the south of the main San Francisco open pit, close to the pit rim. Therefore, the dumps cannot be extended to the north. They are also limited in the east by a property boundary and to the west by ground not yet condemned by exploration. Accordingly, with the re-establishment of the operation, these dumps will be extended further south, where adequate space does exist.

18.1.2 MINE PRODUCTION SCHEDULE

Using the phased pit designs described above, IMC developed a mine production schedule that was based on producing a minimum of 125,000 oz of contained gold per year, which equated to 80,000 oz recovered. By using a cut-off grade of 0.5 g/t Au, IMC was able to keep the annual head grade above 1.00 g/t Au. Production was built up over the preproduction and Year 1 period until about 4.0 Mt of ore above the 0.5 g/t Au cut-off grade were mined in Years 2 and 3. In order to maintain production the cut-off grade was lowered in years 4 & 5 to the internal cut-off garde of 0.23 g/t Au. Some 6.0 Mt of low grade material between 0.5 and 0.28 g/t Au were scheduled from the start of pre-production development (PPD) to Year 3. All material below the internal cut-off grade was considered waste.

Since IMC's work in 2006, the production schedule has been revised by Timmins to reduce the maximum production of crush-leach ore from the open pit to about 3.3 Mt per year from Years 1 through 4. This ore is trucked to the three-stage crushing plant for reduction in size to 100% minus half inch. In addition, lower grade ROM ore that is below the 0.5 g/t Au crush ore cut-off grade, but above a revised 0.283 g/t Au internal or low grade cut-off, is scheduled to be mined contiguously with the crush ore and trucked directly to the leach pads. All material below 0.23 g/t Au is considered waste and will be trucked to the nearby waste dumps.

Table 18.2 shows the production schedule.

Table 18.2
Mine Production Schedule

	Unit	PPD	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Crush-Leach Ore	000 t	464	3,300	3,300	3,300	3,300	2,989	16,653
	g/t Au	0.883	1.023	1.091	1.054	0.972	0.868	1.001
ROM Ore	000 t	447	1,784	1,789	1,367	594	0	5,981
	g/t Au	0.383	0.391	0.393	0.393	0.393		0.392
Total Ore	000 t	911	5,084	5,089	4,667	3,897	2,989	22,634
	g/t Au	0.638	0.801	0.845	0.860	0.884	0.868	0.840
Waste	000 t	4,804	12,647	10,733	8,323	6,045	3,507	46,059
Total Tonnes	000 t	5,715	17,731	15,822	12,990	9,939	6,496	68,693
Strip ratio		5.3	2.5	2.1	1.8	1.6	1.2	2.0

18.1.3 MINE OPERATIONS

All mining activities will be carried out by a mining contracting company. For the purposes of this report the quotation used as the basis of the costs was submitted to Timmims by Grupo Peal S.A. (Peal), with headquarters in León, Spain. Peal has a local office located in Hermosillo, Sonora. The contractor will provide all the required mining equipment and personnel to produce the tonnes scheduled in Table 18.2 above. A brief description of the mining contract conditions (quotation only) is given in Section 18.1.4.1 below.

Timmins will provide contract supervision, geology, engineering and planning and survey services using its own employees.

18.1.3.1 Mining Contract

Peal presented its contract mining offer to Timmins in May, 2007. The terms of the offer included a fixed mobilization and demobilization cost and a unit cost per tonne for all material to be mined. Work included in the unit price per tonne in the contract offer was:

- Construct all roadways required to access the mine, waste dumps and crushing plant;
- Construct access and haulage ramps within the pit limits;
- Maintainenance of all roads, dumps, stockpiles;
- Drilling, blasting loading of ore and waste to the crusher and waste dump;
- Supply and maintenance of all equipment required for mining; and
- Supply of equipment oprators, mechanics and shift supervisors.

The unit price per tonne did not include work such as stripping of vegetation and stockpiling of top soil, construction of diversion ditches, preparation of stockpile bases, supply of water or pumping out of the pits, exploration drilling and preparation of the pit slopes for mining. Table 18.3 lists the mining and support equipment the Peal proposes to provide.

Table 18.3
Contract Mining Equipment

Number	Type	Size
2	Tracked drill	6.5 in dia
1	CAT 5110 Hydraulic Excavator	8 m^3
1	CAT 992 Front end loader	12 m^3
1	CAT 385 backhoe	5.4 m^3
12	CAT 777D Haul trucks	91 t
1	CAT D10T Dozer	
1	CAT 14H Grader	
1	CAT 824 RT Dozer	
1	Tyre manipulator	5T
1	Water Tanker	30,000 L
1	Fuel Truck	5,000 L
2	Personnel transport vehicles	

Technical Outline of the Contract

The following technical parameters were provided to Peal or assumed by Peal in developing the contract pricing.

- Assumed powder factor by Peal = 0.200 kg ANFO per tonne of rock blasted. ANFO price used in the contract was USD 0.464/kg.
- Drill pattern to be used = 4.5 m by 5.0m – drilling using a 6.5 in diameter blasthole drill.
- Price of diesel used in estimate = USD 0.50/L.
- Unit mining cost based on a maximum haulage distance of 2,300 m. Haulage of ore or waste beyond that distance to be charged at USD 0.25/t-km.
- Densities of ore = 2.66 t/m^3 and waste = 2.77 t/m^3.
- Shift schedule based on two shifts of 12 hours per day. Over 360 days per year. Three shift rotations will be used to provide the crews on a 24/7 basis.
- Peal estimated that the crew effective work time will be 20 hours/day or an 83% efficiency rate. This provides for a total of 7,200 hours of operations per year.

The Contract Economic Offer (Quotation Only)

The base unit cost per tonne for the pre-production year in the offer of May, 2007 was USD 1.56/t of ore or waste mined and trucked up to 2,300 m. Micon understands that this base price has remained the same for the revised new production schedule of 3.3 Mt/y of crush-leach ore.

For years beyond the pre-production year, Peal has suggested an inflation factor, K, based on indices that reflect the increases or decreases in labour rates, drilling costs, explosive costs, fuel costs, steel and machine costs. The indices are those published by the government or as mutally agreed between Peal and Timmins.

Peal's mobilization cost was stated as USD 750,000 and the demobilization costs as USD 1,200,000. In nominal terms, demobilization costs are subject to a compounded 5% per year increase from the first year. Micon has assumed that this will approximate to Mexican inflation over that period, and has therefore assumed that, in constant money terms, this cost remains fixed.

18.1.3.2 Owner Mining Requirements

The Peal contract contemplates that mining engineering and design services are to be provided by Timmins. Specifically the services to be provided by Timmins include:

- Obtaining of all permits and licences for mining.
- Mine design and planning, grade controil and surveying services.
- Construction of mine site infrastructure such as offices, workshops, warehouses, fuelling station and explosive magazines.
- Supply of electric power, water and telecommunications.
- Security services, safety plans and personnel and first aid stations.

18.1.3.3 Mine Infrastructure and Logistics

The supply of mine infrastructure such as maintenence workshops, offices and fuel bays is excluded from the Peal mining contract and these facilities will be supplied by Timmins for the use of the contractor.

18.1.3.4 Mine Personnel

Timmins will provide contract supervision, geology, engineering and planning and survey services using their own employees. Figure 18.8 shows an organization chart of the proposed mining operation including the organization of the mining contractor's workforce.

18.2 PROCESSING

18.2.1 LABORATORY

The laboratory is well equipped and clean. The supervisor previously worked for SGS and has over 17 years experience. The equipment consists of jaw crushers and ring mill for fine grinding, furnaces for gold assaying and wet assay facilities. The assay procedures were discussed and a detailed write up for all requirements reviewed. Blank samples are introduced for all batches along with certified standards, and check assays are carried out at a certified laboratory. Micon is satisfied that the facility is adequate for producing reliable results from any samples submitted.

Figure 18.8
Mine Area Organization Chart

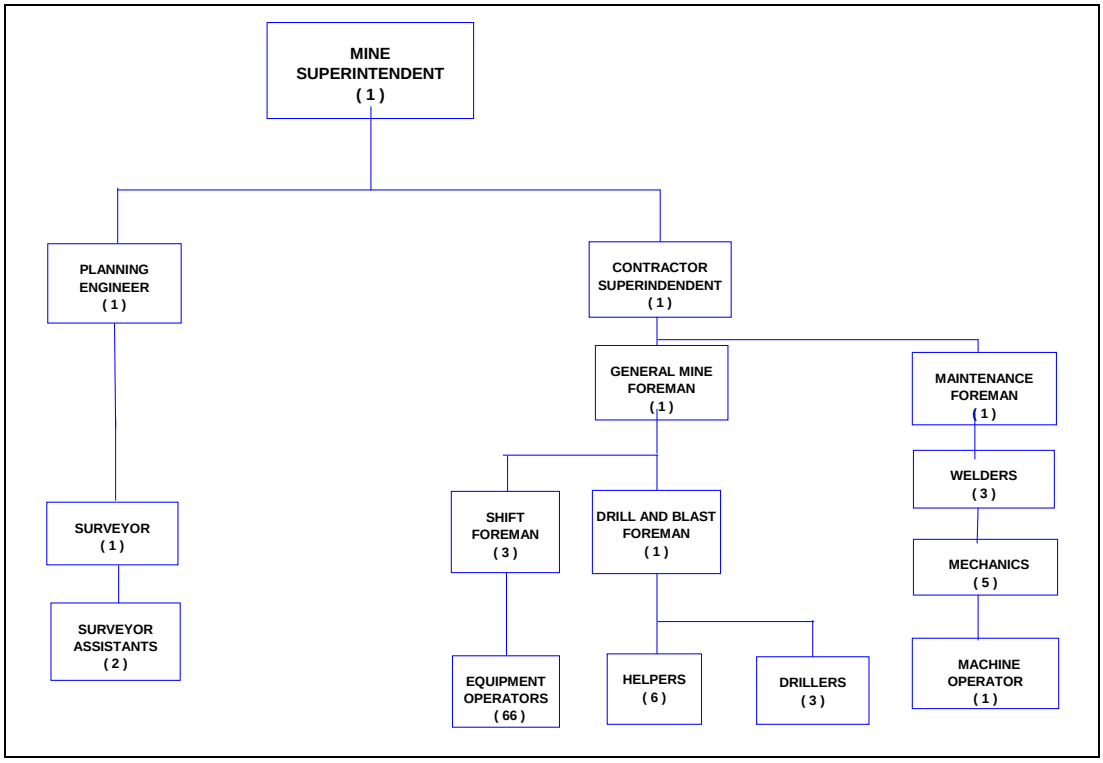


Figure 18.9 shows the sample preparation area of the laboratory. The laboratory has the ability to perform bottle roll tests and has several 2-m high columns for simulation of heap leaching. Timmins also constructed 6-m columns, which represent the planned heap lift height on the leach pad. The columns operate using material at 100% minus ½ in.

Figure 18.9
Mine Laboratory Sample Preparation Area

18.2.2 CRUSHING AND CONVEYING

The existing gyratory primary crusher is still in place but requires some fairly extensive repair work to rebuild worn areas on the lower mating faces of the top spider (see Figure 18.10). Estimates for the rebuild work have been obtained and have been incorporated in the financial model. There is no surge bin, feeder or impact idlers under the primary crusher as would normally be installed to protect the conveyor belts and provide a uniform feed to the following equipment. However, the plant did previously operate in this way and since the extensive civil work required to change the design now would be very expensive, it was decided that the configuration should remain the same as before.

Figure 18.10
(Left) Wear on Lower Mating Faces of Top Spider
(Right) Grasshopper Conveyor Sections

Metso has provided a proposal for the refurbishment of the primary crusher, which has been incorporated into the capital cost estimate. The proposal by Metso was accepted by Timmins and the primary crusher currently being refurbished at Metso's shop with delivery and installation due in April, 2008.

Secondary crushers were part of the original circuit but have now been dismantled as were the "grass hopper" conveyors for feeding ore onto the heaps. Much of the conveyor hardware for feeding the leach pads is on site, although the belts have been removed. Surprisingly, the conveyor rollers and decking appear to be in reasonably good mechanical condition, considering the equipment has been exposed to the elements and not maintained for a number of years. See Figure 18.11, below.

Figure 18.11
Part of Existing Secondary Crushing Circuit and Disused Leach Pad

One reason for the low leach recoveries was apparent during Micon's visit: control of leach pad feed size had been poor and some of material was as large as 3 in (see Figure 18.12). However, in other parts of the heaps, very fine material was also observed, which would be deleterious to leaching, as the fines can reduce permeability.

For the revamped plant, Timmins intends to install secondary and tertiary crushing, with the latter in closed circuit. Sandvik has provided a detailed proposal for the refurbishment of the secondary and tertiary crushing and screening plant which was accepted by Timmins. Timmins has already provided the down payment for the equipment and turn key contract. The detailed proposal by Sandvik was incorporated into the capital estimate.

The foregoing measures will improve particle size control, ensure a more uniform feed size to the pads with consequent better ore exposure for leaching. Despite the size segregation within the pads there were few signs of any significant ponding. It was interesting to note that some natural re-growth on the top of the dumps has occurred since the plant was shut down, an indication of fairly inert material.

Figure 18.12
Coarse Material at the Leach Pad Base

18.2.3 LEACHING

It is Timmins intention to construct a new 47 ha leach pad adjacent to the existing pads. Phase 1 of the new leach pad will occupy 25 ha and be divided into 4 sections to facilitate control of pregnant liquor concentrations. Phase 2, comprised of the remaining of 22 ha, will be constructed in the third year of operation. The new leach pad will require some re-grading of the existing terrain to ensure correct solution flow to the elution plant.

The existing collection ditch liner is in good condition and, although it is not intended to use this part of the plant, its condition is an indication that the existing collection pond liners are also most probably in good condition and these will be used for the plant restart.

Timmins intends to construct a fourth storage pond for intermediate solution strengths, with surplus capacity such that it can also serve as an emergency solution storage pond.

18.2.4 ADSORPTION/DESORPTION/RECOVERY PLANT

The adsorption plant consists of 2 lines of carbon columns each with 5 tanks through which the carbon is advanced counter currently (Figure 18.13). One line of columns contains approximately 2.0 t of carbon and the other 2.5 t. Gold is adsorbed on the carbon to a concentration of approximately 5,000 g/t. Desorption of the carbon is achieved in a Zadra circuit using stainless steel electrodes in a stainless steel electrolytic cell. The stainless cell and cathodes are fairly new and replace the original polypropylene cell with steel wool cathodes. The use of stainless cathodes is more efficient as it does not require the fluxing of voluminous quantities of steel wool which is both time consuming, requires substantially more flux and can lead to inferior grade doré. Adequate carbon regeneration and handling facilities are installed and should require only a minimum amount of work to return them to

service. Currently the plant is being re-habilitated as evidenced in the picture below showing the recently sand-blasted carbon columns on the left.

Figure 18.13
Adsorption Plant Showing the Two Rows of Carbon Columns

18.2.5 PROCESS PLANT LAYOUT

Figure 18.14 shows the proposed site layout, with schematic diagram of the proposed leach pad location and Figure 18.15 illustrates the crushing plant flow diagram.

Figure 18.16 shows the proposed leach pad in more detail, as per the design by Golder Associates. Figures 18.17 to 18.20 show the solution and mass balance for the leaching circuit, and process flow diagrams for the leaching and ADR sections of the plant.

Figure 18.14
Project Site Layout

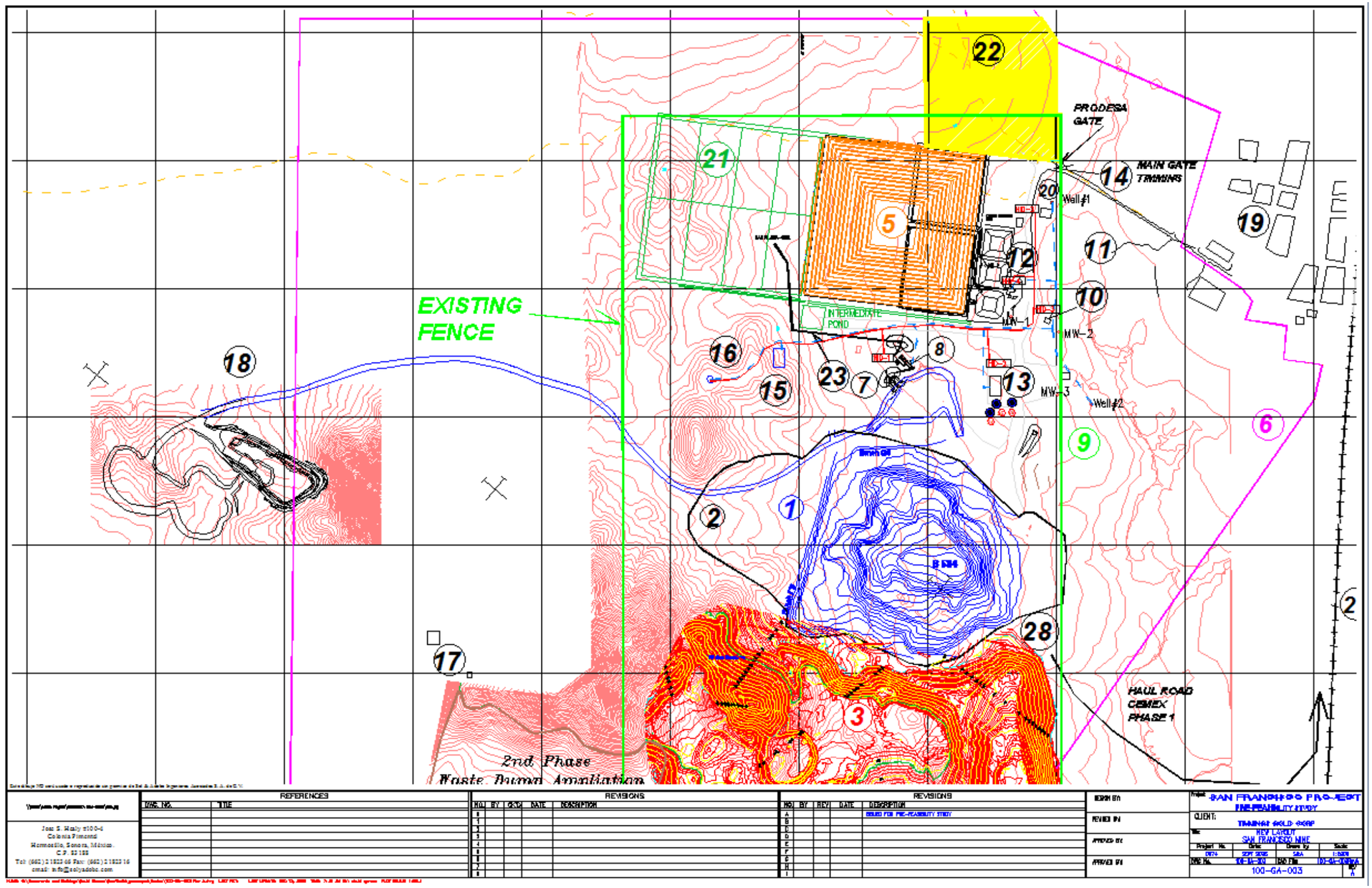


Figure 18.15
Proposed Crushing Circuit

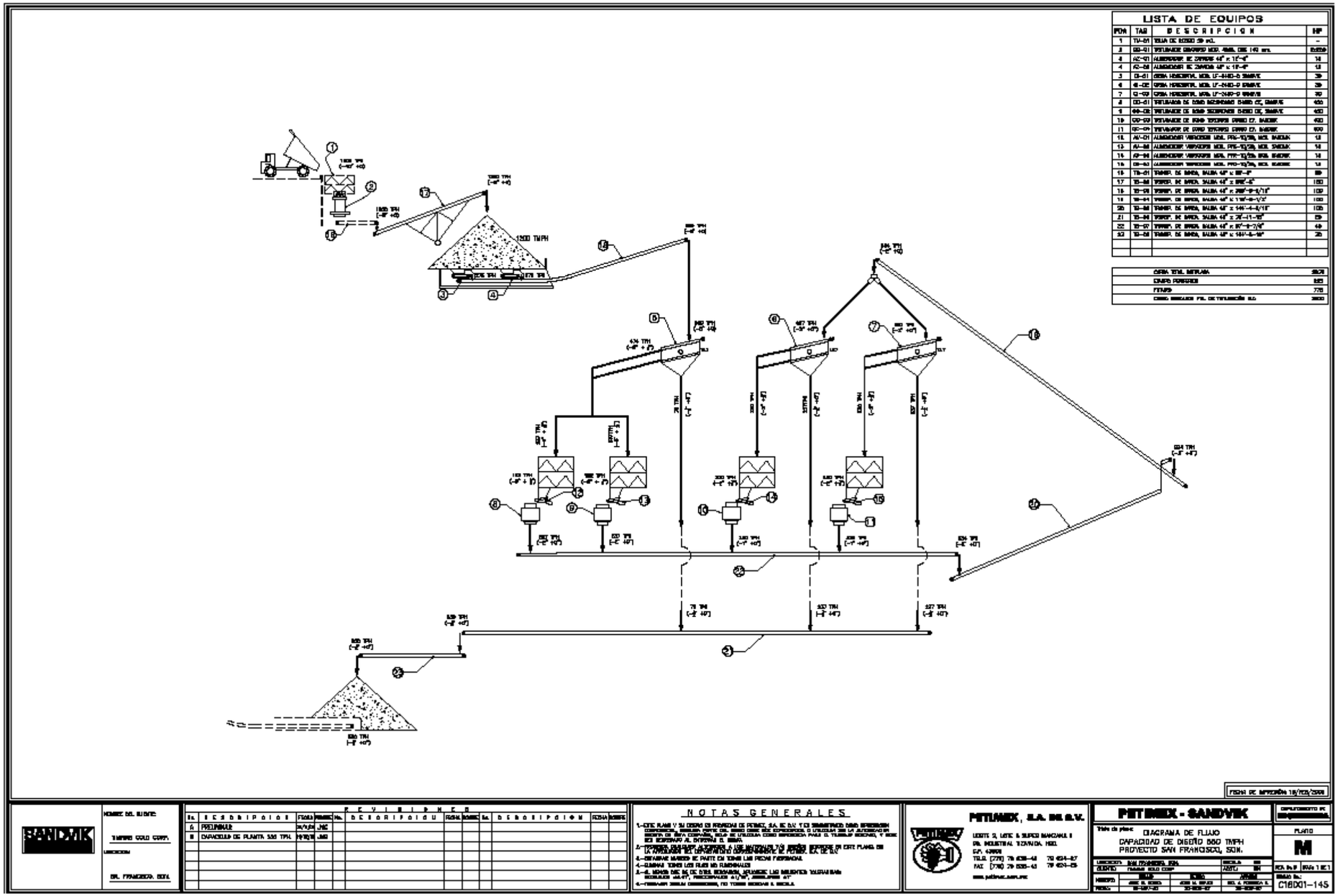


Figure 18.16
Proposed Heap Leach Pad

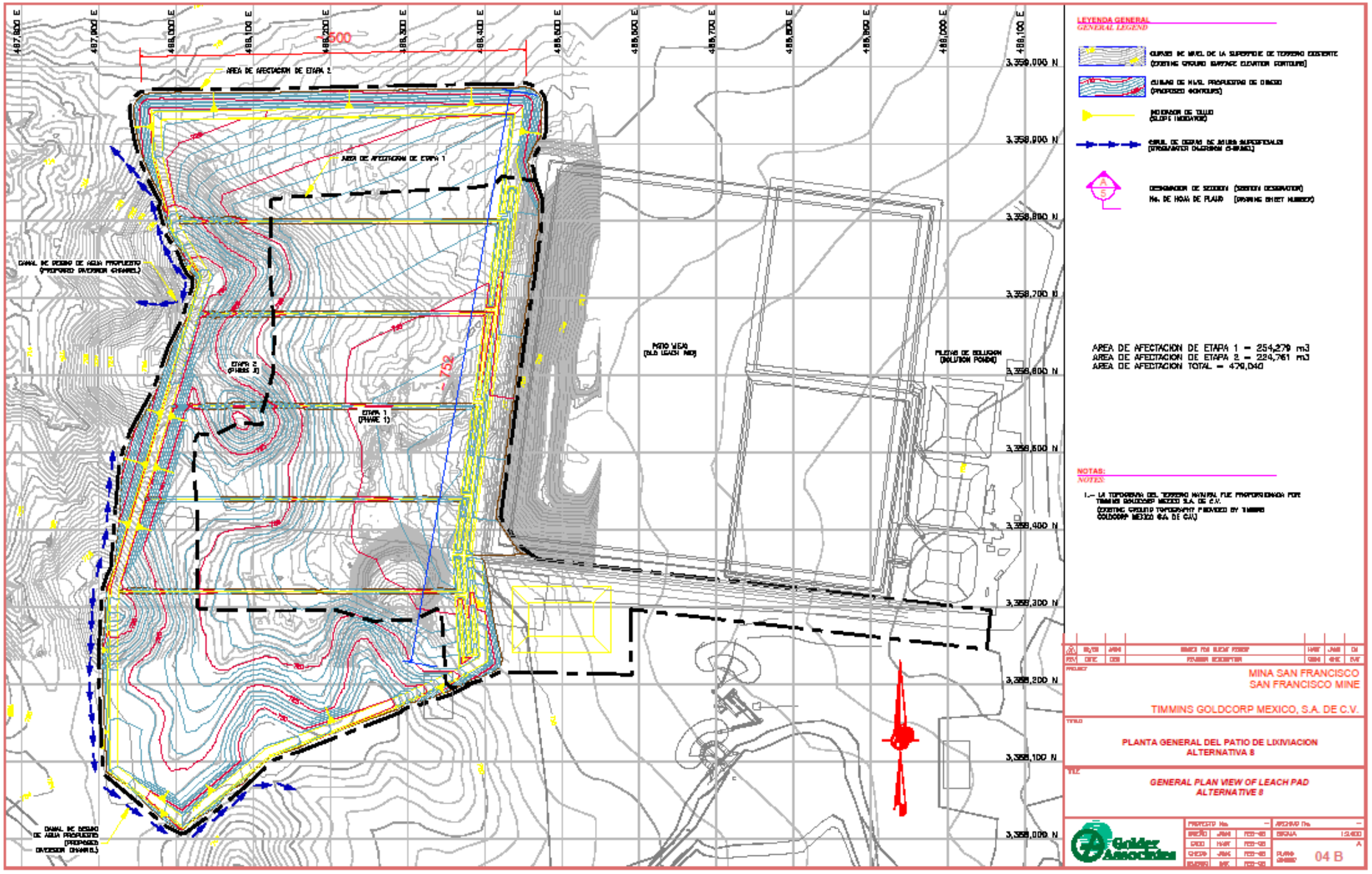


Figure 18.17
Heap Leach Circuit Showing Proposed Solution Balance

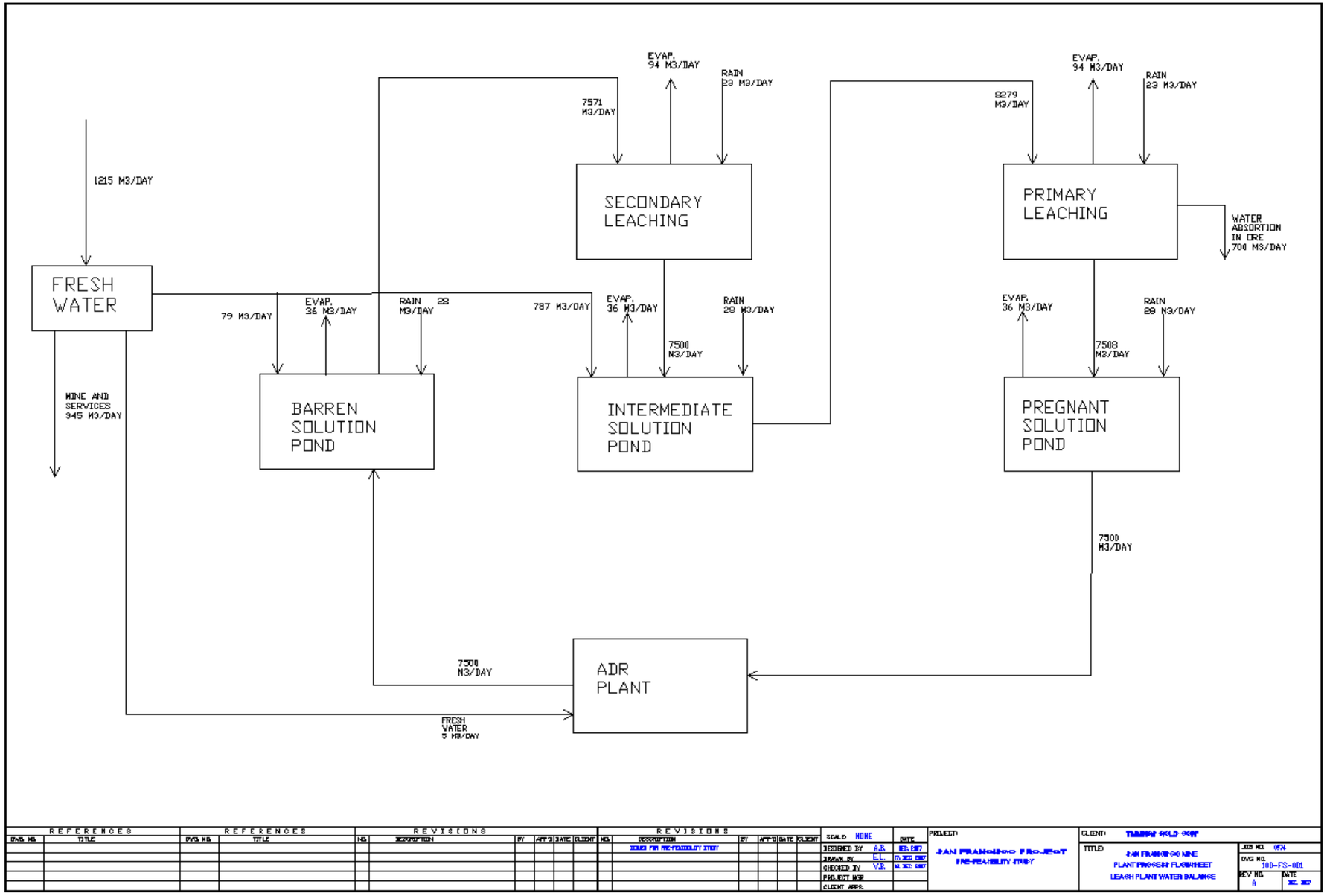


Figure 18.18
Heap Leach Proposed Material Balance

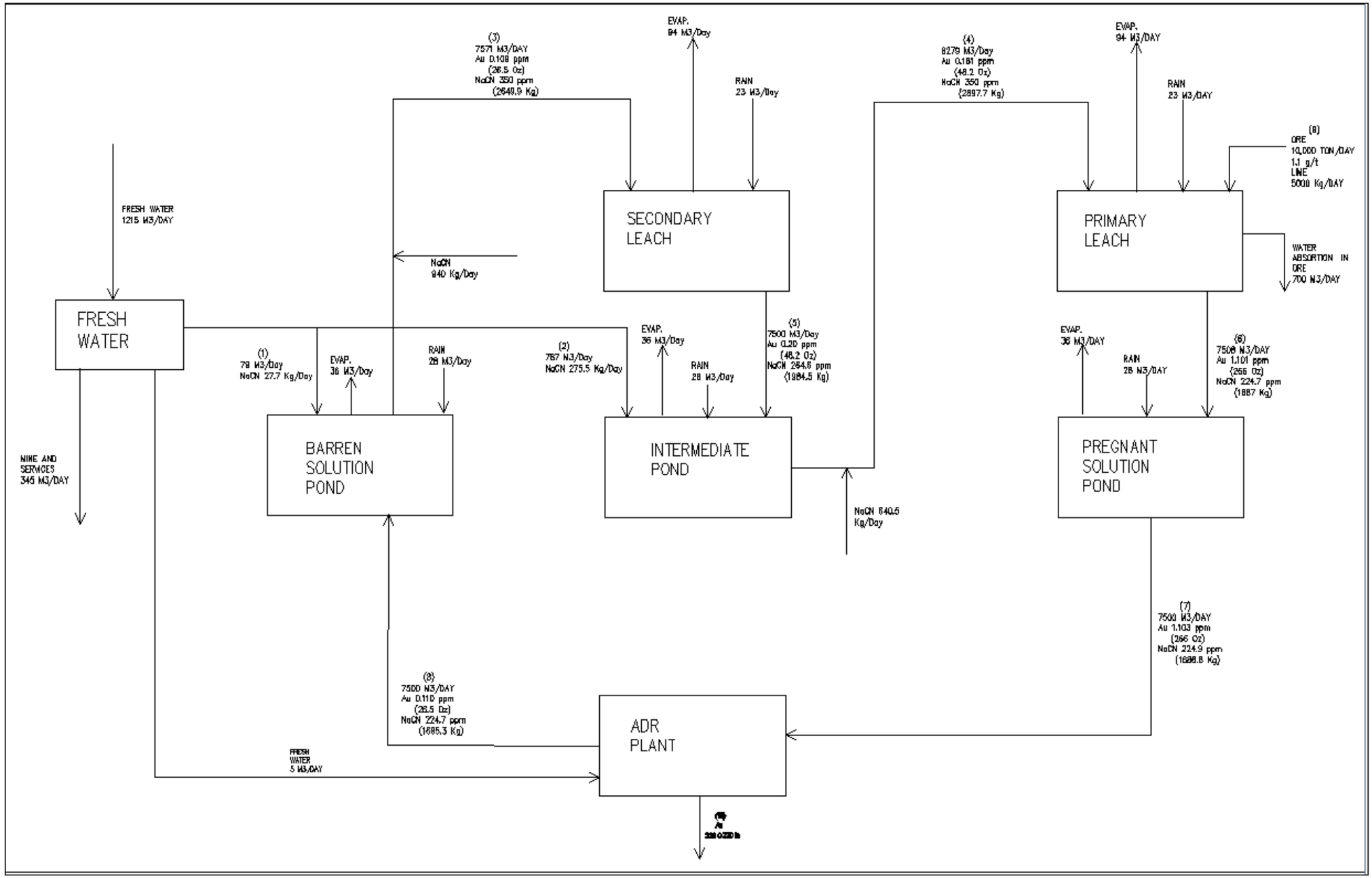
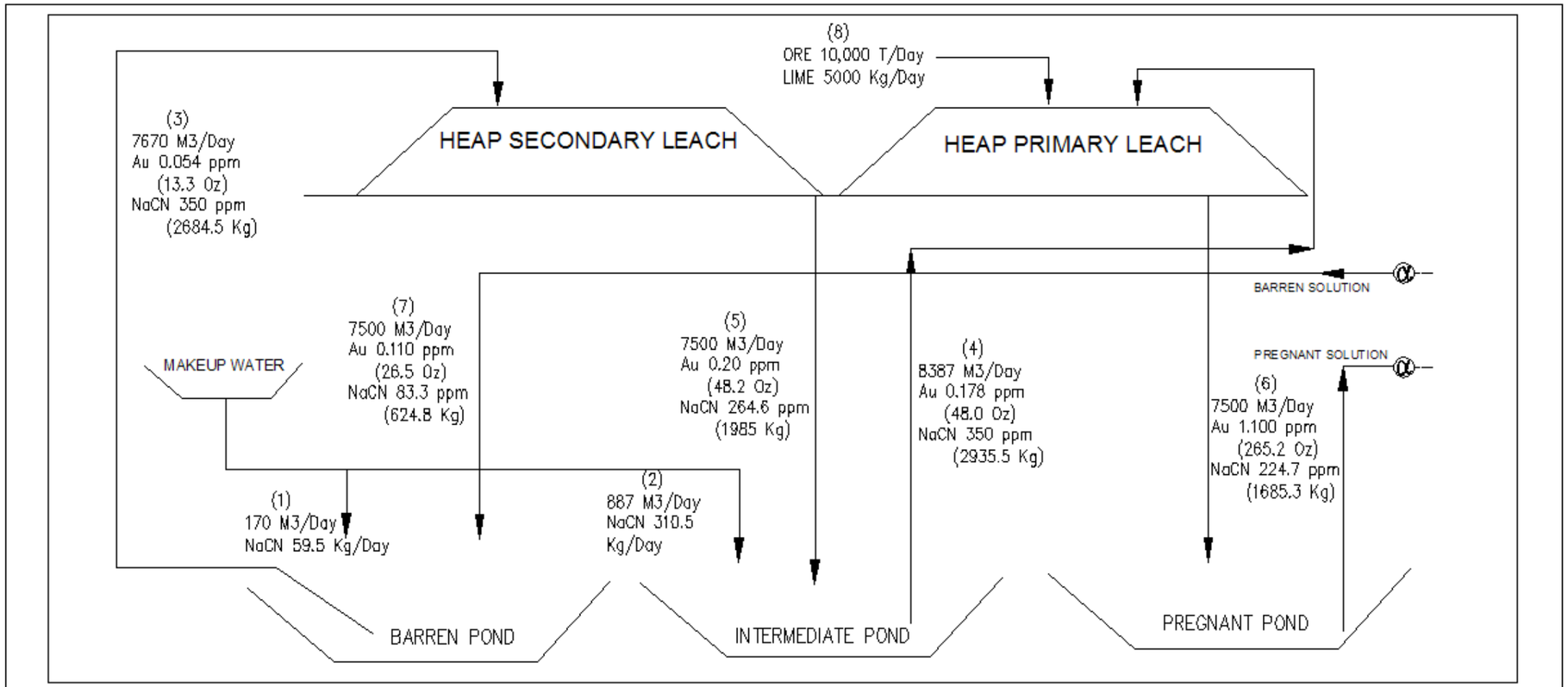


Figure 18.19
Leach Plant Solution Flows



		FRESH WATER		LEACHING SOLUTION		SOLUTION				CARBON			STRIPPING SOLUTION		Au
		BARREN	INTERMED	SECONDARY	PRIMARY	INTERMED	PREG	BARREN	ORE	LOADED	STRIPPED	ACID WASH	PREG	BARREN	DORE
		1	2	3	4	5	6	7	8	9	10	11	12	13	14
FRESH WATER	M³/Day	170	887	-	-	-	-	-	-	-	-	5			
SOLUTION	M³/Day	-	-	7670	8387	7500	7500	7500	10,000	2.0	2.0	-	180	180	
GRADE Au	g/t	-	-	0.054	0.178	0.2	1.10	0.110	1.1	3329	359	-	41.6	4.16	
Au	Kg/Day	-	-	0.41	1.5	1.5	8.25	0.825	11	7.658	0.718	-	7.493	0.749	7.007
Au	OZ/Day			13.3	48.0	48.2	265.2	26.5	353.7	2526	25.3	-	240.7	24.0	226.2
GRADE Ag	g/t	-	-									-			
Ag	Kg/Day	-	-									-			
Ag	OZ/Day	-	-												
pH		-	-	10.5	10.5	10.5	10.5	10.5	-	-	-	3.5	13	13	-
NaCN	ppm	-	-	300	300	2646	2247	2247	-	-	-	-	2000	1700	-
NaCN in Soln	Kg/Day	-	-	2304.5	2935.5	19845	16853	16853	-	-	-	-	360.0	306.0	-
NaCN	Kg/Day	59.5	310.5	5490	6405	-	-	-	-	-	-	-	-	540	-
NaOH	Kg/Day	-	-	-	-	-	-	-	-	-	-	-	-	90.0	-
HCl	Lb /Day	-	-	-	-	-	-	-	-	-	-	65	-	-	-
Carbon	Kg/Day	-	-	-	-	-	-	-	-	-	10	-	-	-	-
LIME	Ton/Day	-	-	-	-	-	-	-	5	-	-	-	-	-	-

REFERENCES		REFERENCES		REVISIONS							REVISIONS						
DWG. NO.	TITLE	DWG. NO.	TITLE	NO.	DESCRIPTION	BY	APP'D	DATE	KLIENT	NO.	DESCRIPTION	BY	APP'D	DATE	KLIENT
											ISSUED FOR PRE-FEASIBILITY STUDY				

SCALE: NONE

		DATE
DESIGNED BY	A.B.	DEC. 2007
DRAWN BY	E.L.	17. DEC. 2007
CHECKED BY	V.B.	18. DEC. 2007
PROJECT MGR.		
CLIENT APPR.		

PROJECT: SAN FRANCISCO PROJECT PRE-FEASIBILITY STUDY

CLIENT: TAMPOR GOLD CORP

TITLE: SAN FRANCISCO MINE PLANT FINISHER FLOWSHEET LEACH PLANT MATERIAL FLOWS

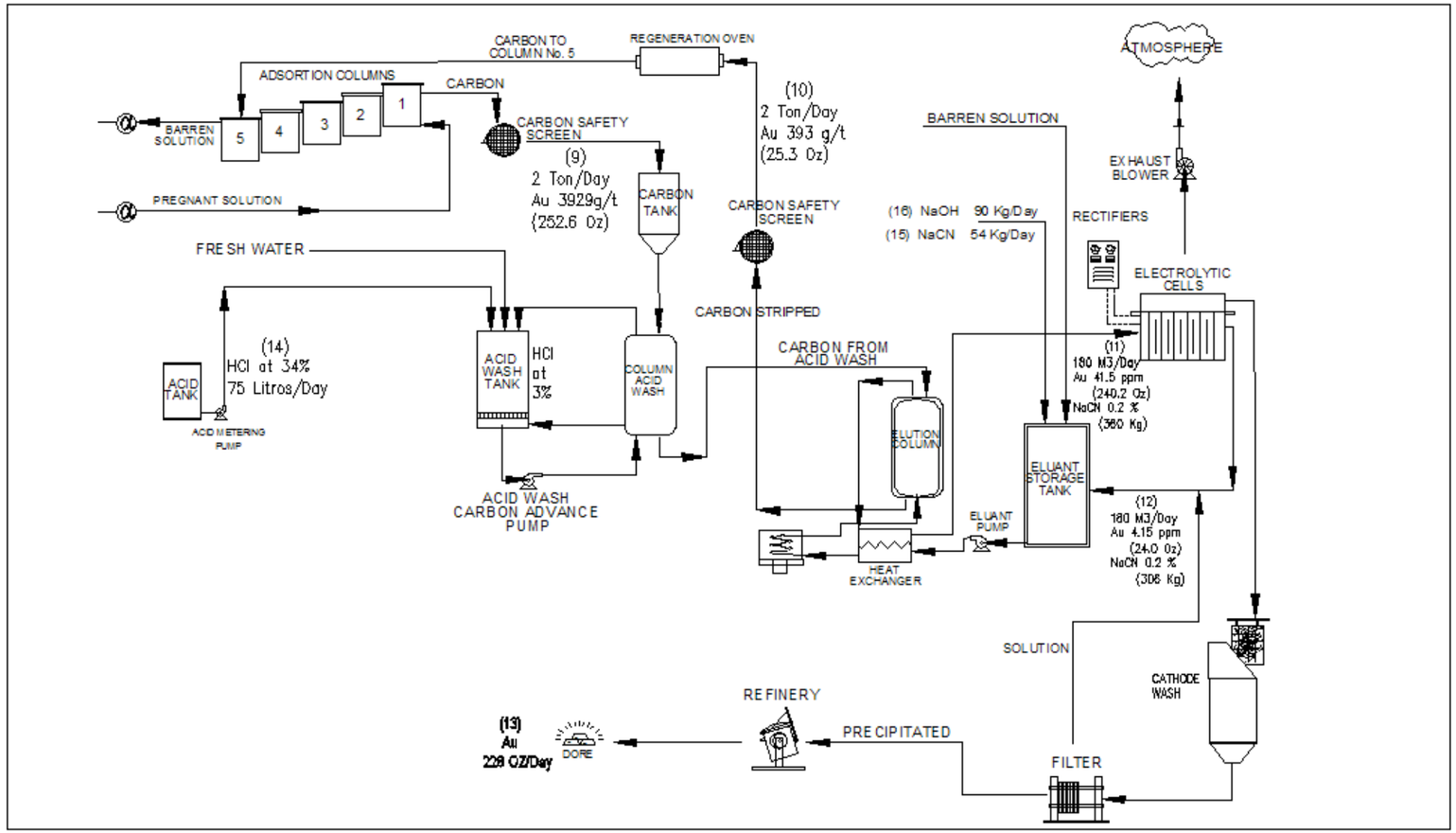
JOB NO. 0874

DWG NO. TDO-FS-003

REV NO. A

DATE DEC. 2007

Figure 18.20
Existing ADR Circuit



CONSUMPTION		MONTHLY	ANNUAL
FRESH WATER	M^3	31,860	382,320
NaCN	Kg	60,135	721,620
NaOH	Kg	2,700	32,400
HCl	Lts	1,950	23,400
Carbon	Kg	2,000	24,000
LIME	Ton	150	1,800

PRODUCTION		MONTHLY	ANNUAL
Au	Kg	213	2,340
Au	OZ	6,840	75,240
Ag	Kg		
Ag	OZ		

REFERENCES | REFERENCES | REVISIONS | REVISIONS

SCALE: NONE
DESIGNED BY A.B. DEC. 2007
DRAWN BY E.L. 27 DEC. 2007
CHECKED BY V.B. 28 DEC. 2007
PROJECT MGR
CLIENT APPR.

PROJECT: SAN FRANCISCO PROJECT
PRE-FEASIBILITY STUDY

CLIENT: TRIBUNE GOLD CORP

TITLE: SAN FRANCISCO MINE
PLANT PROSESS FLOWSHEET
LEACH PLANT GOLD RECOVERY BALANCE

JOB NO. 0204
DWG NO. 100-FS-0104
REV NO. A
DATE DEC. 2007

18.2.6 PROCESS DESIGN CRITERIA

Table 18.4 shows the principal design criteria applied to the project.

Table 18.4
Process Design Criteria

Heap Leaching	Unit	Capacity
Treatment rate	t/d	11,000
Leach Cycle	days	70
Active heap	t	770,000
Swell factor		35%
Heap bulk density	t/m^3	1.93
Active heap	m^3	399,808
Lift height	m	6.00
Active area	m^2	66,635
Irrigation rate	$L/m^2/h$	10.00
Pumping rate	m^3/h	666.3
Pumping rate	m^3/t ore	1.454
ADR Section	**Unit**	**Capacity**
Average ore grade	g/t Au	1.00
Gold leached	%	70%
Gold leached/day	g/d	7,706
Preg Solution (ave)	g/t Au	0.48
Adsoption stages	Number	5
Stage Efficiency	%	50%
Barren solution	g/t Au	0.02
Adsorption recovery	%	96.9%
Gold adsorbed/day	g/d	7,465
Carbon Loading	Au g/t	3,600
Carbon per Column	kg/d (=kg/stage)	2,070
Carbon in Circuit	5 columns (kg)	10,350
Acid wash/Elution	2 columns	4,140
Regeneration,etc	2 columns	4,140
Carbon inventory	kg total	18,630

It is evident from the above that a single line of 5 columns each with at least 2 t of carbon should be sufficient to handle the expected production from the project. The existence of the second line therefore provides a high level of redundant capacity which could cater for unexpected fluctuations in solution tenor and provide a good deal of flexibility in the operating schedule.

18.2.7 MANPOWER

The operating cost estimate makes provision for the following levels of manpower:

Maintenance	43
Crushing	42
Adsorption/Desorption/Recovery	18
Laboratory	13
Total Processing Department	**116**

It is intended to operate two 12 hour shifts, seven days per week, a four day on/four day off schedule thus requiring four crews of up to 29 persons each; on day shift, there will be a need for heap leach solution piping movements and also additional people in the laboratory. The provision for labour to operate the crushing system comprising one primary and three secondary/tertiary crushers plus conveying to the leach dumps is considered generous. Also, since it is intended to only operate one set on carbon columns, fewer operators will be required in this plant.

The labour estimate is therefore regarded as conservative. Once steady state operations have been attained, natural attrition may be allowed to reduce the headcount and improve labour productivity.

18.2.8 CONSUMABLES & MAINTENANCE REQUIREMENTS

Regent consumptions (Table 18.5) are in line with industry norms although the cyanide and lime consumptions considered are higher than those experienced during testwork. However, since plant conditions are rarely controlled as well as laboratory conditions, it is a conservative assumption and should allow more flexibility in the operation.

Table 18.5
Operating Cost - Process Reagents

Reagent	Unit	Cons	Consumption	Unit cost		Annual Cost
		Unit/t	(kg/y)	USD/kg	USD/oz	USD/y
Sodium Cyanide	kg	0.200	720,000	2.00	19.20	1,440,000
Lime	kg	0.500	1,800,000	0.10	2.40	180,000
Caustic Soda	kg	0.009	32,400	1.02	0.44	33,048
Hydrochloric Acid	L	0.010	36,000	0.40	0.19	14,400
Carbon	kg	0.009	32,400	2.95	1.27	95,580
Anti Scalent	L	0.013	46,800	2.60	1.62	121,680
Propane	L	0.040	144,000	5.16	9.91	743,040
Borax	kg		2,304	1.00	0.03	1,958
Silice	kg		1,404	1.10	0.03	2,555
Sodium Carbonate	kg		468	0.52	0.01	243
Potassium Nitrate	kg		230	0.96	0.01	221
Fluorite	kg		230	2.06	0.01	475
Total Reagent					**35.10**	**2,633,201**

(1) Consumption based on similar sized plant operating data
(2) Water consumption based on 9% takeup in the ore.

18.2.9 PROCESS PRODUCTION SCHEDULE

Table 18.6 shows the forecast production from high grade (crushed) ore and from low grade ore (treated uncrushed as ROM).

Table 18.6
Process Production Schedule

		2008	2009	2010	2011	2012	2013	Total
High Grade Ore Treated	000 t	464	3,300	3,300	3,300	3,300	2,989	16,653
	g/t Au	0.88	1.02	1.09	1.05	0.97	0.87	1.00
	Au (kg)	410	3,376	3,600	3,477	3,207	2,594	16,664
Gold recovery	%	70.00	70.00	70.00	70.00	70.00	70.00	70.00
	Au (kg)	287	2,363	2,520	2,434	2,245	1,816	11,665
Low Grade Ore Treated	000 t	447	1,784	1,789	1,367	594	-	5,981
	g/t Au	0.38	0.39	0.39	0.39	0.39	-	0.39
	Au (kg)	171	698	703	537	233	-	2,342
Gold Recovery	%	40.00	40.00	40.00	40.00	40.00	40.00	40.00
	Au (kg)	68	279	281	215	93	-	937
Total Ore Heaped	000 t	911	5,084	5,089	4,667	3,894	2,989	22,634
Gold Production	kg	355	2,642	2,801	2,649	2,339	1,816	12,602
Gold Inventory	kg	44	113	120	114	100	-	
Gold Sales	kg	311	2,573	2,794	2,656	2,352	1,916	12,602
Gold Sales	**oz**	**9,995**	**82,734**	**89,829**	**85,377**	**75,615**	**61,598**	**405,148**

18.2.10 SUMMARY

The metallurgical treatment facilities (ADR) are currently being revamped and are expected to be in “as new condition” in preparation for the plant start-up. Adequate metallurgical testwork has been carried out and the projected average gold recovery for the material is justified. Nevertheless, to ensure that the projected recoveries are achieved, it is essential that the high grade ore be crushed to the size used in the testwork.

18.3 ADMINISTRATION, ENGINEERING AND INFRASTRUCTURE

18.3.1 MANPOWER STRUCTURE

Table 18.7 shows the total manpower proposed for the San Francisco project, excluding employees of the mining contractor and commercial security personnel.

Table 18.7
Manpower structure

Description	Number
General Manager	1
Accounting	6
Purchasing	4
Human Resources	14
Subtotal G&A	**25**
Environmental	3
Geology	7
Mining	5
Subtotal Technical services	**15**
Crushing	42
Maintenance	43
Adsorption/Desorption/Recovery	18
Laboratory	13
Subtotal Processing Department	**116**
Grand Total	**156**

18.3.2 OFFICES, WORKSHOPS AND STORES

Office space form the mine is provided in an existing, purpose-built structure of approximately 450 m^2 located on the property southeast of the ADR complex. The building has adequate working space for the on-site mine administration and also provides basic catering and ablution facilities.

An existing vehicle workshop south of the ADR complex and north of the open pit occupies more than 660 m^2 and can accommodate the off-road haul trucks, excavators and ancillary vehicles envisaged for use in the open pit mining operation. This builing requires some refurbishment before it can be brought back into use. Figure 18.21 shows the rear of the building, with some cladding missing.

A general mine store of almost 200 m^2 located north of the ADR complex accommodates process reagents and mechanical spares. Bulk lime for the heap leach process is stored in a silo near the crushing plant.

Figure 18.21
Mine Vehicle Workshop Building
(Rear View)

18.3.3 ELECTRICAL POWER SUPPLY

Electrical power supply to the mine is delivered through a 33-kV overhead line from the utility, CFE. From the main metering point, the power is distributed to the crushing and screening plant at 4,160 V and to other site infrastructure at 480 V. At the crushing and screening plant, separate transformers feed the principal equipment. Installed transformer capacity is planned as follows:

	kVA
• Primary Crusher	1500
• Secondary/Tertiary Crushers	2500
• Stacker conveyors	1000
• Leach solution pumps	1000
• ADR complex	500
• Laboratory	150
• Office	75
• Workshop	75
• Open Pit	75
• Water Well #1	45
• Water Well #2	45
• Lighting	50

Figures 18.22 and 18.23 show single line diagrams of the electrical systems.

Figure 18.22
Electrical Supply – Single Line Diagram (4160 V)

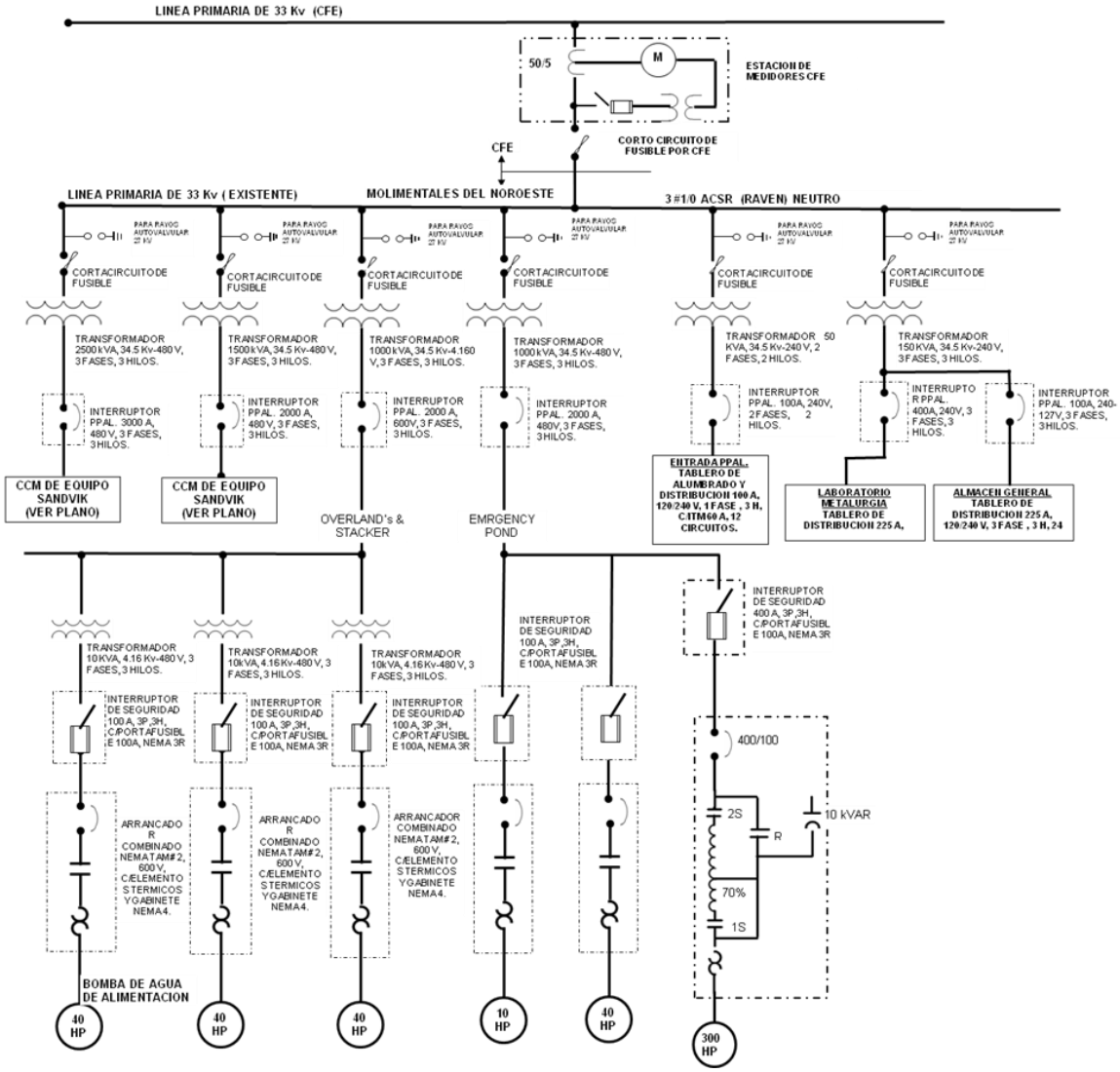
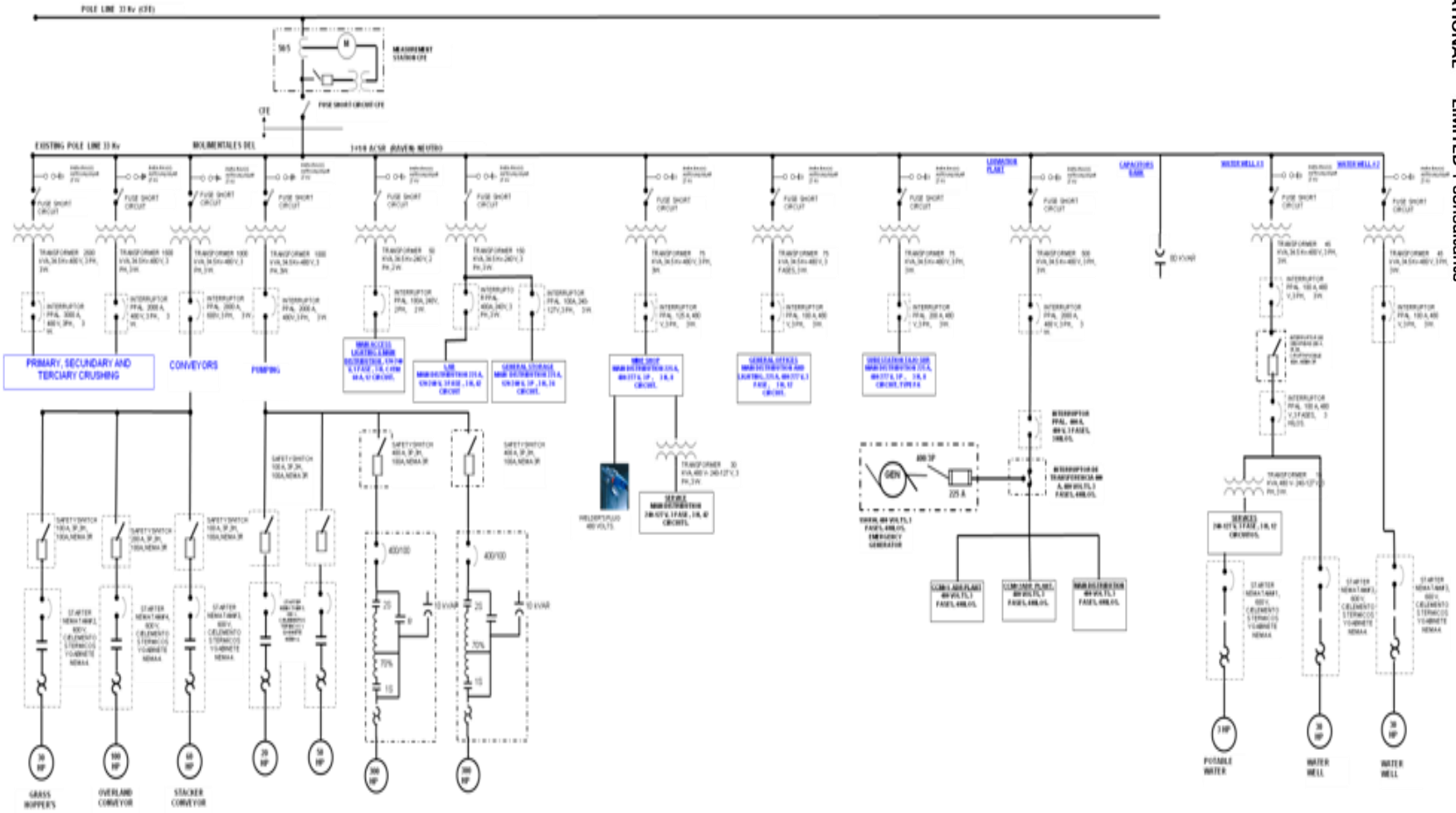


Figure 18.23
Electrical Supply – Single Line Diagram (480 V)



18.3.4 WATER SUPPLY

Approximately 1,000 m^3 per day of fresh water is required in the leaching and ADR plants. Some water will be available from dewatering of the open pit and seasonal rainfall within the leach system. However, the principal source of supply will be two existing water wells on the property.

18.4 ENVIRONMENTAL AND SOCIAL CONSIDERATIONS

Timmins is in the process of renewing its permits for mine operations. The following sections describe the legislative framework and process for the permitting, the environmental and social baseline conditions at the mine, the potential environmental and social impacts, and protection and mitigation measures that are being proposed to minimize these impacts.

18.4.1 MEXICAN LEGAL AND ADMINISTRATIVE FRAMEWORK

Mexican laws and regulations emanate from the Political Constitution of the United Mexican States. Environmental matters are no exception. The Mexican Constitution states that every person in the country has the right to enjoy and live in a healthy environment (Article 4). In accordance with this mandate, regulations must be issued in order to protect the environment, to achieve ecological equilibrium and to regulate human settlements (Article 27). The Federal Congress is empowered to approve laws related to mining, national waters and environmental protection (Article 73). Non hazardous wastes and urban development are under the jurisdiction of the municipalities (Article 115).

18.4.1.1 Mining Act and Regulations (LM)

The Mining Act regulates all mining activities in Mexico, specifically the granting of Mining Concessions. The Mining Bureau of the Economy Secretariat is the competent authority regarding these matters.

The Mining Act makes specific reference to environmental matters. Mining concessionaries must carry out their activities according to environmental regulations. (LM Article 27, Paragraph IV & LM Article 39). However, mining authorities do not have the faculties to enforce the environmental law. In addition to the Mining Concession, if a project is to be developed within the limits of a Natural Protected Area, the miner must get a special permit from the Comisión Nacional de Áreas Naturales Protegidas (CONANP).

18.4.1.2 General Law of Ecological Equilibrium and Environmental Protection (LGEEPA)

The General Law of Ecological Equilibrium and Environmental Protection is the main act that regulates all environmental aspects. The mining industry is regulated under this federal jurisdiction (LGEEPA Article 5, Paragraph XIV).

All activities that may significantly affect the surrounding environment is required to submit to the Dirección General de Impacto Ambiental (DGIRA) an Environmental Impact Manifest in order to obtain an Environmental Impact Permit. Mining is considered a high impact and risk activity (LGEEPA Articles 28 and Regulations for Environmental Impact). All activities must comply with existing related standards. Mining activities standards are specifically mentioned in Articles 108 and 109 of the LGEEPA.

All commercial and industrial activities must have control of their atmospheric emissions and comply with all related standards (LGEEPA, Article 111). Additionally, all high risk activities (above threshold quantities for toxic or flammable chemicals) must submit a Risk Assessment to the DGIRA (LGEEPA, Article 147).

The Mexican environmental enforcement agency (PROFEPA) is entitled to carry out site inspections and to apply different kind of sanctions, ranging from fines up to partial and/or complete shut down of operations and facilities, temporarily or permanently (LGEEPA, Article 171).

All atmospheric emissions coming from federal jurisdiction sources, such as mining, must comply with all standards included in the related Atmospheric Protection Regulations (Reglamento de la Ley General del Equilibrio Ecologico y la Protección al Ambiente en Materia de Prevención y Control de la Contaminación de la Atmosfera) regarding this matter (Articles 11 and 16). The main obligations for atmospheric emissions are to use control equipment, to keep an emissions inventory, to operate sampling sites, carry out perimeter monitoring (Article 17), and to obtain an Operating License from the Secretariat (Article 18).

Mining projects must prepare an Environmental Impact Manifest (MIA) according to the LGEEPA Environmental Impact Assessment Regulations (Reglamento de la Ley General del Equilibrio Ecológico y la Protección al Ambiente en Materia de Evaluación del Impacto Ambiental). The mode and scope of this document are contained in this regulation (Article 5). High-risk activities, must also submit to the authorities a Risk Assessment (Articles 17 and 18). In special cases, the Secretariat could require insurances or financial guarantees in order to assure that the project owner complies with the law and the specific permit conditions (Article 51).

18.4.1.3 General Law of Sustainable Forestry Development (LDFS)

This law was enacted with the idea to promote an environmentally sustainable forestry sector. It relates to the mining sector because, in principle, all land in Mexico is considered to be forestland (LDFS Article 7, Paragraphs XL and XLV). Any activity that requires the use of land other than in urban areas is required to apply for a permit in order to change the use of the land (LDFS, Article 58, Paragraph I).

Such change is approved by SEMARNAT, subject to the previous approval of the Environmental Impact Manifest of a project, and can require a compensatory payment in favour of the Mexican Forestry Fund (LDFS, Articles 117 and 188). Said payment is

determined by an assessment of the environmental services that the land provides multiplied by an estimated fee for reforestation of an equivalent ecosystem.

18.4.1.4 National Waters Law (LAN)

The Mexican Constitution stipulates that all National Waters belong to the Mexican State. The National Waters Law lists the National Waters including main rivers and lakes, and the underground waters as a whole, and regulates the management of these waters. The basic premise is that any intention to use and/or exploit national waters has to be supported by a Concession granted by the competent authority, the National Commission for Water (CONAGUA).

All water discharges must be permitted and must comply with the relevant quality standards. Under Article 124, it is not necessary to get a water concession for water coming from mining works (e.g., water pumping to avoid mine flooding), it is not necessary to get a water concession.

18.4.1.5 General Law for Integral Waste Management and Prevention (LGPGIR)

Solid wastes are a particular problem in Mexico. As such, the Mexican Congress decided to enact a law to control these wastes. This law classifies all waste into three categories: urban wastes, hazardous wastes, and wastes requiring special handling.

Mining and metallurgical activity wastes (tailings, slag, waste rock, etc.) are under an exceptional consideration because such residues can be disposed in situ within the land property of the concessionaire (LGPGIR, Article 17). The law dedicates a special chapter to soil environmental liabilities. This chapter states that all entities responsible for soil contamination are also responsible for its remediation (LGPGIR, Article 68).

All mining and metallurgical activities are required to submit a Waste Management Plan to SEMARNAT for wastes considered to be of high volume and low risk (LGPGIR, Article 33, and LGPGIR Regulations).

The Waste Management Regulations (Reglamento de la Ley General Para la Prevención y Getión Integral de los Residuos) governs high volume wastes from the mining industry. The list of these wastes is specified in Article 32. Mining and metallurgical wastes must be subject to a management plan (Article 33) and their final disposal is regulated by the applicable Mexican standard (Article 34).

18.4.1.6 International Treaties

As a nation, Mexico is a signatory to a significant number of International Environmental Treaties. Additionally, in all the Commercial Treaties to which Mexico is a party, environmental regulations are included.

The International Treaties that could have some relationship with mining activities operating in the country are:

- Vienna Convention for the Protection of the Ozone Layer, Vienna 1986 and Montreal Protocol Related to the Substances that Deplete the Ozone Layer, Montreal 1987.

- Convention on the Control of Transboundary Movement of Hazardous Wastes, Basel 1989.

- UN Convention on Climate Change, New York 1992.

- Convention on Biological Diversity, Rio de Janeiro 1992.

- Convention on International Trade in Endangered Species of Flora and Fauna, Washington 1973.

- North America Free Trade Agreement, side agreement on environmental matters, 1993.

According to the Mexican Constitution, all international treaties signed by the Executive Branch and approved by the Senate are considered binding regulations in Mexico. For practical purposes, the Mexican Congress incorporates the rules and principles of such Treaties in the national legislation.

18.4.1.7 Federal Income Tax Law (ISR)

Income tax constitutes the main source of government taxation funds; hence, the importance of this act that regulates its management. As applicable to environmental issues, it recognizes that taxpayers are entitled to deduct 100% of the amount invested in any environmental protection and control equipment, facilities and civil works. In other words, such investments are not subject to depreciation over extended periods (ISR Article 41, Paragraph XIV).

18.4.1.8 Federal Criminal Code (CPF)

The federal criminal code establishes certain kinds of environmental offences, specifically regarding:

- Responsibility for polluting in general.
- Responsibility for atmospheric pollution.
- Responsibility for water pollution.
- Responsibility for adverse effects on biodiversity.
- Illegal waste management.
- For registering false records.
- Non-compliance with environmental authorities' requirements.

Penalties regarding these offences range from one to nine years in prison and fines from 300 to 3,000 wage days (specifically for the responsible person).

18.4.1.9 Mexican Environmental Standards (Norma Oficial Mexicana, NOM)

Mexican Standards cover all different aspects of economic activity. In general, they specify the parameters and compliance requirements for each activity. Some standards are specific to mining and some general industrial standards also apply to mining. Some of the norms detail specific requirements for sampling and testing contaminants. In particular, the following standards are potentially applicable to the project, some of which have applied to the past mining activity at San Francisco Mine:

- NOM-001-SEMARNAT-1996 Establishes maximum permissible limits for contaminants in discharge waters.
- NOM-043-SEMARNAT-1993 Establishes maximum permissible levels for solid particulates for point source atmospheric emissions.
- NOM-052-SEMARNAT-1993 Establishes the characteristics, a list and the toxicity limits of hazardous wastes.
- NOM-053-SEMARNAT-1993 Establishes procedures to determine the constituents and toxicity of hazardous wastes.
- NOM-085-SEMARNAT-1994 Standards for point source atmospheric emissions from combustion of solid, liquid, and/or gas fossil fuels.
- NOM-120-SEMARNAT-1997 Establishes environmental protection measures for mining exploration in dry climates.

18.4.1.10 San Francisco Mine Environmental Permitting

The San Francisco mine was previously operated by Geomaque de Mexico, S.A. de C.V. under the Instituto Nacional de Ecologia, Direccion General de Normatividad Ambiental permit resolution #551, A.O.O.DGNA 10539 dated November 16, 1994. A Land Use and Construction Licence (#85/95) was issued by the municipality of Santa Ana, Sonora on April 11, 1995. A Change of Land Use authorization was applied for in 1994, but was never issued by the federal government. Geomaque received authorization for use of hazardous materials from the Instituto Nacional de Ecologia, Direccion General de Normatividad Ambiental, D.O.O.DGNA 01425 on March 12, 1996. SEMARNAP issued a Licence #121 for air emissions, DFS-D-0380-97 on April 24, 1997.

Geomaque received authorization to exploit the Chicharra pit on November 10, 1999, D.O.O.DGOEIA 007225, from Insituto Nacional de Ecología Dirección General de Ordenamiento Ecológico e Impacto Ambiental of the Secretaria de Medio Ambiente Recursos Naturales Y Pesca (SEMARNAP).

Geomaque announced to the government on August 12, 2002 that the mine was suspending operations due to metal prices and high operating costs.

The Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT) completed a compliance audit on March 30, 2001 and concluded that Geomaque was not compliant with many of the conditions of its permits including not complying with management requirements for air emissions, hazardous wastes, emergency procedures, and reforestation programs. Geomaque responded to these issues in a report issued July 12, 2001.

Timmins acquired the property in 2007. Timmins, through its subsidiary Molimentales del Noroeste de S.A. de C.V., submitted a site specific environmental impact assessment (Manifesto Impacto Ambiental, Modalidad Particular), and a level 2 Risk Assessment to SEMARNAT on December 12, 2007 in order to obtain the necessary permits to reactivate the mine. It is expected that Timmins will be required to undertake compliance-related activities such as reforestation and groundwater monitoring which were the responsibility of the previous operator.

In addition, Timmins submitted an application for a Change of Land Use permit which was approved by SEMARNAT on February 22, 2008 (Authorization number DFS/SGPA/US/0401/08). This authorization is supported by a compensation payment from Molimentales del Noroeste de S.A. de C.V. that is to be spent on reforestation and restoration activities. This is a key permit for the project.

Timmins plans to apply for to the program (La Auditoría Ambiental) to be certified for environmental management excellence by Procuraduría Federal de Protección al Ambiente (PROFEPA). This program is a proactive approach set up by the government in which companies are subject to less stringent reporting requirements over time as they demonstrate a continued high level of environmental management performance.

18.4.1.11 Permitting Schedule

Permitting is expected to be completed early in 2008 as indicated in the work schedule submitted as part of the MIA to government (see Table 18.8).

Table 18.8
San Francisco Mine Work Schedule

Work Program	2008					
	JAN	FEB	MAR	APR	MAY	JUN
Obtain Authorizations	———	———				
Acquisition of Parts and Equipment	———	———	———			
Maintenance and Refurbishment of Process Equipment		———	———	———		
Exploration Program		———	———	———	———	———
Preparation of Mine and Leach Pad Areas			———	———	———	———
Start of Operations						———

NB: Subsequent to the above submission, the start of operations was revised to September, 2008.

18.4.2 OTHER RELEVANT LEGISLATION AND PROGRAMS

18.4.2.1 Ejido Jesús García Lands

The project lies on Ejido Jesús García lands. An annual land rent is paid to the ejido to temporarily occupy these lands. This agreement #G-2695-2-HMO was dated November 30, 1998.

18.4.2.2 Municipal Development Program 2007-2009

The principal objectives of the Municipal Development Plan of Santa Ana, Sonora, are to improve the quality of life, to be more self sufficient, to reduce unemployment, and make it more attractive to visitors. In addition, as part of the economic development, the plan is to revitalize ranching, encourage self employment and tourism enterprises, and boost regional projects.

18.4.2.3 State Development Plan

Among other things, the hope is for a regional development plan that is sustainable, equitable, diverse, and supportable, a product that incorporates knowledge and technology. It needs to include all economic, social, and political aspects and conform to the Sonoran diversity and plurality. The plan considers the diverse elements that have a bearing on regional development in growth areas determined to have potential for economic development. A strategy is then developed to help generate capital, public resources and incorporate developments into the surrounding population and communities to achieve the desirable socio-economic level.

The state plan defines Santa Ana as a municipality with a dynamic economy, with potential to improve the quality of life in the area by integrating its industries and local competitive advantage with its population growth.

18.4.2.4 Ecological and Protected Areas Plans

It was confirmed that the project area is not considered to be a zone of interest for biological conservation. The project area is not located near any areas being considered as protected areas. Similarly, the project area is not in a hydrological priority area by CONABIO in publications to date.

18.4.2.5 National Development Plan 2007-2012

The San Francisco project is compatible with the national development plan. In particular pertaining to the following aspects:

Strategy 5.1 - Support clean, environmentally friendly technologies into the development sector. This includes modernization of technologies and use of clean technologies to reduce

the pressure on natural resources, reduce pollution, and improve the value of economic activities.

Strategy 6.3 - Promote the establishment and enforcement of legal guarantees for sustainable development of economic activities. Environmental legislation needs final revision to empower environmental management with the necessary tools to guarantee a clean environment and protect natural resources. This needs to be coordinated in a manner that includes legislative power and diverse aspects of the society to properly analyze and implement the legal text.

18.4.3 ENVIRONMENTAL AND SOCIAL BASELINE

The environmental and social baseline and impact assessment is summarized in the following sections based on the impact assessment completed for Molimentales del Noroeste, S.A. de C.V. by engineer consultant Jesús Enrique Flores Ruiz.

18.4.3.1 Study Area

The project is located in the north-central area of Sonora State in the municipality of Santa Ana. The project is located 24 km south of Santa Ana, near to Estación Llano at 30°20’58” North and 11°10’15”West. The nearest important communities include Magdalena de Kino, Santa Ana, Benjamín Hill, Querobabi, Trincheras, and Estación Llano. The main four-lane highway (Highway #15) is located just east of the project which connects the project to Hermosillo, Benjamín Hill to the south, and Santa Ana, Magdalena, and Nogales to the north. The surrounding area is connected by a network of small dirt roads.

18.4.3.2 Climate

Temperatures at the project area range between 42 and 44°C. The area is classified as the BWhw(x’) climate zone which is described as dry to very dry.

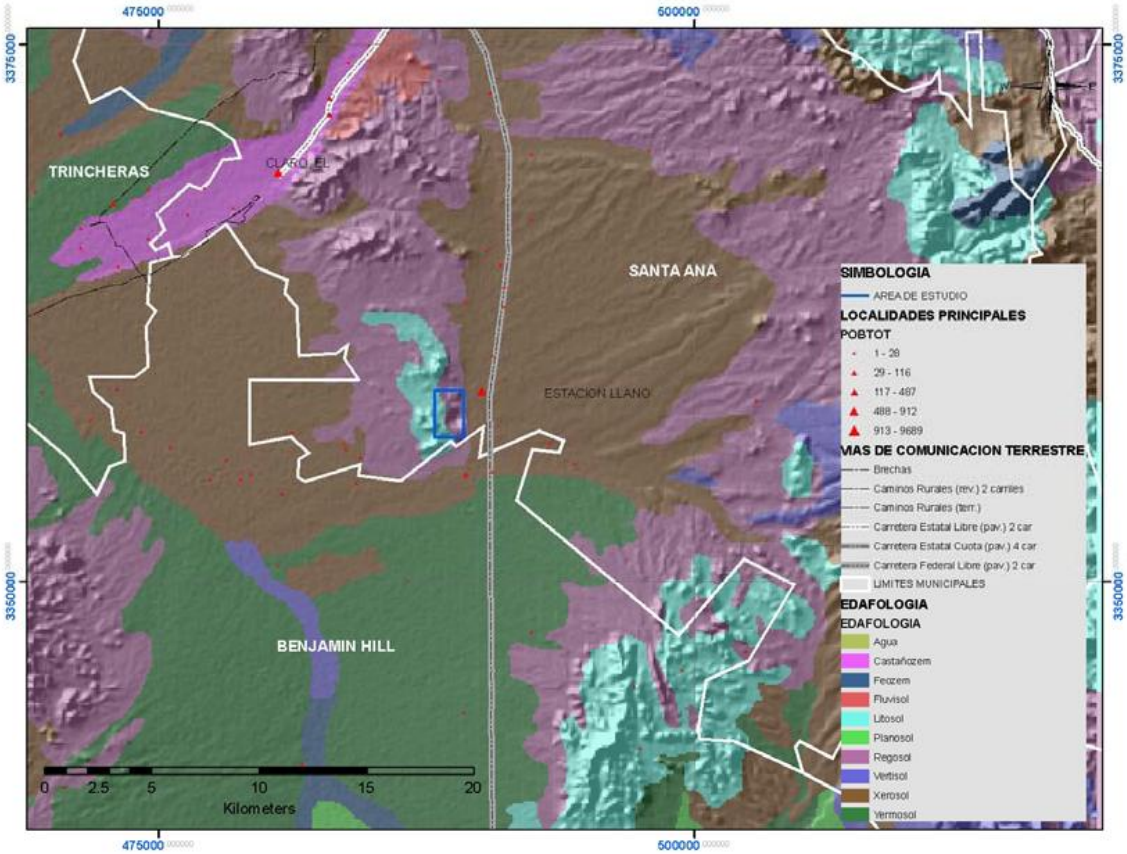
18.4.3.3 Geomorphology

The project area is located in an area that ranges from broad valley plains to foothills just west of the Sierra Madre Occidental mountain range. Elevations range from approximately 700 m to 825 m in the project area.

18.4.3.4 Soils

There are a variety of soil types in the project area, but the four predominant types are regisol, xerosol, yermosol, and litosol. Regisols and lithosols (alluvial, sandy and rocky soils) are found in the higher areas with xerosols and yermosols (clayey, low nutrient soils) found in the lower areas (Figure 18.24).

Figure 18.24
Soils Map



Note: Project area outlined in blue; lithosols in light blue, regosols in purple, xerosols in brown, yermosols in green.

18.4.3.5 Hydrology and Water Use

The project area is located in sub-basins of three main river systems, Río Álamo (blue, Río Alisos (brown), and Río Magdalena (purple) as shown on Figure 18.25. The Río Álamo is the most important which originates in the mountains east of the project and extends to the west to its confluence with the Río Magdalena, 24 km to the west. The Río Magdalena discharges on the coast at Caborca. The Río Alisos originates south of Nogales and also joins with the other two rivers to the west.

Principal aquifers include Magdalena (blue) and La Tinaja (green) as shown on Figure 18.26. Other minor aquifers in the area include Coyotillo, Río Zanjón, and Caborca. The project area is located over the La Tinaja aquifer, which is underexploited according to data from CONAGUA.

The aquifer in the arroyo El Llano watershed to a mean phreatic depth of 30 m in the project area is the property of the Ejido Jesús García, Héroe de Nacozari. The access well is located on the outskirts of the town of Estación Llano.

Figure 18.25
Surface Watersheds

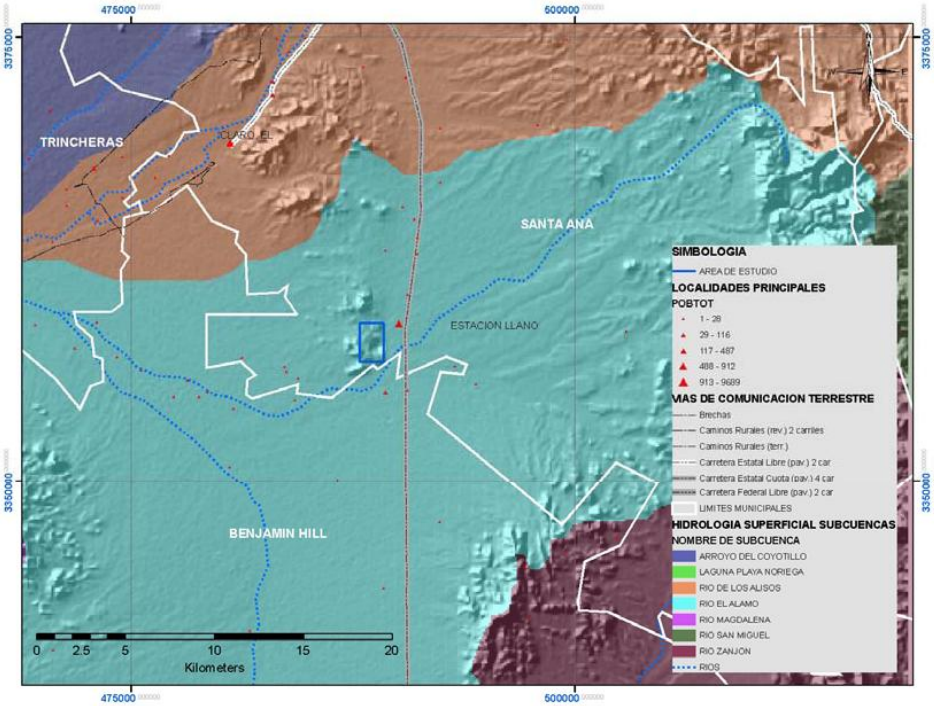
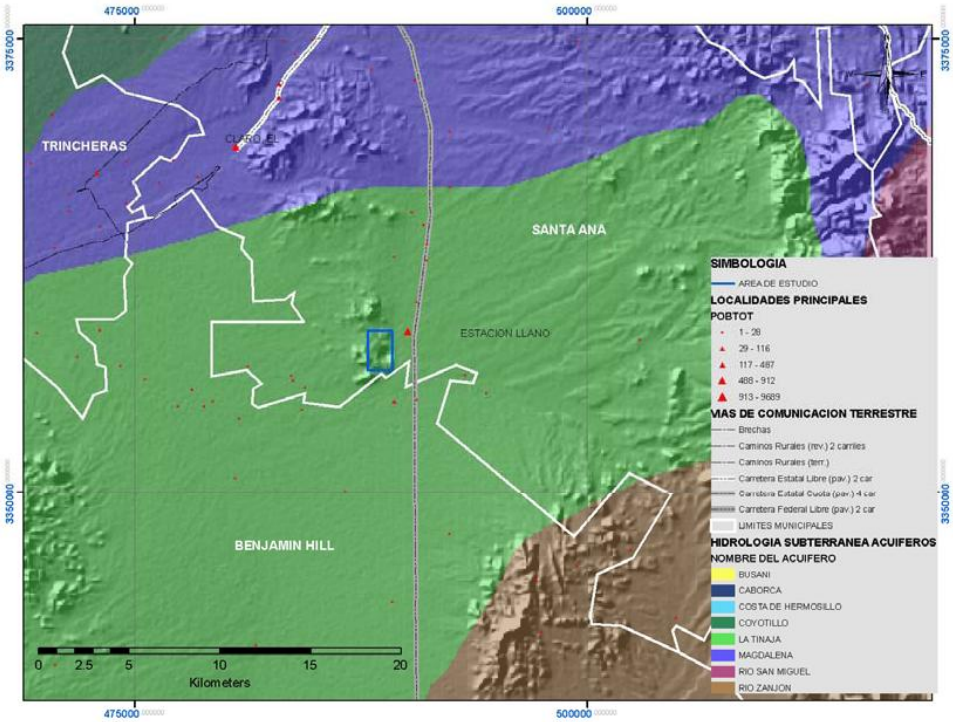


Figure 18.26
Groundwater Divides



18.4.3.6 Water Quality

A set of water quality samples were taken in 2007 to determine the water quality regime due to past mining (Table 18.9). This provides Timmins with a new baseline prior to restarting the mine and helps with impact assessment and mitigation planning. Some residual cyanide is evident in groundwater wells downstream of the leach pad, although the concentrations are below toxic levels. Total dissolved solids are elevated, mostly related to nitrates and sulphates. Iron, zinc, arsenic, selenium, and manganese concentrations are elevated in the groundwater wells. Iron and selenium levels are elevated in the pit lake; however, metals in the wells and pit lake are within effluent discharge standards.

Table 18.9
2007 Water Quality

Parameter	Unit	Pit Water	Groundwater Well at Plant Site	Groundwater Well #1
		Oct.12/07	Oct.12/07	Oct.12/07
pH	Std. units	7.90	7.26	7.73
Temperature	^{o}C	24.0	24.0	27.0
Total Suspended Solids (TSS)	mg/L	2.2	27.8	<1.5
Total Dissolved Solids	mg/L	1,490	1,540	625
Cyanide, total (CN)	mg/L	<0.005	0.176	0.022
Cyanide, free	mg/L	<0.02	<0.02	<0.02
Cyanide, WAD	mg/L	<0.005	0.019	0.009
Nitrate N as NO_3	mg/L	47.6	97.4	36.6
Nitrite N as NO_2	mg/L	0.058	0.017	0.005
Total Kjeldahl N	mg/L	<1	<1	<1
Total P	mg/L	<0.48	0.6	<0.48
Residual Free Chlorine	mg/L	<0.2	<0.2	<0.2
Fluoride (F^-)	mg/L	3.0	0.9	0.7
Biological Oxygen Demand (BOD)	mg/L	<2	2	<2
Chemical Oxygen Demand (COD)	mg/L	<15	<15	<15
Fecal Coliform	mg/L	11.0	49.0	2.0
Chlorine (Cl_2)	mg/L	136	190	49
Sulphur (S)	mg/L	3.6	4.1	3.8
Sulphate (SO_4^{2-})	mg/L	460	150	18
Oil and Grease	mg/L	<7	<7	<7
Phenols	mg/L	<0.1	<0.1	0.1
Total				
Aluminium (Al)	mg/L	<0.050	1.07	<0.050
Arsenic (As)	mg/L	0.005	0.015	<0.005
Barium (Ba)	mg/L	0.057	0.167	0.513
Boron (B)	mg/L	0.267	0.269	0.212
Cadmium (Cd)	mg/L	<0.0025	<0.0025	<0.0025
Chromium (Cr^{+6})	mg/L	<0.10	<0.10	<0.10

Parameter	Unit	Pit Water	Groundwater Well at Plant Site	Groundwater Well #1
Chromium, total (Cr)	mg/L	<0.005	<0.005	<0.005
Cobalt (Co)	mg/L	<0.010	0.138	0.030
Copper (Cu)	mg/L	<0.020	<0.020	<0.020
Iron (Fe)	mg/L	0.011	0.672	0.015
Lead (Pb)	mg/L	<0.010	<0.010	<0.010
Manganese (Mn)	mg/L	<0.010	0.092	<0.010
Mercury (Hg)	mg/L	<0.0003	<0.0003	<0.0003
Nickel (Ni)	mg/L	<0.010	<0.010	<0.010
Selenium (Se)	mg/L	0.148	0.050	0.046
Silver (Ag)	mg/L	<0.0025	<0.0025	<0.0025
Tin (Sn)	mg/L	<0.050	<0.050	<0.050
Zinc (Zn)	mg/L	<0.010	0.012	0.054
Dissolved				
Aluminum (Al)	mg/L	<0.050	1.10	<0.050
Arsenic (As)	mg/L	<0.005	0.014	0.006
Barium (Ba)	mg/L	0.055	0.171	0.487
Boron (B)	mg/L	0.231	0.212	0.138
Cadmium (Cd)	mg/L	<0.0025	<0.0025	<0.0025
Chromium (Cr)	mg/L	<0.005	<0.005	<0.005
Cobalt (Co)	mg/L	<0.010	0.143	0.029
Copper (Cu)	mg/L	<0.020	<0.020	<0.020
Iron (Fe)	mg/L	0.028	0.668	0.017
Lead (Pb)	mg/L	<0.010	<0.010	<0.010
Manganese (Mn)	mg/L	<0.010	0.089	<0.010

18.4.3.7 Soil Use and Vegetation

The zone is characterized by its principal vegetation type, desert scrubland, which can be found in more than 66% of the region (Figure 18.27). However, other localized groundcover occurs. Mesquite covers around 14% of the region in the lowlands and hills next to rivers and central portions of arroyos. Pasture lands are found dispersed in the region. In the high areas, pastureland covers a bit more than 11% of the land. Agriculture lands are generally found along the margins and lowlands of the river valleys. Forests can be found, and are generally associated with higher, cooler areas east of the project.

In the project area, much of the vegetation has been removed by mining activity (Figure 18.28). In addition, the surrounding area has been affected by development in the nearby towns, and intensive farming. Therefore, the surrounding vegetation and ecosystem has converted to secondary grown and is inhabited by invasive species.

Figure 18.27
Typical Desert Scrubland Vegetation

Figure 18.28
Vegetation Study Areas

18.4.3.8 Fauna

There is little abundance or diversity of wildlife species because of anthropogenic activities in the area such as mining, urbanization, agriculture, ranching, and the nearby highway. There is also little food and water because of the competition with the surrounding livestock of cows, pigs, horses, and mules.

Wildlife species observed (or evidence of) in the area includes hare (*Lepus alleni*), rabbit (*Sylvilagus sp*.), squirrel (*Citellus sp*.), skunk (*Mephitis sp*.), and coyote (*Canis latrans*). Birds include quail (*Callipepla gambelli*), white-winged dove (*Zenaida asiatica*), and inca dove (*Columbina inca*). Reptiles include snakes (*Crotalus sp*.), various lizards (*Sceloporus sp*. and *Phrynosoma sp*.).

18.4.3.9 Social Environment

The project lies in the municipality of Santa Ana (2005 population of 14,638; source INEGI census statistics, www.inegi.gob.mx). Estación Llano is the closest town (2005 population 1,056) and is the village for the Ejido Jesús García. The next largest community north is Magdalena with a 2005 population of 25,500, and to the south, Benjamín Hill has a 2005 population of 5,285.

Estación Llano is located immediately adjacent to the San Francisco project. The town has 243 households (2005 statistic); 98% have electricity; 98% are connected to the water distribution network with 77% with running water inside the house. Five percent of households have computers.

Santa Ana and Magdelena are considered to be the nearest communities where mine employees will come from or move to.

In Sonora state in 2004, 41% of the population was economically active. In 2007, the national unemployment rate ranged between 3.23 and 4.02%. Based on historical statistics, the employment rate in Santa Ana is similarly high, above 90%. Santa Ana has a population typically employed in the service sector.

Potable water in Santa Ana is sourced from wells to satisfy the population demands. Of the 3,712 households (2005 statistics), 90% have running water within the house and 5% outside of the house, 92% of households are connected to the town sewer system, 98.5% of households have electricity and 19% of households have a computer.

Santa Ana is located at an intersection of communication routes that make it a service centre, although, farming is considered to be of significant importance to the way of life of the municipality. Potential human resources are important. The male component of the human resources can generally be channelled easily to industries and mining, although there is more permanence and stability if female human resources can also be channelled into these sectors.

The social stability conditions in the municipality are very good, as exists in the majority of the rest of the state. The majority of the population consists of workers with a desire for progress.

18.4.4 ENVIRONMENTAL AND SOCIAL IMPACTS AND MITIGATION

18.4.4.1 Surface and Ground Water Management

The project is designed to avoid discharges to the environment. All solutions will be contained within the leach pads, solution ponds, collection ditches, and recovery plant. The facilities will have impermeable liners and have the capacity to contain storm events.

Diversions will be used to separate clean rain water from other solution waters, grey waters and sewage.

18.4.4.2 Atmospheric Emissions Management

Atmospheric emissions will be minimized by regular equipment maintenance program. This will help minimize emissions due to fuel combustion.

Dust and noise may be generated from explosives use in the pit, although these effects are localized and can be minimized with regular controlled blasting.

The main source of dust emissions is expected from the access roads principally during the dry season. This will be minimized with road watering as necessary. Dust is also expected from ore crushing and haulage. If this is seen as causing health problems, mist emitters could be considered to help trap dust.

18.4.4.3 Soil Management

Un-mineralized soils will be deposited in a location near the pit. Water runoff will be controlled to reduce erosion. Throughout site preparation and operations, measures will be taken to reduce erosion and help conserve the soil structure.

18.4.4.4 Waste Management

Cyanide-bearing solutions will be recycled within the process and will not be discharged.

All other hazardous wastes will be managed using best management practices and follow legislated standards. For hazardous wastes such as used oil, etc. will be stored at an on-site temporary storage facility prior to collection by an accredited company and disposal in an appropriate off-site regulated facility. Hazardous waste management will follow the necessary protocols for tracking materials and employee training.

Non-hazardous wastes will be transferred to an approved landfill site. Containers will be washed to remove any chemical residues prior to landfill disposal.

Fuel and lubricant storage will be in tanks within a synthetically lined bermed area with a concrete base designed with a capacity to contain the volumes of the tanks.

Maintenance sheds will have grease traps and oil sumps to recover all used oil and grease for proper disposal.

The following measures will be implemented at various stages of the operation:

- Fill will be obtained from planned cuts and excavations to minimize the requirements for new fill. All excavated material will be used as fill.
- Portable toilets will be used in all work areas to prevent smells and illnesses.
- Construction equipment and waste materials will be removed from watershed basins to prevent water contamination.
- Specific areas will be set up for equipment storage.
- Emergency response and spill contingency plans will be implemented including response equipment (e.g. spill kits) and fire fighting equipment.

18.4.4.5 Flora

Closure and restoration of vegetative cover will be considered during the clearing and mine planning. The area must be restored to a reasonable level for future use considering the current use. Vegetative cover must be completed to reduce erosion. Disturbance of native vegetation and already established secondary growth should be minimized. Clearing will be strictly controlled on site to only necessary areas.

Cleared areas will be revegetated with plants rescued from other cleared areas. Species with high aesthetic value will be chosen such as cactus, mesquite (*Prosoppis laevigata*), palo verde (*Parkinsonia aculeate*), elephant tree (*Pachycormus discolor*), and shrubs such as limberbush (*Jatropha sp*.), and greasewood (*Larrea tridentata*). Native species with high tolerance for arid conditions will be chosen to ensure the vegetative cover is self sustaining. It is recommended that a greenhouse be constructed for conserving species and growing cuttings and seedlings to add to the success of the reclamation program.

Transplants will use appropriate methods to maximize success. Mechanized shovels will be used for species over two metres, and bushes and trees less than two metres will be manually transplanted.

18.4.4.6 Fauna

Wildlife impacts will be minimized by establishing strict no hunting policies in the area of the project. The area will be enclosed by a three barbed wire fence that allows passage of small animals. Four inch plastic balls will be placed on the leach collection ponds to prevent birds from using the ponds. Nets will be placed over collection ditches containing hazardous solutions.

18.4.4.7 Closure and Reclamation

Following closure, the area will be reclaimed and monitored over a period of two years. Activities will follow the legislated and permitted requirements.

18.4.4.8 Visual Effects

Visual effects will be minimized at the end of mining through placement of soils and revegetating all areas with native species.

18.4.4.9 Social Impacts

The potential mine is very important for the community. It is expected to result in increased investment in the area, and will generate direct and indirect employment. It is expected to have a significant impact since it will provide a greater diversity of employment alternatives that will advance the diversity of the local economy that will also attract regional development.

18.4.4.10 Residual Effects and Significance of Effects

A Leopold Matrix approach was taken for identifying project interactions and effects in the December, 2007 impact assessment. Those effects were then assessed for significance based on an evaluation of the extent, magnitude, duration, frequency, cumulative effects, reversibility, and social and economic value.

Based on the evaluation, there are expected to be moderately significant adverse effects on water flows and quality, and vegetation species of interest during the initial site preparation phase. Moderately significant positive effects are expected for the communities due to the direct and indirect employment and benefits that are expected to be realized during the construction and operation stages.

18.4.5 ENVIRONMENTAL AND SOCIAL COSTS

Environmental and social operations cost estimates include:

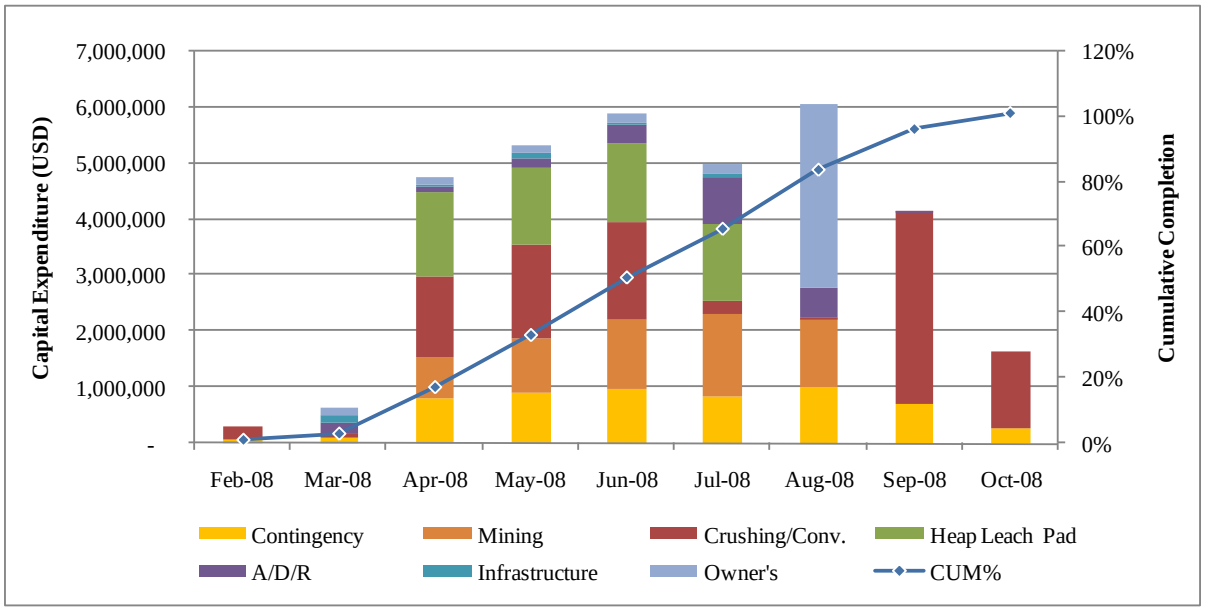
- Three environmental staff.

- Outside consultants.
- Laboratory costs.
- Environmental audit costs.
- Temporary occupation rent for Ejido Jesús García lands.
- Reclamation bonding required by government to cover revegetation costs of disturbed land.
- Training costs.

18.5 PROJECT IMPLEMENTATION SCHEDULE

The project construction and implementation schedule, represented by capital expenditure, is shown in Figure 18.29.

Figure 18.29
Project Construction Schedule



The project implementation schedule calls for the completion of heap leach pad construction by mid-2008, in order that loading of the pad can commence in the third quarter, and leaching can commence at the end of that quarter. Refurbishment of the crushing plant must therefore be completed around mid-year. Expenditure shown later than this in the schedule represents at deferred payment in respect of the work carried out on the secondary crushing and conveying systems by Sandvik. The pumping, irrigation and ADR plant must also be commissioned during the third quarter.

Such a timetable for development is ambitious, though Micon notes that refurbishment of the ADR plant was well advanced at the time of its visit in 2007. Critical path items are seen to be pre-stripping of the open pit and construction of the leach pad and irrigation/ADR systems. While the crushing system is important to the achievement of planned levels of recovery, it would be possible to commission the leach pad with uncrushed ROM ore if the crushing system fell behind schedule.

Slippage of the project schedule by up to three months is unlikely to have a material impact on the returns generated, as such slippage will most likely be accompanied by a deferral of some capital expenditure into later accounting periods.

18.6 CAPITAL EXPENDITURES

18.6.1 BASIS OF ESTIMATE

The estimate of capital expenditure has been prepared on the basis of supplier quotes for:

- Mining pre-strip and operation (Grupo Peal).
- Heap leach pad construction and liner installation (Construplan) based on drawings prepared by Golder Associates.
- Refurbishment of the primary crushing circuit (Metso).
- Refurbishment and replacement of the secondary/tertiary crushing, screening and conveying/stacking plant, based on suppliers own drawings (Sandvik).
- Refurbishment of the ADR circuits (local contractors).

Other items have been based on supplier-offered rates and/or local staff experience.

The base date for the estimate is December, 2007.

18.6.2 PROJECT CAPITAL COSTS ESTIMATE

The overall initial capital estimate for the project is given in Table 18.10.

Table 18.10
Initial Capital Cost Estimate

	USD 000
Mining	5,675
Primary Crusher Rehabilitation	759
Secondary Crushing	9,483
Leach Pad Construction	5,641
Adsorption Plant Rehabilitation	2,171
Infrastructure	406
Indirect Costs	4,006
Contingency	5,628
Total Initial Capital	**33,769**

18.6.3 MINING CAPITAL COSTS

Table 18.11 shows a breakdown of the mining capital expenditure.

Table 18.11
Mining Capital Cost Estimate

	USD 000
Mobilization	750
Pre-stripping	4,925
Total Mining Initial Capital	**5,675**

Mining capital costs represent the principal mining contractor establishment charges, which are needed in order to refurbish and equip the existing workshop structure. Pre-production mining charges are included for pre-stripping of waste rock.

Light vehicles and equipment for the owner's supervisory and technical services teams are included under infrastructural capital, below.

18.6.4 PROCESS PLANT CAPITAL COSTS

A breakdown of the initial process plant capital estimate is given in Table 18.12.

Metso has provided a quotation for refurbishment of the primary crusher, while Sandvik has provided design drawings and a quotation for the refurbishment of existing and supply of additional equipment for the secondary and tertiary crushing, screening and ore stacking conveyors.

Golder Associates has prepared a design for the construction of the heap leach pad, an additional emergency pond and a launder to carry pregnant solution to the existing solution ponds. This pad, to be constructed in two phases, will occupy a total area of 47.8 ha. Only the first phase of the pad (approximately 25 ha) is included in initial capital. Phase 2 is provided for as ongoing capital.

Earthmoving and geomembrane liner costs have been estimated on the basis quantities taken from this design, and unit rates from contractor Grupo Construcciones S.A de C.V. (Construplan).

Costs for rehabilitation of the existing ADR section are based on local contractor cost estimates.

Table 18.12
Initial Capital – Process Plant

	USD 000
Primary Crusher Rehabilitation	**759**
Primary Crusher Rehab (Metso)	444
Replacement wear parts	276
Belt #1	39
Secondary/Tertiary Crusher Rehabilitation	**9,483**
Sandvik turn-key	4,532
Sandvik equipment	2,664
Sandvik spares	260
Reclaim tunnel	70
Overland conveyor #1	255
Overland conveyor #2	245
Fines tunnel	67
Grasshopper	1,087
Support Equip	288
Lime silo & addition	15
Leach Pad Construction	**5,641**
Design	94
Earthworks and line installation	4,997
Liner protection layer and drainage pipes	350
Emergency ppond excavation & lining	200
Adsorption Desorption Refining (ADR)	**2,171**
ADR Plant Rehabilitation	2,171
Total Processing Initial Capital	**18,054**

18.6.5 INFRASTRUCTURE CAPITAL COSTS

Initial infrastructural capital requirements are shown in Table 18.13.

Table 18.13
Initial Capital – Infrastructure

	USD 000
Pickup trucks (4)	112
Minibus (2)	76
Admin Vehicles (2)	36
Laboratory Equipment	110
Offices & ICT	72
Total Initial Capital Infrastructure	**406**

Light vehicles comprise four 4WD pickup trucks to service the open pit, workshps and leach pads, two minibuses for staff transport to/from Santa Ana, and two administrative/managers' vehicles.

Existing utility connections to the electrical and water supply infrastructure are understood to be adequate for the resumption of production at San Francisco at the anticpated rates and, therefore, no additional capital has been provided for this purpose.

Information and communications technology (ICT) required for the project comprises accounting, stores and maintenance, and technical software systems.

18.6.6 INDIRECT CAPITAL COSTS

Indirect initial capital costs are shown in Table 18.14.

Table 18.14
Indirect Capital Costs

	USD 000
Change of land use permits	200
Owner's costs	3,118
Pre-production salaries	673
Other costs	15
Change of land use	200
Indirect Initial Capital Costs	**4,006**

Provision has been made in the estimate of Indirect initial capital costs for Timmins staff salaries and overheads in the pre-production period, as well as environmental monitoring and the cost of permits for Change of Land Use.

It is assumed that all IVA (or VAT) will be recouped, and is therefore not provided for in the cost estimate.

18.6.7 CONTINGENCY ON INITIAL CAPITAL

A contingency of 20%, amounting to a total of USD 5.628 million has been provided in the cash flow model. This amount is commensurate with the expected level of accuracy of the cost estimates in the preliminary feasibility study.

18.6.8 SUSTAINING CAPITAL EXPENDITURE

Sustaining capital expenditure is given in Table 18.15.

Costs for demobilization of the mining contractor have been set at $1.20 million. On mine closure, redundancy costs have been provided for at 6 months salary costs, in line with accepted norms for Mexico.

Phase 2 of the leach pad construction has been provided for at a unit rate 10% higher than that in Phase 1, to allow for increased cut and fill requirements. Additional pumps, pipes, valves and dripper lines are also allowed for.

Table 18.15
Life-of-Mine Sustaining Capital Expenditure

	USD 000
Mining (demobilization)	1,200
Crushing/Screening/Conveying	2,048
Leach Pad Ph 2 - pad construction	4,840
Leach Pad Ph 2 - solution distr./recov.	440
ADR plant	434
Infrastructure	81
Closure/ Redundancies	3,616
Total Sustaining Capital	**12,659**

A provision has been made for unspecified ongoing capital replacement expenditure in the crushing, screening and conveying plant, to cover any additional heap construction conveying equipment needs and/or replacement of existing equipment additional to the working cost allowance for maintenance. This allowance has been set at 4% per year of the initial capital cost for this plant.

18.6.9 SCHEDULE OF CAPITAL EXPENDITURE

Table 18.16 shows the annual capital expenditure schedule.

Table 18.16
Annual Capital Expenditure Schedule
(USD 000)

	2008	2009	2010	2011	2012	2013
Mining	5,675					
Primary Crusher Rehabilitation	759					
Secondary crushing and screening	9,483					
Leach Pad Construction	5,641					
Adsorption Plant Rehabilitation	2,171					
Infrastructure	406					
Indirect Costs	4,006					
Contingency	6,163	-				
Mining (demobilization)						1,200
Crushing/Screening/Conveying		410	410	410	410	410
Leach Pad - Phase 2			4,840			
Ph 2 - soln distrib/collection			440			
Adsorption plant, etc		87	87	87	87	87
Infrastructure		16	16	16	16	16
Closure/Redundancies						3,231
Total Annual Capital Costs	**33,769**	**513**	**5,793**	**513**	**513**	**4,944**

18.7 OPERATING COSTS

18.7.1 BASIS OF ESTIMATE

Operating cost estimates have been prepared from a zero base, using unit costs for labour, utilities and reagents and consumables based on local market conditions. Labour costs include appropriate provision for salary burden.

18.7.2 PROJECT OPERATING COSTS

Table 18.17 shows a summary of the project operating costs over the life of the mine, and as a cost per tonne of ore heaped, and per ounce of gold sold. Note that only high grade ore is crushed before it is is placed on the leach pad. Lower grade ore is heaped uncrushed (i.e., as ROM ore), and so the costs applicable to each are shown separately in the table.

Table 18.17
Cash Operating Costs

	LOM Total (USD 000)	High Grade Ore (USD/t)	Low Grade Ore (USD/t)	LOM Average (USD/t)	LOM Average (USD/oz Au)
G&A	6,821	0.30	0.30	0.30	16.84
Mining	113,070	4.97	5.06	5.00	279.08
Crushing	22,161	1.33	-	0.98	54.70
Processing	19,736	0.87	0.87	0.87	48.71
Laboratory	3,686	0.16	0.16	0.16	9.10
Social & Envir. Mgmt	1,677	0.07	0.07	0.07	4.14
Total	**167,020**	**7.71**	**6.47**	**7.38**	**412.57**

18.7.3 MINING OPERATING COSTS

Mining operating costs are based on a proposal received from Peal, which covers the provision of drill, blast, load and haul services to the mine. A base cost of USD 1.56/t, subject to rise and fall with input costs, is applied to the first 2,300 m haul distance. Thereafter, an overhaul charge of USD 0.25/t-km is applied.

In addition to contractor costs, provision has been made for supervisory and technical services labour costs, and for power supply to the workshops and explosives magazine.

The proposed mining schedule gives rise to the tonnages and operating costs shown in Table 18.18. An amount of USD 4.925 million relating to pre-stripping of waste rock prior to the start of production in the first year is taken to initial mining capital.

Table 18.18
Mining Production and Operating Cost Schedule

	Unit	2008	2009	2010	2011	2012	2013	Total
Waste	000 t	4,804	12,647	10,733	8,323	6,045	3,507	46,059
High Grade Ore	000 t	464	3,300	3,300	3,300	3,300	2,989	16,653
Low Grade Ore	000 t	447	1,784	1,789	1,367	594	-	5,981
Waste Rock	USD/t	1.56	1.60	1.73	1.86	1.86	1.96	1.73
Crusher Feed	USD/t	1.56	1.56	1.56	1.56	1.56	1.61	1.57
ROM ore	USD/t	1.56	1.57	1.70	1.75	1.75	1.75	1.66
Waste Rock	USD 000	7,494	20,286	18,536	15,464	11,232	6,867	79,878
Crusher Feed	USD 000	724	5,148	5,148	5,148	5,148	4,803	26,119
ROM Ore	USD 000	697	2,794	3,045	2,385	1,037	-	9,958
Power	USD 000	13	39	39	39	39	30	198
Labour	USD 000	123	360	360	360	360	278	1,841
Less Pre-production cost	USD 000	(4,925)						(4,925)
Total	USD 000	**4,127**	**28,626**	**27,127**	**23,396**	**17,815**	**11,978**	**113,070**
Average	**USD/t**	**4.53**	**5.63**	**5.33**	**5.01**	**4.57**	**4.01**	**5.00**

18.7.4 PROCESS OPERATING COSTS

Process operating costs include crushing and screening costs for high grade ore, as well as heap leaching, ADR costs for high grade and low grade ore, and laboratory costs. All associated power and labour cost are included. Table 18.19 shows the costs for each of these areas.

In the opinion of Micon, Timmins has appropriately estimated labour requirements. Depending on the experience of the labour recruited, the total complement may prove to be somewhat in excess of the number of people actually required. However, this is a prudent assumption at the start of the operation, and will allow natural attrition to cater for any possible overstaffing which may become evident as the operation matures.

The crushing, leaching, laboratory and refining cost appears slightly low when compared with other operations, The overall cost is, however, within the range experienced by comparable operations.

Table 18.19
Process Operating Costs
(USD 000)

	2009	2010	2011	2012	2013	2014	Total
Ore Crushing costs							
Manganese steel	205	1,460	1,460	1,460	1,460	1,323	7,369
Screen and conveyors	47	334	334	334	334	302	1,684
Power	188	1,340	1,340	1,340	1,340	1,213	6,761
Labour	426	1,241	1,241	1,241	1,241	957	6,347
Sub-total	*866*	*4,375*	*4,375*	*4,375*	*4,375*	*3,796*	*22,161*
Heap Leach/ADR							
Sodium Cyanide	364	2,034	2,036	1,867	1,558	1,196	9,054
Lime	46	254	254	233	195	149	1,132
Caustic Soda	9	51	51	47	39	30	227
Hydrochloric Acid	4	22	22	20	17	13	99
Activated Carbon	24	132	132	121	101	78	588
Antiscalant	34	187	188	172	144	110	835
Propane	18	102	102	93	78	60	453
Borax	0.5	3.0	3.0	2.8	2.3	1.8	13
Silica	0.7	3.9	3.9	3.6	3.0	2.3	18
Sodium Carbonate	0.1	0.4	0.4	0.3	0.3	0.2	2
Potassium Nitrate	0.1	0.3	0.3	0.3	0.3	0.2	2
Fluorite	0.1	0.7	0.7	0.7	0.6	0.4	3
Water	94	523	523	480	400	307	2,327
Power	89	495	496	455	379	291	2,205
Labour	186	544	544	544	544	419	2,780
Sub-total	*869*	*4,352*	*4,356*	*4,040*	*3,461*	*2,659*	*19,736*
Laboratory							
Reagents & Fuel	7	40	40	37	31	24	180
Assay Consumables	57	316	316	290	242	186	1,408
Wear & Maint. Parts	2	10	10	9	8	6	45
Supplies and Services	0.6	3	3	3	2	2	14
Power	12	43	43	43	43	43	225
Labour	122	355	355	355	355	274	1,816
Sub-total	*200*	*767*	*768*	*737*	*681*	*534*	*3,686*
Grand Total	**1,935**	**9,494**	**9,498**	**9,152**	**8,516**	**6,988**	**45,583**
Average USD/t	**2.12**	**1.87**	**1.87**	**1.96**	**2.19**	**2.34**	**2.01**

18.7.5 GENERAL AND ADMINISTRATION OPERATING COSTS

General and administration operating costs are given in Table 18.20. In addition to management supervision and office running costs, these include provisions for reclamation bonding and annual occupancy (rental) payments to the local ejido community. Payments for the maintenance of mining title are also included here.

Table 18.20
General and Administrative Costs
(USD 000)

	2009	2010	2011	2012	2013	2014	Total
Labour	251	732	732	732	732	564	3,742
Power	8	46	46	42	35	27	205
Accounting (excl. labour)	14	40	40	40	40	31	205
Safety (excl. labour)	14	40	40	40	40	31	205
Security (excl. labour)	22	64	64	64	64	49	327
Janitorial services	3	10	10	10	10	8	51
Office Supplies, Postage	5	15	15	15	15	12	77
Maintenance Supplies	2	5	5	5	5	4	26
Telephone	6	18	18	18	18	14	92
Licenses, Fees, etc	2	5	5	5	5	4	26
Claims Assessment	26	75	75	75	75	58	384
Legal	21	60	60	60	60	46	307
Insurances	43	125	125	125	125	96	639
Subs, Dues, PR, and Donations	9	25	25	25	25	19	128
Travel, Lodging, and Meals	10	30	30	30	30	23	153
Reclamation Bonding	9	25	25	25	25	19	128
Training	9	25	25	25	25	19	128
Total	**452**	**1,340**	**1,340**	**1,336**	**1,329**	**1,025**	**6,821**
Average USD/t	**0.50**	**0.26**	**0.26**	**0.29**	**0.34**	**0.34**	**0.30**

18.7.6 SOCIAL AND ENVIRONMENTAL MANAGEMENT OPERATING COSTS

Social and environmental management operating costs are shown in Table 18.21. Included in this are costs for external consultants, laboratories and contractors assisting with the ongoing monitoring program, and a provision for annual audit.

Table 18.21
Social and Environmental Management Operating Costs
(USD 000)

	2009	2010	2011	2012	2013	2014	Total
Labour (incl. in G&A)							
Outside Services	10	30	30	30	30	23	153
Outside Analytical	26	75	75	75	75	58	384
Outside Consultants	2	5	5	5	5	4	26
Outside Contractors	2	5	5	5	5	4	26
Audit	7	20	20	20	20	15	102
Supplies & Services	0	1	1	1	1	1	5
Ejido land - rental	62	180	180	180	180	139	921
Training costs	4	12	12	12	12	9	61
Total	**112**	**328**	**328**	**328**	**328**	**253**	**1,677**
Average USD/t	**0.16**	**0.06**	**0.06**	**0.06**	**0.06**	**0.05**	**0.06**

18.7.7 SCHEDULE OF OPERATING COSTS

Table 18.22 shows the annual cash operating costs for the project.

Table 18.22
Schedule of Annual Operating Costs
(USD 000)

	2009	2010	2011	2012	2013	2014	TOTAL
G&A	452	1,340	1,340	1,336	1,329	1,025	6,821
Mining	4,127	28,626	27,127	23,396	17,815	11,978	113,070
Crushing	866	4,375	4,375	4,375	4,375	3,796	22,161
Leaching, ADR	869	4,352	4,356	4,040	3,461	2,659	19,736
Laboratory	200	767	768	737	681	534	3,686
Social & Env.Mgmt	112	328	328	328	328	253	1,677
Total	**6,626**	**39,788**	**38,294**	**34,212**	**27,988**	**20,244**	**167,152**
Average USD/t	**7.27**	**7.83**	**7.52**	**7.33**	**7.19**	**6.77**	**7.38**

18.8 ECONOMIC ANALYSIS

18.8.1 MACRO-ECONOMIC ASSUMPTIONS

Economic analysis of the San Francisco Project has been carried out in United States dollars (USD). Conversion of local Mexican costs – principally labour – has been made at the rate of MXN 11.00:USD 1.00, which is approximately the average over the past five years.

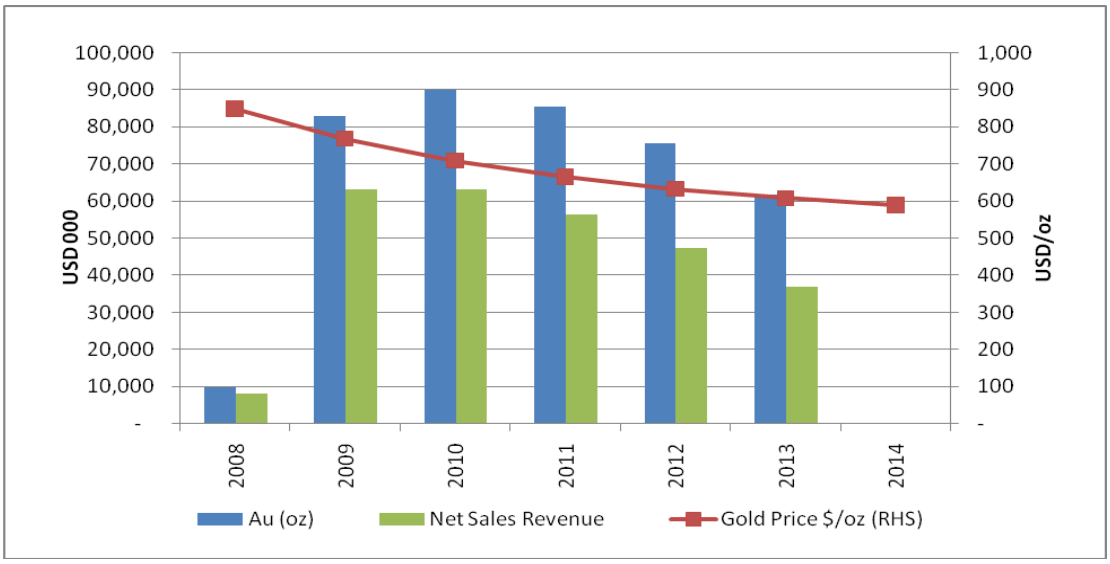
The cash flow projection has been made in constant, first quarter 2008 money terms. Taxable losses carried forward have been discounted at 4% per year to account for inflationary loss in their economic value. The net cash flows have been discounted to March 31, 2008 terms for the purposes of calculating NPV. A base discount rate of 10% per year has been selected as most likely to represent the weighted average cost of capital. Discount rates of 5% and 15% have also been used to test the sensitivity of the cash flow to this parameter with results shown as NPV_5 and $NPV_{15,}$ respectively.

18.8.2 METAL PRICE FORECASTS

Revenue projections are based on a gold price which is forecast to decline in real terms from the 2008 price level, towards a long-term median price. The average price of USD 850/oz for 2008 is taken from the median of analysts forecasts published in early 2008 by the London Bullion Market Association (LBMA). The long-term median price used is USD 522.39/oz, being calculated from annual average prices in real terms for the period 1995 to 2007.

Figure 18.30 shows the declining price curve used to generate the revenue forecasts. For the base case production profile, this results in a weighted average price of USD 686.63/oz, before royalties and refining charges.

Figure 18.30
Gold Production, Price and Revenue Forecast



18.8.3 ROYALTIES & TAXES

As described earlier in this report, any liability for NSR royalty interests in the San Francisco mine are understood not to have been transferred to the present owners. Therefore, no royalty has been provided for.

Mexican corporate taxation has been provided for at the rate of 28%, which Timmins expects to represent the maximum rate applicable. The World Bank/IFC website estimates a typical effective rate of tax on profits in Mexico of 22.4%.

18.8.4 WORKING CAPITAL

Provision has been made for working capital represented by 7 and 30 days of accounts payable and receivable, respectively, as well as 45 days stores and 30 days of product inventory.

18.8.5 BASE CASE CASH FLOW MODEL

A summary of the base annual cash cash flow model is given in Table 18.23. LOM totals for the undiscounted and discounted cash flows are provided in Table 18.24, and the base case cash flow profile is presented graphically in Figure 18.8.

It will be seen from the above that the net present value of the project cash flow at a discount rate of 10%/y (NPV_{10}) for base case evaluates to approximately USD 25.5 million, and that the cash flow has an internal rate of return of 38.5%. The average cash cost of production equates to USD 412.57/oz gold, or USD 7.38/t treated (including leached uncrushed ROM material).

Table 18.23
Summary of Project Base Case Annual Cash Flows
(USD 000)

		2008	2009	2010	2011	2012	2013	TOTAL
Gold Price	USD/oz	850.00	768.78	709.88	666.36	633.72	608.96	686.63
Revenue	Gross Sales (USD 000)	8,495	63,604	63,767	56,891	47,919	37,511	278,188
less	Refining charges	(17)	(145)	(157)	(149)	(132)	(108)	(709)
less	Bullion delivery	(372)	(397)	(397)	(394)	(390)	(385)	(2,334)
	Net Sales Revenue	8,106	63,063	63,213	56,348	47,397	37,018	275,145
Cash op. costs	G&A costs	452	1,340	1,340	1,336	1,329	1,025	6,821
	Mining costs	4,127	28,626	27,127	23,396	17,815	11,978	113,070
	Crushing costs	866	4,375	4,375	4,375	4,375	3,796	22,161
	Leaching, ADR costs	869	4,352	4,356	4,040	3,461	2,659	19,736
	Laboratory costs	200	767	768	737	681	534	3,686
	Social & Environmental Mgmt	112	328	328	328	328	253	1,677
	Total cash operating costs	6,626	39,788	38,294	34,212	27,988	20,244	167,152
Net Cash Operating Margin (EBITDA)		1,480	23,274	24,920	22,136	19,409	16,774	107,994
Capital Expenditure	Initial/expansioncapital	33,769	-	-	-	-	-	33,769
	Sustainingcapital	-	513	5,793	513	513	5,328	12,659
	ChangesinWorkingCapital	1,601	1,716	(118)	(348)	(467)	(2,384)	-
Net cash flow before tax		(33,889)	21,045	19,245	21,972	19,363	13,830	61,565
Taxation payable		-	2,834	5,109	4,278	3,433	1,661	17,313
Net cash flow after tax		**(33,889)**	**18,212**	**14,136**	**17,694**	**15,930**	**12,169**	**44,252**
Discounted Cash Flow (10 %/y, discounted to 3/2008)		(33,091)	16,166	11,408	12,981	10,624	7,378	25,466
Cumulative DCF (10 %/y)		(33,091)	(16,925)	(5,518)	7,464	18,088	25,466	
Payback period on discounted cash flow (years)								3.43
Ave. Revenue per tonne treated		8.90	12.40	12.42	12.07	12.17	12.38	12.16
Ave. Cost per tonne treated		7.27	7.83	7.52	7.33	7.19	6.77	7.38
Operating Margin		18.3%	36.9%	39.4%	39.3%	40.9%	45.3%	39.2%

Table 18.24
Summary of Project Base Case Discounted Cash Flow and Unit Costs
(USD 000)

IRR 38.5%		LOM Total (Undisc)	NPV_5	NPV_{10}	NPV_{15}	USD per tonne	LOM Average (USD/oz)
Revenue	Gross Sales	278,188	241,900	212,654	188,763	12.29	686.63
less	Refining charges	(709)	(613)	(536)	(473)	(0.03)	(1.75)
less	Bullion delivery	(2,334)	(2,047)	(1,818)	(1,632)	(0.10)	(5.76)
	Net Sales Revenue	275,145	239,240	210,300	186,658	12.16	679.12
Cash op.costs	G&A costs	6,821	5,921	5,200	4,614	0.30	16.84
	Mining costs	113,070	99,032	87,640	78,273	5.00	279.08
	Crushing costs	22,161	19,118	16,688	14,720	0.98	54.70
	Processing costs	19,736	17,174	15,111	13,426	0.87	48.71
	Laboratory costs	3,686	3,202	2,814	2,498	0.16	9.10
	Social & Env.Mgt	1,677	1,456	1,278	1,134	0.07	4.14
	Total cash op.costs	167,152	145,903	128,731	114,665	7.38	412.57
Net Cash Operating Margin		107,994	93,337	81,570	71,993	4.77	266.55
Capital Exp.	Initial/exp. capital	33,769	33,359	32,974	32,609	1.49	83.35
	Sustaining capital	12,659	10,651	9,079	7,827	0.56	31.25
Change in Working Capital		-	569	979	1,277	-	-
Net cash flow before tax		61,565	48,757	38,538	30,280	2.72	151.96
Taxation payable		17,313	14,969	13,072	11,518	0.76	42.73
Net cash flow after tax		**44,252**	**33,788**	**25,466**	**18,761**	**1.96**	**109.22**

Figure 18.31 shows graphically the base case cash flow. Using an annual discount rate of 10%, the discounted cash flow has a payback period of 2.5 years, or approximately half the LOM.

18.8.6 SENSITIVITY STUDY

Sensitivity of the NPV_{10} to changes in gold price, operating and capital costs has been analyzed (Figure 18.32). While project revenues are directly proportional to process recovery and grade, project operating costs are largely insensitive to these factors, and hence gold price may also be used as a proxy in this model both for changes in recovery and reserve grade as revenue drivers.

The sensitivity results show that, as expected, the project is most sensitive to the revenue drivers described above. An adverse change of 20% is just sufficient to produce a slightly negative NPV_{10}.

Figure 18.31
Base Case Cash Flow Profile

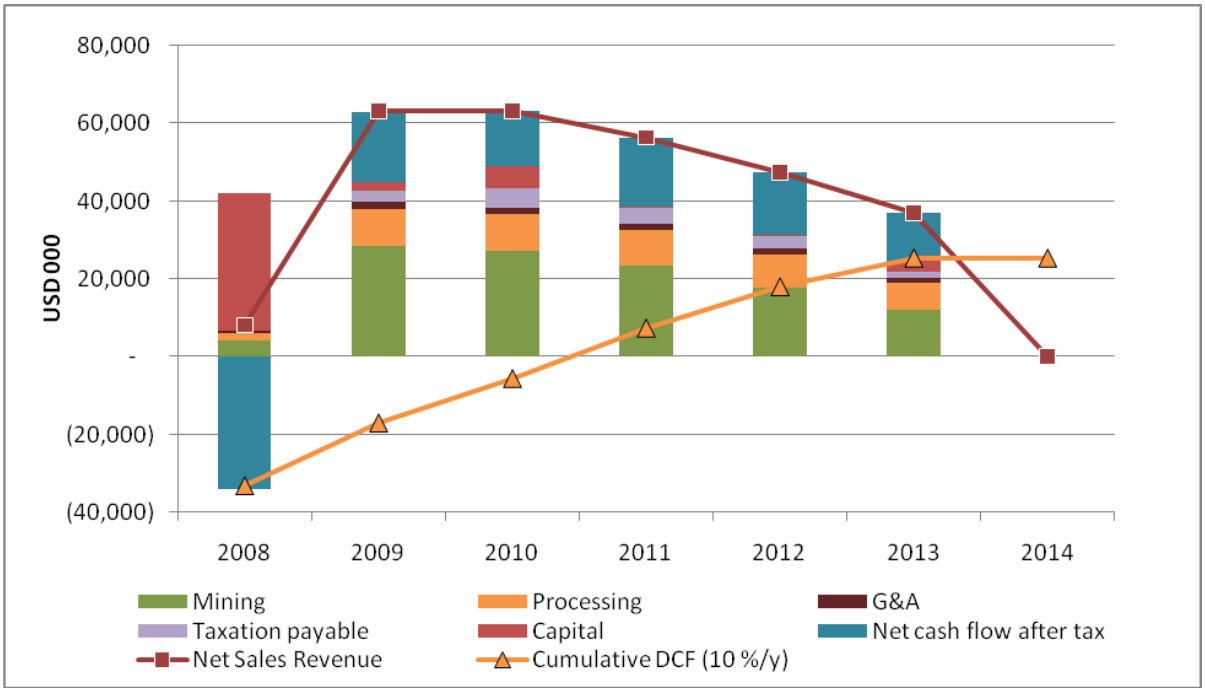
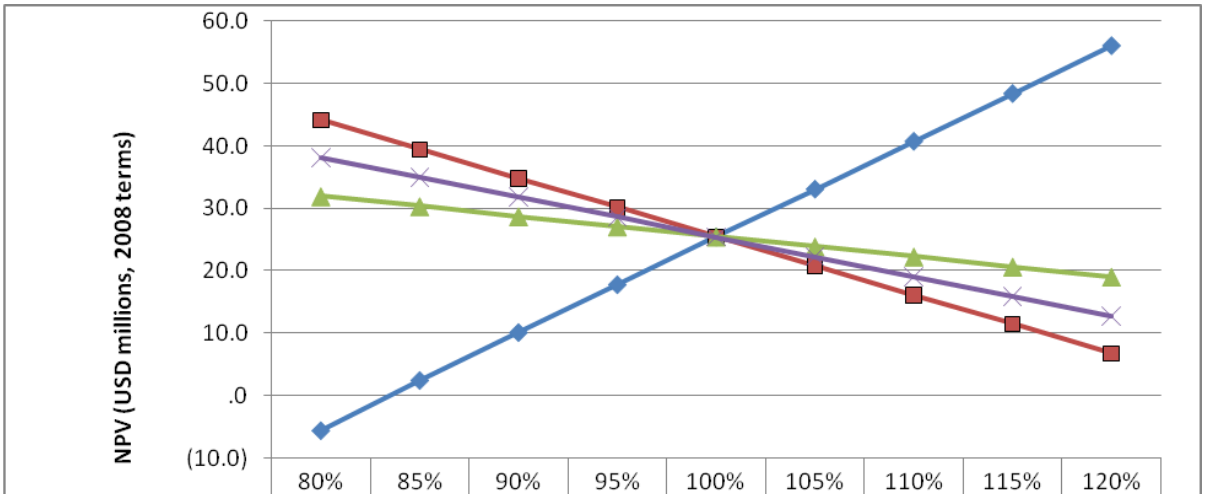


Figure 18.32
Sensitivity of NPV_{10} to Prices, Operating and Capital Costs



	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue drivers	(5.5)	2.5	10.2	17.8	25.5	33.1	40.8	48.4	56.1
Operating costs	44.1	39.5	34.8	30.1	25.5	20.8	16.1	11.5	6.8
Capital costs	31.9	30.3	28.7	27.1	25.5	23.8	22.2	20.6	19.0
Mining cost	38.2	35.0	31.8	28.6	25.5	22.3	19.1	15.9	12.7

The project is also moderately sensitive to operating costs, with a 20% adverse change sufficient to reduce NPV_{10} to less than USD 7 million. Mining costs are a very significant portion of total operating costs, and the sensitivity to mining costs is evidence of the need to carefully manage the mining contractor under the proposed project implementation.

The project is least sensitive to project capital costs, with a 20% adverse change reducing NPV_{10} by around USD 6.5 million to USD 19.0 million.

18.8.7 ALTERNATIVE PRICING SCENARIOS

Two further sensitivity studies were carried out to reflect more optimistic gold price scenarios, wherein the gold price is maintained at (i) USD 850/oz and (ii) USD 1,000/oz over the LOM period. These scenarios reflect maintenance over the LOM period of the LBMA forecast average for 2008, and maintenance of prices recently achieved in the spot market, respectively.

18.8.7.1 USD 850/oz Gold Scenario

Using a constant gold price of USD 850/oz over the LOM period, with all else unchanged from the base case, results in the discounted cash flow summarized in Table 18.25.

Table 18.25
Discounted Cash Flow for USD 850/oz Gold Scenario
(USD 000)

IRR 66.2%		LOM Total (Undisc)	NPV_5	NPV_{10}	NPV_{15}	USD per tonne	LOM Average (USD/oz)
Revenue	Gross Sales	344,375	297,664	260,188	229,709	15.21	850.00
less	Refining charges	(709)	(613)	(536)	(473)	(0.03)	(1.75)
less	Bullion delivery	(2,364)	(2,073)	(1,840)	(1,651)	(0.10)	(5.84)
	Net Sales Revenue	341,302	294,979	257,813	227,586	15.08	842.41
	Total cash op.costs	167,152	145,903	128,731	114,665	7.38	412.57
Net Cash Operating Margin		174,151	149,075	129,082	112,921	7.69	429.84
Capital Exp.	Initial/exp. capital	33,769	33,359	32,974	32,609	1.49	83.35
	Sustaining capital	12,659	10,651	9,079	7,827	0.56	31.25
Changes in Working Capital		-	611	1,049	1,366	-	-
Net cash flow before tax		127,722	104,454	85,981	71,119	5.64	315.25
Taxation payable		35,837	30,576	26,376	22,978	1.58	88.45
Net cash flow after tax		**91,885**	**73,878**	**59,605**	**48,141**	**4.06**	**226.79**

18.8.7.2 USD 1,000/oz Gold Scenario

Using a constant gold price of $1,000/oz over the LOM period, with all else unchanged from the base case, results in the discounted cash flow summarized in Table 18.26.

Table 18.26
Discounted Cash Flow for USD 1,000/oz Gold Scenario
(USD 000)

IRR 97.7%		LOM TOTAL (Undisc)	NPV_5	NPV_{10}	NPV_{15}	USD per tonne	LOM Average (USD/oz)
Revenue	Gross Sales	405,148	350,193	306,104	270,246	17.90	1,000.00
less	Refining charges	(709)	(613)	(536)	(473)	(0.03)	(1.75)
less	Bullion delivery	(2,392)	(2,097)	(1,861)	(1,669)	(0.11)	(5.90)
	Net Sales Revenue	402,046	347,484	303,707	268,104	17.76	992.35
	Total cash op.costs	167,152	145,903	128,731	114,665	7.38	412.57
Net Cash Operating Margin		234,895	201,580	174,977	153,439	10.38	579.78
Capital Exp.	Initial/exp. capital	33,769	33,359	32,974	32,609	1.49	83.35
	Sustaining capital	12,659	10,651	9,079	7,827	0.56	31.25
Changes in Working Capital		-	657	1,128	1,467		-
Net cash flow before tax		188,467	156,913	131,797	111,536	8.33	465.18
Taxation payable		52,830	45,242	39,175	34,257	2.33	130.40
Net cash flow after tax		**135,637**	**111,670**	**92,622**	**77,279**	**5.99**	**334.78**

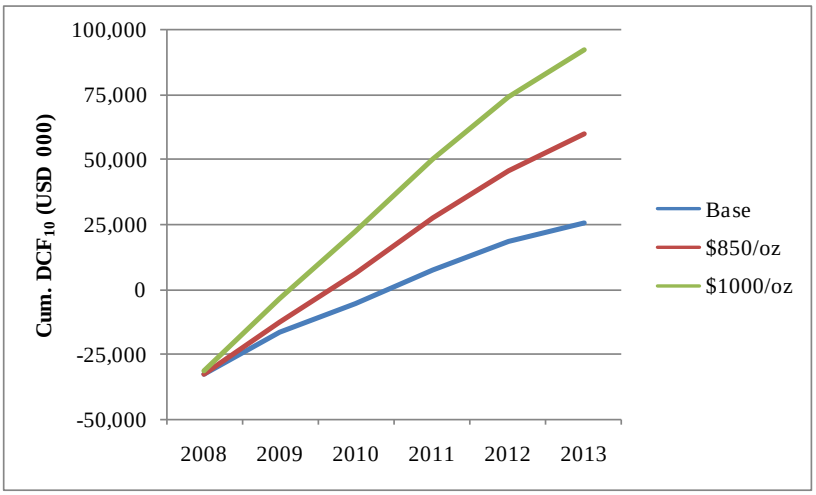
18.8.7.3 Comparison of Discounted Cash Flows

Table 18.27 and Figure 18.33 summarize the results shown above and the cumulative cash flows discounted at 10%/y for the Base Case and the more optimistic pricing scenarios described above, demonstrating the increase in NPV_{10} and shortened discounted pay-back periods achievable in the event a higher gold price is achieved over the LOM period.

Table 18.27
Summary of Results for Price Sensitivity Study

	Pre-Tax		After Tax		Discounted Payback Period (y)
	IRR (%)	NPV_{10} (USD 000)	IRR (%)	NPV_{10} (USD 000)	
Base Case	51.7	38,538	38.5	25,466	2.43
USD 850/oz	86.7	85,981	66.2	59,605	1.68
USD 1,000/oz	129.2	131,797	97.7	92,622	1.15

Figure 18.33
Comparison of Discounted Cash Flows for Gold Price Scenarios



19.0 INTERPRETATION AND CONCLUSIONS

In 2005, TMM/Timmins acquired the San Francisco project and all the historical technical, geological and geochemical data, compiled at the mine site by Geomaque. There is an abundance of data not only related to the mine site but also on the exploration programs on the project conducted by Geomaque during its tenure.

After acquiring the project, Timmins undertook a review of all available geological data surrounding the previous mining areas and identified a number of immediate exploration targets both to the northwest and to the southeast of the San Francisco pit which required further work. Timmins also spent the first few months staking the surrounding area and laying out an appropriate drilling program with the objective of confirming and extending the known mineralization around the open pit.

In August and September, 2005, Timmins conducted its first exploration program on the San Francisco project which comprised 14 reverse circulation drill holes and totalled 1,467.71 m. The initial drilling program was partially successful in confirming the results of the previous drilling programs conducted by Geomaque and expanding the limits of the known gold mineralization near the San Francisco and La Chicharra pits.

As a result of the successful 2005 exploration drilling program Timmins undertook a further exploration drilling program from September to November, 2006. The 2006 drilling program consisted of 56 holes comprised of 28 diamond drill holes and 28 reverse circulation drill holes totalling 7,310.62 m. While the drilling was primarily concentrated to the north and northwest of the San Francisco pit and to the north and northwest of the La Chicharra pit, a number of widely spaced holes were drilled to test specific geological and geochemical targets around the San Francisco pit and to the south and west of the La Chicharra and La Severiana areas.

The total estimated cost for the 2006 exploration program at the San Francisco project was USD 2,177,104. This total includes USD 134,793 spent by Timmins to pay value added taxes.

The 2006 exploration drilling program and its results were discussed in a February, 2007 Technical Report entitled “NI 43-101 Technical Report and Resource Estimate for the San Francisco Gold Property, Estación Llano, Sonora, Mexico.” This report was filed on the SEDAR website on February 27, 2007 by TMM.

The results of the 2006 drilling program for the area around the San Francisco pit have confirmed the historical drilling results and the combination of both the old and new drilling has enabled IMC to conduct a new resource estimate for the remaining mineralization not exploited by Geomaque for this area.

The resource estimate by IMC for the zones of mineralization remaining at the San Francisco pit is contained in Table 19.1.

Table 19.1
IMC Mineral Resource Estimate for the San Francisco Project
(0.23 g/t Gold Cut-off Grade)

Category	Tonnes (000 t)	Grade (g/t gold)	Contained Gold (oz)
Measured Mineral Resource	5,352	0.912	156,930
Indicated Mineral Resource	22,296	0.781	559,860
Total Measured + Indicated Resources	**27,648**	**0.806**	**716,790**
Inferred Mineral Resource	2,506	0.788	63,490

Table provided by Independent Mining Consultants, Inc.

This mineral resource has provided the basis for Timmins to conduct further exploration and to conduct the preliminary feasibility study described herein.

The preliminary feasibility study has demonstrated that, under current economic conditions, the San Francisco deposit can be mined profitably and treated by crushing and heap leaching.

Contract mining costs comprise a large proportion of the total cash operating costs for the project. Careful management of the working relationship with the selected contractor will therefore contribute significantly to the cost efficiency of the whole mine.

Metallurgical testwork conducted both off- and on-site has confirmed the importance of particle size to leach kinetics and overall gold recovery. Thus, refurbishment and efficient operation of the crushing and screening plant will be key to achieving the forecast gold production. The identification and detailed characterization of rock types within the mineral reserve will facilitate production planning and control.

The project lies close to existing infrastructure which will facilitate access, communications and connection to the available power and water supplies.

The proximity of the mine to Estación Llano makes essential the maintenance of cordial relations with the local community, and the monitoring of environmental impacts as identified in the study.

19.1 CONCLUSIONS

With the San Francisco project, TMM/Timmins have acquired a former gold mine that has known extensions to the gold mineralization both to the northwest and southeast of the San Francisco pit, as well as at depth below the present pit floor. A number of other targets remain to be explored on the property that Timmins intends to investigate further.

Since 2005, Timmins' drilling programs have continued to confirm the nature of the mineralization which Geomaque encountered during its exploration programs to the northwest of the San Francisco open pit and have been partially successful in confirming the nature of the mineralization which Geomaque encountered during its exploration programs to

the southeast of the San Francisco pit. Timmins has now combined the historical exploration work conducted by Geomaque with the results of both its drilling programs to determine further exploration targets in the immediate area to the northwest and southeast of the open pit. As well, Timmins has also combined the historical drilling information with the drilling information from its two drilling campaigns into a common database from which IMC estaimated a mineral resource for the area around the San Francisco pit.

Engineering work carried out by IMC and others, reviewed by Micon and described in this report, has demonstrated to the level of confidence expected of a preliminary feasibility study that the project could be profitably re-opened as an open pit mining, crushing and heap leaching operation. The in-pit portion of the Measured and Indicated mineral resource has therefore been classified as a Mineral Reserve (Table 19.2).

Table 19.2
Mineral Reserve Estimate

Case	Reserve Class	Cut-off Grade (g/t Au)	Reserve (000 t)	Grade (g/t)	Gold (000 oz)
High Grade Crusher feed	Probable	0.50	12,000	1.05	403.7
Low Grade Crusher feed	Probable	0.23	4,653	0.88	132.0
Sub-total Crusher feed	Probable		16,653	1.01	535.7
Low Grade ROM leach	Probable	0.28	5,981	0.39	75.3
Grand Total	**Probable**		**22,634**	**0.84**	**611.0**

Compiled by Micon from schedules provided by Independent Mining Consultants, Inc.

The San Francisco project should be considered as an advanced-stage exploration project. Micon believes that further analysis of the results of Timmins' exploration programs, followed by further exploration programs based on this work is warranted and may assist Timmins in outlining further resources at both the San Francisco and La Chicharra pits and exploration targets elsewhere on the property.

Micon considers that the results of the preliminary feasibility study warrant TMM/Timmins conducting further exploration, engineering and plant rehabilitation work on the project along with the pursuit of all outstanding statutory permits and licences, with the objective of re-starting production at the earliest opportunity.

20.0 RECOMMENDATIONS

Micon concurs with the intention of TMM/Timmins to bring the San Francisco mine project into production and recommends that:

1) Timmins continue to pursue all outstanding statutory permits and licences;

2) Timmins procede with development of the San Franscisco open pit mine, crushing, heap leaching and gold recovery plant as described in the preliminary feasibility study summarized in this report;

3) Timmins should prepare bench-by-bench estimates for the distribution of rock types comprising the mineral reserve within the open pit which can be used to improve the precision of the annual leaching recovery forecasts.

4) Prior to construction of the second phase of its heap leach pad, Timmins should investigate the potential for further expansion of the open pit towards the north-west, near the proposed leach pad area. Micon considers that this would involve both exploration drilling and engineering studies to further delimit the open pit potential using a gold price forecast prepared at that time.

5) Timmins continues to compile the San Francisco data into a single database for the property which will assist in preparing further computer-generated resource estimates; and to document and review the general QA/QC program that has been set-up within the company and apply this program in all future exploration programs.

6) In future exploration drilling all gold assays should be duplicated and samples which show a range of assays greater than 10% should be assayed by the screen metallics procedure.

Given the amount of work conducted previously at the San Francisco project on the known exploration targets and areas of mineralization, the property should be regarded as an advanced-stage project with significant economic potential.

Micon has reviewed the results of the historical exploration programs and the results from the, 2005 and 2006 drilling programs conducted by Timmins and, in light of the observations made in the Conclusions and Recommendations Section of this report, supports the concepts outlined by Timmins for further exploration. It is Micon's opinion that the property merits further exploration and economic evaluation and that TMM/Timmins' proposed exploration plans are properly conceived and justified.

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APPENDIX I
Check Assay Results for the 2006 San Francisco Drilling

Diamond Drilling				Reverse Circulation Drilling			
Drill Hole Number	**Sample Number**	**ACME Assay 1 (g/t gold)**	**ALS Chemex Assay 2 (g/t gold)**	**Drill Hole Number**	**Sample Number**	**ACME Assay 1 (g/t gold)**	**ALS Chemex Assay 2 (g/t gold)**
TFD-04	642676	1.257	1.265	TF-19	618810	0.111	0.097
	642685	0.024	0.033		618829	0.011	0.013
	642695	0.002	<0.005		618839	0.003	0.005
	642705	0.009	0.015		618849	0.022	0.027
	642715	0.005	0.006		618859	0.019	0.021
	642725	0.015	0.007		618869	0.009	0.020
	642735	0.551	0.420		618879	0.014	0.014
	642745	0.664	0.696		618889	0.020	0.015
	642755	<0.002	<0.005		618899	0.061	0.047
	642765	0.018	0.018		618909	0.025	0.036
	642775	0.036	0.034		618919	0.034	0.033
	642785	0.003	<0.005		618929	0.012	0.010
	642790	0.008	0.005		618939	0.291	0.266
	642799	0.005	<0.005		618949	0.107	0.102
	642809	<0.002	<0.005		618959	0.056	0.048
	642819	0.007	<0.005		618969	0.009	0.006
	642830	0.012	0.008	TF-25	618983	0.028	0.025
	642839	0.617	0.585		618996	0.081	0.091
	642849	0.014	0.005		619008	0.024	0.010
	642859	0.024	0.023		619019	0.009	0.005
	642869	0.002	<0.005		619030	0.037	0.021
	642879	0.003	<0.005		619040	0.022	0.015
	642889	0.007	<0.005		619052	0.005	0.008
	642899	0.009	0.011		619063	0.012	0.014
TFD-05	642466	0.050	0.072		619073	0.004	0.009
	642475	0.012	0.007	TF-26	619090	0.013	0.010
	642485	0.019	0.032		619103	0.008	0.009
	642495	0.004	<0.005		619113	0.004	<0.005
	642505	0.002	<0.005		619124	0.010	0.013
	642515	0.008	0.006		619134	0.009	0.005
	642525	0.003	<0.005		619145	0.004	<0.005
	642535	0.221	0.210		619155	0.010	0.008
	642545	0.509	0.499		619166	0.021	0.022
	642555	0.006	<0.005		619176	<0.002	0.033
	642565	0.007	0.009		619186	2.550	2.560
	642575	<0.002	<0.005		619194 (**3**)	3.632	1.565
	642585	0.394	0.370		619207	0.199	0.225
	642595	0.005	0.005	TF-27	619221	0.115	0.114
	642605	0.002	<0.005		619229	0.032	0.012
	642615	0.009	0.013		619244	0.018	0.013
	642625	0.036	0.031		619254	<0.002	0.006
	642635	0.005	0.008		619262	0.005	0.012
	642645	0.028	0.031	TF-28	619272	0.014	0.006
	642655	0.020	0.025		619285 (**3**)	0.590	0.064
	642665	0.280	0.331		619299	<0.002	<0.005
	642675	0.008	0.006		619311	0.004	<0.005
TFD-06	643265	0.006	0.006		619329	<0.002	<0.005
	643274	0.007	0.025	TF-29	619632	0.005	0.009
	643284	0.002	<0.005		619646	0.028	0.013
	643294	0.002	<0.005		619656	0.005	0.006
	643304	0.006	<0.005		619668	0.031	0.033
	643314	0.003	<0.005	TF-30	619585	0.031	0.044
	643324	0.005	<0.005		619595	0.032	0.029
	643334	0.003	<0.005		619604	0.025	0.006
	643344	0.383	0.435		619616	0.074	0.076
	643354	0.004	<0.005	TF-31	619537	0.004	0.006
	643361	0.004	<0.005		619552	0.026	0.022
	643364	0.004	<0.005		619558	0.007	<0.005
	643370	0.019	0.023		619567	0.005	<0.005
	643380	0.005	0.005		619581	0.002	<0.005
	643390	0.009	0.019	TF-32	619339	0.020	0.017

Diamond Drilling			
Drill Hole Number	**Sample Number**	**ACME Assay 1 (g/t gold)**	**ALS Chemex Assay 2 (g/t gold)**
	643400	0.035	0.040
	643410	0.004	<0.005
	643420	0.003	<0.005
	643430	<0.002	<0.005
	643440	0.009	0.016
	643451	0.340	0.310
	643460	0.046	0.049
TFD-07	643600	0.009	0.008
	643612	0.004	<0.005
	643627	0.008	0.007
	643633	0.025	0.022
	643646	0.036	0.024
	643658	0.010	0.010
	643668	0.019	0.018
	643676	0.009	0.006
	643689	0.002	<0.005
	643700	0.004	<0.005
	643714	<0.002	<0.005
	643723	0.006	<0.005
	643732	0.004	0.006
	643741	0.002	0.007
	643748	<0.002	<0.005
TFD-08	642905	0.009	0.007
	642914	0.006	0.007
	642924	0.007	<0.005
	642934	0.004	<0.005
	642944	0.004	<0.005
	642954	0.202	0.127
	642964	<0.002	0.007
	642974	0.006	<0.005
	642984	0.010	0.009
	642994	0.007	<0.005
	643201	0.006	0.006
	643210	0.005	<0.005
	643220	0.006	0.005
	643230	<0.002	<0.005
	643240	0.005	<0.005
	643250	0.002	<0.005
	643261	0.007	<0.005
TFD-09	643760	0.004	<0.005
	643774	<0.002	<0.005
	643787 (**3**)	0.720	0.089
	643797	0.087	0.081
	643808	0.007	<0.005
	643819	0.004	<0.005
	643829	0.016	0.014
	643833	0.014	0.014
	643840	0.018	0.014
	643849	0.014	0.012
	643859	0.007	0.005
	643875	0.004	<0.005
TFD-10	643466	0.012	<0.005
	643474	0.015	0.018
	643484	0.022	0.026
	643494	0.018	0.007
	643504	0.141	0.183
	643514	0.155	0.165
	643524	0.285	0.260
	643534	0.012	0.015
	643544	0.005	0.006
	643554	0.008	0.010
	643564	0.006	<0.005
	643574	0.006	0.009
	643584	0.003	<0.005

Reverse Circulation Drilling			
Drill Hole Number	**Sample Number**	**ACME Assay 1 (g/t gold)**	**ALS Chemex Assay 2 (g/t gold)**
	619351	0.006	0.005
	619360	0.006	<0.005
	619373	0.003	<0.005
TF-33	619383	0.026	0.068
	619393	0.006	0.019
	619406	0.005	<0.005
	619418	0.024	0.027
TF-34	619427	<0.002	<0.005
	619435	0.023	0.020
	619445	0.005	<0.005
	619456	0.066	0.063
	619470	0.011	0.005
TF-35	619482	<0.002	<0.005
	619489	0.006	0.006
	619500	<0.002	<0.005
	619518	0.009	0.007
	619531	0.003	0.009
TF-36	619685	0.014	0.013
	619692	0.006	<0.005
TF-37	619801	0.002	<0.005
	619814	0.026	0.005
	619832	0.020	0.006
TF-38	619845	0.010	0.008
	619855	0.008	0.008
	619870	0.006	0.006
	619881	0.007	0.007
	619890	0.008	<0.005
	619898	0.015	0.010
	619905	<0.002	0.006
TF-39	619916	0.027	0.027
	619927	0.011	0.006
	619941	0.012	0.011
	619955	<0.002	<0.005
	619962	0.005	<0.005
	619974	0.007	0.005
	619987	0.006	0.006
TF-41	726576	0.006	<0.005
	726585	0.008	0.007
	726596	<0.002	<0.005
	726607	0.004	0.008
	726618	0.006	0.005
	726627	0.004	<0.005

1 Note: The gold analysis was done using Acme method Group 3B with lower and upper detection limits of 0.002 and 10 ppm respectively.

2 Note: The gold analysis was done using ALS Chemex method Au-AA23 with lower and upper detection limits of 0.005 and 10 ppm respectively.

3 Note: Samples 619194 (TF-26), 619285 (TF-28) and 643787 (TFD-09) were removed from the interpretation of the check data base due to possible contamination of the sample either during one or both assay procedures or nuggety gold in the sample. These samples need to be reassayed to determine the reason for the wide differences in the assays.

Diamond Drilling				Reverse Circulation Drilling			
Drill Hole Number	Sample Number	ACME Assay 1 (g/t gold)	ALS Chemex Assay 2 (g/t gold)	Drill Hole Number	Sample Number	ACME Assay 1 (g/t gold)	ALS Chemex Assay 2 (g/t gold)
TFD-11	640104	0.002	<0.005				
	640112	0.407	0.392				
	643879	0.017	0.012				
	643890	0.010	0.007				
	643900	0.115	0.029				
	643907	0.010	<0.005				
	643918	0.240	0.019				
	643929	0.005	0.005				
	643935	0.004	<0.005				
	643946	0.003	<0.005				
	643956	1.233	1.335				
	643968	1.488	1.360				
	643985	0.129	0.126				
	643992	0.107	0.123				
TFD-12	640003	0.009	<0.005				
	640014	0.010	0.005				
	640025	0.001	0.005				
	640042	0.147	0.016				
	640055	0.024	0.022				
	640064	0.228	0.252				
	640075	0.115	0.135				
	640082	0.345	0.305				
	640092	0.416	0.335				
TFD-13	640219	0.006	<0.005				
	640227	0.009	0.005				
	640246	0.002	<0.005				
	640259	0.247	0.221				
	640269	<0.002	<0.005				
	640284	0.036	0.035				
TFD-14	640454	0.048	0.055				
	640464	0.008	0.006				
	640476	0.008	<0.005				
	640484	0.010	<0.005				
	640496	0.066	0.069				
	640509	0.008	<0.005				
	640516	0.003	<0.005				
	640526	0.002	<0.005				
	640538	0.086	0.076				
TFD-15	640120	0.043	0.028				
	640134	0.007	0.005				
	640151	0.050	0.042				
	640160	0.005	0.007				
	640174	0.011	0.016				
	640183	0.011	0.009				
	640196	0.005	<0.005				
	640201	0.003	<0.005				
	640211	0.006	<0.005				
TFD-16	640666	0.007	<0.005				
	640674	0.023	0.022				
	640688	0.020	<0.005				
	640696	0.004	<0.005				
	640706	0.005	<0.005				
	640717	0.002	0.010				
TFD-17	640406	0.005	<0.005				
	640416	0.005	<0.005				
	640431	0.006	<0.005				
	640442	0.029	0.043				
TFD-20	640772	0.005	0.005				
	640781	0.015	0.007				
	640795	0.005	<0.005				
	640806	0.005	<0.005				

Table provided by Timmins

APPENDIX II
Assaying Results for the Individual Standard Reference Samples, 2006 San Francisco Project Drilling

Combined In House Standard ReferenceSample (561093 + 561195)					In House Standard Reference Sample (636077)				
Drilling Type	Hole Number	Sample Number	Acme Assay (g/t gold)	Standard Assay (g/t gold)	Drilling Type	Hole Number	Sample Number	Acme Assay (g/t gold)	Standard Assay (g/t gold)
Reverse Circulation	TF-15	618446	0.494	0.520	Reverse Circulation	TF-15	618464	0.230	0.259
		618487	0.608	0.520			618503	0.264	0.259
		618525	0.531	0.520			618544	0.215	0.259
		618562	0.606	0.520			618622	0.259	0.259
		618581	0.602	0.520		TF-16	618639	0.248	0.259
		618605	0.521	0.520			618697	0.238	0.259
	TF-16	618654	0.609	0.520			618753	0.250	0.259
		618675	0.523	0.520		TF-18	618033	0.196	0.259
		618713	0.461	0.520			618088	0.249	0.259
		618732	0.531	0.520			618143	0.242	0.259
		618774	0.704	0.520		TF-19	618871	0.396	0.259
		618793	0.585	0.520			618933	0.222	0.259
	TF-17	726650	0.541	0.520		TF-20 R	618363	0.231	0.259
		726664	0.498	0.520			618388	0.222	0.259
	TF-18	618012	0.528	0.520		TF-21	726807	0.273	0.259
		618051	0.603	0.520		TF-22	726935	0.194	0.259
		618069	0.504	0.520		TF-25	618990	0.177	0.259
		618107	0.495	0.520			619041	0.201	0.259
		618127	0.559	0.520			619081	0.397	0.259
		618162	0.536	0.520		TF-26	619118	0.204	0.259
		618183	0.464	0.520			619179	0.218	0.259
	TF-19	618820	0.784	0.520		TF-27	619238	0.222	0.259
		618835	0.511	0.520		TF-28	619298	0.252	0.259
		618851	0.472	0.520		TF-29	619645	0.200	0.259
		618895	0.561	0.520		TF-31	619568	0.207	0.259
		618912	0.582	0.520		TF-32	619362	0.221	0.259
		618952	0.567	0.520		TF-34	619436	0.201	0.259
	TF-20 R	618249	0.464	0.520		TF-35	619511	0.198	0.259
		618269	0.519	0.520		TF-36	619708	0.227	0.259
		618287	0.506	0.520		TF-37	619824	0.201	0.259
		618304	0.433	0.520		TF-38	619848	0.226	0.259
		618329	0.498	0.520			619902	0.253	0.259
		618348	0.468	0.520		TF-39	619942	0.271	0.259
		618405	0.602	0.520		TF-40	726506	0.211	0.259
	TF-21	726793	0.481	0.520			726522	0.315	0.259
	TF-22	726921	0.445	0.520			726560	0.212	0.259
	TF-23	726753	0.525	0.520		TF-41	726600	0.193	0.259
	TF-24	726712	0.590	0.520		TF-42	726881	0.184	0.259
	TF-25	619005	0.495	0.520	Diamond Drilling	TFD-01	643056	0.223	0.259
		619026	0.494	0.520			643127	0.227	0.259
		619064	0.535	0.520		TFD-02	642240	0.193	0.259
	TF-26	619101	0.499	0.520			642326	0.211	0.259
		619137	0.469	0.520			642410	0.188	0.259
		619156	0.514	0.520		TFD-03	642025	0.230	0.259
		619196	0.516	0.520			642100	0.232	0.259
	TF-27	619224	0.582	0.520			642222	0.240	0.259
		619258	0.503	0.520		TFD-04	642774	0.287	0.259
	TF-28	619281	0.491	0.520			642896	0.015	0.259
		619317	0.522	0.520		TFD-05	642482	0.207	0.259
	TF-29	619663	0.471	0.520			642566	0.203	0.259
	TF-30	619596	0.487	0.520			642650	0.201	0.259
		619614	0.520	0.520		TFD-06	643337	0.219	0.259
	TF-31	619550	0.634	0.520			643422	0.214	0.259
	TF-32	619344	0.490	0.520		TFD-07	643662	0.224	0.259
	TF-33	619392	0.534	0.520			643746	0.129	0.259
		619407	0.478	0.520		TFD-08	643260	0.217	0.259
	TF-34	619453	0.531	0.520		TFD-09	643795	0.203	0.259
		619476	0.454	0.520		TFD-10	643481	0.271	0.259
	TF-35	619493	0.460	0.520			643565	0.254	0.259

Combined In House Standard ReferenceSample (561093 + 561195)				
Drilling Type	**Hole Number**	**Sample Number**	**Acme Assay (g/t gold)**	**Standard Assay (g/t gold)**
		619529	0.431	0.520
	TF-36	619728	0.535	0.520
		619749	0.470	0.520
	TF-37	619770	0.510	0.520
		619786	0.506	0.520
		619805	0.574	0.520
	TF-38	619864	0.575	0.520
		619883	0.483	0.520
	TF-39	619928	0.510	0.520
		619961	0.588	0.520
		619982	0.532	0.520
	TF-41	726619	0.570	0.520
Diamond Drilling	TFD-01	643028	0.524	0.520
		643084	0.558	0.520
		643099	0.554	0.520
		643153	0.473	0.520
		643183	0.531	0.520
	TFD-02	642269	0.641	0.520
		642298	0.581	0.520
		642354	0.505	0.520
		642382	0.534	0.520
		642438	0.517	0.520
		642465	0.492	0.520
	TFD-03	642006	0.495	0.520
		642068	0.514	0.520
		642128	0.625	0.520
		642139	0.545	0.520
		642181	0.554	0.520
		642200	0.540	0.520
	TFD-04	642693	0.450	0.520
		642721	0.541	0.520
		642801	0.885	0.520
		642829	0.706	0.520
		642857	0.459	0.520
		642885	0.472	0.520
	TFD-05	642538	0.431	0.520
		642594	0.401	0.520
		642622	0.554	0.520
	TFD-06	643281	0.459	0.520
		643309	0.607	0.520
		643365	0.524	0.520
		643393	0.512	0.520
		643450	0.585	0.520
	TFD-07	643605	0.440	0.520
		643634	0.543	0.520
		643690	0.408	0.520
		643718	0.415	0.520
	TFD-08	642920	0.465	0.520
		642975	0.433	0.520
		643203	0.536	0.520
		643231	0.595	0.520
	TFD-09	643767	0.452	0.520
		643823	0.531	0.520
		643851	0.525	0.520
	TFD-10	643509	0.637	0.520
		643537	0.497	0.520
	TFD-11	643924	0.574	0.520
		640108	0.504	0.520
	TFD-12	640017	0.433	0.520
		640073	0.605	0.520
	TFD-13	640230	0.520	0.520
	TFD-14	640465	0.520	0.520

In House Standard Reference Sample (636077)				
Drilling Type	**Hole Number**	**Sample Number**	**Acme Assay (g/t gold)**	**Standard Assay (g/t gold)**
	TFD-11	643895	0.385	0.259
		643952	0.258	0.259
		643980	0.191	0.259
	TFD-12	640045	0.189	0.259
	TFD-13	640258	0.158	0.259
		640286	0.362	0.259
	TFD-15	640189	0.221	0.259
	TFD-16	640705	0.223	0.259
	TFD-17	640445	0.182	0.259
Purchased Standard Reference Sample (OxG46)				
Reverse Circulation	TF-17	726681	0.968	1.037
	TF-19	618970	1.024	1.037
	TF-21	726824	1.026	1.037
	TF-22	726952	0.950	1.037
	TF-23	726767	1.013	1.037
	TF-24	726726	0.974	1.037
	TF-36	619690	1.058	1.037
	TF-40	726541	1.000	1.037
	TF-41	726586	1.040	1.037
	TF-42	726896	1.085	1.037
Diamond Drilling	TFD-03	642037	0.971	1.037
	TFD-19	640619	0.937	1.037
		640647	0.935	1.037
	TFD-21	640762	0.920	1.037
	TFD-22	640835	0.949	1.037
		640863	1.055	1.037
		640891	0.986	1.037
	TFD-24	728115	1.004	1.037
		728143	0.961	1.037
	TFD-25	728017	0.989	1.037
		728045	1.001	1.037
		728073	1.010	1.037
	TFD-26	728297	1.003	1.037
		728325	1.027	1.037
		728353	0.976	1.037
	TFD-27	728234	1.008	1.037
		728262	1.025	1.037
	TFD-28	728388	1.011	1.037
		728416	0.952	1.037
		728443	0.948	1.037

Combined In House Standard ReferenceSample (561093 + 561195)					In House Standard Reference Sample (636077)				
Drilling Type	**Hole Number**	**Sample Number**	**Acme Assay (g/t gold)**	**Standard Assay (g/t gold)**	**Drilling Type**	**Hole Number**	**Sample Number**	**Acme Assay (g/t gold)**	**Standard Assay (g/t gold)**
		640493	0.590	0.520					
		640521	0.410	0.520					
	TFD-15	640133	0.510	0.520					
		640161	0.522	0.520					
	TFD-16	640676	0.558	0.520					
	TFD-17	640417	0.495	0.520					
	TFD-20	640784	0.462	0.520					
		640812	0.498	0.520					

Table provided by Timmins

APPENDIX III
Individual Blank Assays for the 2006 San Francisco Project Drilling

Diamond Drilling			Reverse Circulation Drilling		
Drill Hole Number	Sample Number	Gold (g/t)	Drill Hole Number	Sample Number	Gold (g/t)
TFD-01	643017	<0.005	TF-15	618440	<0.005
	643045	<0.005		618457	<0.005
	643073	<0.005		618479	0.005
	643106	<0.005		618497	<0.005
	643137	<0.005		618519	<0.005
	643166	0.015		618538	<0.005
TFD-02	642252	<0.005		618556	<0.005
	642280	0.005		618573	0.011
	642308	<0.005		618596	<0.005
	642337	0.006		618613	<0.005
	642365	<0.005	TF-16	618634	<0.005
	642393	<0.005		618648	<0.005
	642420	<0.005		618667	0.007
	642449	<0.005		618689	0.009
TFD-03	642017	<0.005		618706	0.005
	642047	0.013		618725	<0.005
	642082	0.006		618747	0.015
	642112	0.007		618766	0.073
	642165	<0.005		618783	<0.005
	642217	<0.005		618805	<0.005
TFD-04	642704	0.002	TF-17	726641	0.004
	642732	0.003		726660	0.002
	642757	0.003		726676	0.002
	642784	0.004	TF-18	618007	0.025
	642812	<0.002		618028	0.01
	642840	0.002		618045	<0.005
	642868	0.004		618064	<0.005
TFD-05	642493	0.008		618083	0.005
	642521	0.005		618102	0.008
	642549	<0.002		618121	0.007
	642577	0.004		618139	<0.005
	642605	0.002		618157	<0.005
	642633	<0.002		618178	0.012
	642661	<0.002	TF-19	618813	0.008
TFD-06	643292	0.002		618828	0.002
	643320	0.005		618846	0.009
	643348	0.006		618866	0.004
	643376	0.006		618890	0.009
	643404	0.005		618907	0.005
	643433	0.003		618925	0.010
	643461	Lost		618947	0.002
TFD-07	643616	0.003		618963	0.007
	643645	0.004	TF-20 R	618243	0.006
	643673	0.002		618261	0.017
	643701	<0.002		618282	<0.005
	643729	0.006		618301	0.005
TFD-08	642931	0.002		618320	<0.005
	642959	<0.002		618339	<0.005
	642986	0.005		618358	<0.005
	643214	0.002		618382	<0.005
	643242	0.002		618398	0.006
TFD-09	643778	<0.002		618421	0.003
	643806	<0.002	TF-21	726784	<0.002
	643834	0.002		726803	0.009
	643862	0.004		726819	0.002
TFD-10	643492	0.003	TF-22	726912	0.003
	643520	0.008		726931	0.006
	643548	0.005		726947	0.006
	643576	0.005	TF-23	726744	0.006
TFD-11	643906	0.004		726763	<0.002
	643935	0.004	TF-24	726705	0.002

Diamond Drilling			Reverse Circulation Drilling		
Drill Hole Number	**Sample Number**	**Gold (g/t)**	**Drill Hole Number**	**Sample Number**	**Gold (g/t)**
	643963	0.017		726721	0.007
	643991	0.003	TF-25	618987	0.009
TFD-12	640028	0.01		619000	0.012
	640056	<0.002		619018	0.008
	640084	0.002		619036	0.003
TFD-13	640241	<0.002		619057	0.002
	640269	<0.002		619076	0.005
	640297	0.002	TF-26	619096	0.004
TFD-14	640476	0.008		619113	0.007
	640504	0.010		619132	0.003
	640532	0.002		619149	<0.002
TFD-15	640144	0.003		619171	0.008
	640172	0.044		619190	0.018
	640200	0.003	TF-27	619219	0.013
TFD-16	640687	0.004		619233	0.003
TFD-17	640428	0.019		619252	<0.002
TFD-18	640573	0.004	TF-28	619274	0.009
	640601	0.005		619292	0.006
TFD-19	640630	0.002		619312	0.002
	640658	0.006	TF-29	619638	<0.002
TFD-20	640795	0.004		619659	0.005
TFD-21	640745	0.005	TF-30	619589	0.004
TFD-22	640846	0.012		619610	0.010
	640874	0.006	TF-31	619543	0.009
	640902	0.003		619564	0.004
TFD-23	640943	<0.002	TF-32	619338	0.003
	640971	<0.002		619353	0.003
	640999	0.005		619373	<0.002
	646327	<0.002	TF-33	619387	0.004
TFD-24	728126	0.018		619403	0.002
	728154	0.003		619421	0.002
	728182	0.026	TF-34	619429	0.002
TFD-25	728028	0.004		619447	0.003
	728056	<0.002		619467	0.004
	728084	0.003	TF-35	619487	<0.002
TFD-26	728308	0.004		619504	<0.002
	728336	0.007		619523	0.010
	728364	<0.002	TF-36	619683	0.004
TFD-27	728217	0.032		619704	0.004
	728245	0.017		619722	0.006
	728273	0.004		619744	0.002
TFD-28	728399	0.272	TF-37	619761	<0.002
	728426	<0.002		619781	0.003
				619799	0.007
				619819	0.003
			TF-38	619841	0.002
				619859	0.002
				619879	0.002
				619897	<0.002
			TF-39	619921	<0.002
				619937	<0.002
				619955	0.006
			TF-40	619978	<0.002
				726517	<0.002
				726537	<0.002
				726555	<0.002
			TF-41	726579	<0.002
				726595	0.003
				726613	0.000

Table provided by Timmins

APPENDIX IV
Assay Results for Individual Duplicate Samples, 2006 Drilling

Diamond Drilling				
Drill Hole Number	Original Assay**		Duplicate Assay**	
	Sample No.	Gold (g/t)	Sample No.	Gold (g/t)
TFD-01	643005	<0.005	643006	<0.005
	643033	<0.005	643034	<0.005
	643061	0.011	643062	0.015
	643088	<0.005	643089	<0.005
	643117	>10.0	643118	>10.0
	643143	0.119	643144	0.097
	643175	0.01	643176	<0.005
TFD-02	642229	0.01	642230	0.011
	642257	0.016	642258	0.011
	642285	0.008	642286	0.009
	642314	0.056	642315	0.032
	642342	<0.005	642343	<0.005
	642370	0.01	642371	0.011
	642399	0.006	642400	<0.005
	642426	0.155	642427	0.235
	642454	0.113	642455	0.076
TFD-03	642012	<0.005	642013	0.005
	642029	<0.005	642030	<0.005
	642059	0.053	642060	0.211
	642089	<0.005	642090	0.006
	642119	0.678	642120	0.368
	642172	0.054	642173	0.15
	642194	0.024	642195	0.007
	642211	0.007	642212	0.005
TFD-04	642681	0.564	642682	0.868
	642709	0.005	642710	0.003
	642737	0.287	642738	0.284
	642762	0.004	642763	0.003
	642789	0.006	642790	0.008
	642817	0.111	642818	0.003
	642845	0.002	642846	0.004
	642873	0.008	642874	0.002
	642901	<0.002	642902	<0.002
TFD-05	642470	2.009	642471	0.090
	642498	0.019	642499	0.023
	642526	0.004	642527	0.002
	642554	0.005	642555	0.006
	642582	0.044	642583	0.543
	642610	0.125	642611	0.007
	642638	0.007	642639	0.005
	642666	0.056	642667	0.006
TFD-06	643269	0.381	643270	0.006
	643297	0.005	643298	0.008
	643325	0.004	643326	0.005
	643353	0.004	643354	0.002
	643381	0.011	643382	0.013
	643410	0.003	643411	0.003
	643438	0.663	643439	0.494
TFD-07	643593	<0.002	643594	0.002
	643622	0.003	643623	0.004
	643650	0.025	643651	0.008
	643678	0.023	643679	0.044
	643706	0.005	643707	0.008
	643734	0.347	643735	0.424
TFD-08	642908	<0.002	642909	0.004
	642936	0.047	642937	0.047
	642991	0.006	642992	0.007
	643219	0.010	643220	0.008
	643247	0.007	643248	0.009
TFD-09	643755	0.003	643756	0.003

Reverse Circulation Drilling				
Drill Hole Number	Original Assay**		Duplicate Assay**	
	Sample No.	Gold (g/t)	Sample No.	Gold (g/t)
TF-15	618450	0.617	618451	0.617
	618471	0.023	618472	0.015
	618490	0.256	618491	0.254
	618509	0.085	618510	0.087
	618529	0.008	618530	0.006
	618548	<0.005	618549	<0.005
	618568	<0.005	618569	<0.005
	618585	0.094	618586	0.087
	618607	0.028	618608	0.037
	618626	0.340	618627	0.351
TF-16	618642	0.021	618643	0.018
	618661	0.011	618662	0.008
	618680	0.21	618681	0.204
	618700	0.007	618701	0.006
	618719	0.005	618720	<0.005
	618739	0.006	618740	0.005
	618758	0.013	618759	0.015
	618777	0.041	618778	0.042
	618797	<0.005	618798	<0.005
TF-17	726654	0.004	726655	0.004
	726669	0.005	726670	0.006
	726687	0.614	726688	1.306
TF-18	618017	0.014	618018	0.017
	618038	<0.005	618039	0.005
	618057	0.006	618058	0.006
	618074	0.009	618075	0.014
	618095	2.59	618096	3.87
	618112	0.115	618113	0.109
	618133	0.006	618134	0.008
	618150	0.05	618151	0.031
	618168	0.018	618169	0.017
	618190	0.009	618191	0.013
TF-19	618822	0.015	618823	0.020
	618837	0.004	618838	<0.002
	618857	0.014	618858	0.009
	618875	0.002	618876	0.005
	618898	0.039	618899	0.061
	618919	0.034	618920	0.033
	618939	0.291	618940	0.229
	618956	0.016	618957	0.020
TF-20 R	618209	1.975	618210	1.975
	618228	0.011	618229	0.011
	618254	1.565	618255	1.53
	618273	0.006	618274	0.007
	618294	0.01	618295	0.007
	618310	0.005	618311	0.007
	618336	0.008	618337	0.009
	618353	0.006	618354	0.007
	618370	0.005	618371	<0.005
	618393	<0.005	618394	<0.005
	618413	0.019	618414	0.016
	618430	0.016	618431	0.012
TF-21	726797	0.016	726798	0.017
	726812	0.008	726813	0.006
TF-22	726925	0.006	726926	0.004
	726940	0.063	726941	0.064
TF-23	726757	0.008	726758	0.007
	726772	0.021	726773	0.02
TF-24	726715	0.002	726716	0.007
	726731	0.03	726732	0.041

Diamond Drilling					Reverse Circulation Drilling				
Drill Hole Number	Original Assay**		Duplicate Assay**		Drill Hole Number	Original Assay**		Duplicate Assay**	
	Sample No.	Gold (g/t)	Sample No.	Gold (g/t)		Sample No.	Gold (g/t)	Sample No.	Gold (g/t)
	643783	0.035	643784	0.042	TF-25	618992	0.124	618993	0.219
	643811	0.016	643812	0.012		619010	0.023	619011	0.033
	643839	0.03	643840	0.018		619029	0.032	619030	0.037
	643867	0.015	643868	0.014		619051	0.006	619052	0.005
TFD-10	643469	0.029	643470	0.013		619069	<0.002	619070	0.005
	643497	0.031	643498	0.013	TF-26	619103	0.008	619104	0.009
	643525	0.013	643526	0.015		619125	0.015	619126	0.011
	643553	0.007	643554	0.008		619142	0.012	619143	0.008
	643581	0.011	643582	0.004		619161	0.023	619162	0.02
TFD-11	643883	0.004	643884	0.008		619183	0.773	619184	0.730
	643911	0.007	643912	0.008		619202	0.033	619203	0.033
	643940	0.003	643941	<0.002	TF-27	619226	0.815	619227	0.672
	643968	1.488	643969	1.992		619244	0.018	619245	0.019
	643996	0.204	643997	0.096		619263	0.017	619264	0.035
TFD-12	640005	0.013	640006	0.01	TF-28	619285	0.059	619286	0.042
	640033	0.006	640034	0.022		619303	0.002	619304	<0.002
	640061	0.554	640062	0.428		619322	<0.002	619323	<0.002
	640089	0.228	640090	0.209	TF-29	619648	1.074	619649	0.088
TFD-13	640218	0.005	640219	0.006		619668	0.031	619669	0.042
	640246	0.002	640247	0.003	TF-30	619599	0.020	619600	0.019
	640274	1.417	640275	1.800		619618	0.010	619619	0.010
TFD-14	640452	0.027	640453	0.013	TF-31	619553	0.016	619554	0.008
	640481	0.008	640482	0.015		619572	0.003	619573	0.003
	640509	0.008	640510	0.006	TF-32	619347	0.014	619348	0.009
	640537	0.093	640538	0.086		619365	0.004	619366	0.019
TFD-15	640121	0.031	640122	0.023	TF-33	619396	0.008	619397	0.012
	640149	0.037	640150	0.053		619413	0.004	619414	0.003
	640177	0.029	640178	0.011	TF-34	619440	0.006	619441	0.004
	640205	0.004	640206	<0.002		619458	0.015	619459	0.026
TFD-16	640664	0.004	640665	<0.002	TF-35	619497	0.004	619498	0.004
	640692	0.005	640693	0.005		619516	0.005	619517	0.007
TFD-17	640405	0.005	640406	0.005		619532	0.002	619533	0.002
	640433	0.009	640434	<0.002	TF-36	619693	0.007	619694	0.011
TFD-18	640550	0.003	640551	0.004		619712	0.003	619713	0.006
	640578	0.471	640579	0.006		619731	0.004	619732	0.002
TFD-19	640607	0.010	640608	0.051		619752	0.008	619753	0.007
	640635	0.002	640636	0.001	TF-37	619773	0.002	619774	0.010
TFD-20	640771	0.003	640772	0.005		619792	0.004	619793	0.008
	640800	0.003	640801	0.004		619812	0.014	619813	0.012
TFD-21	640722	0.002	640723	<0.005		619829	0.007	619830	0.009
	640750	0.001	640751	0.002	TF38	619852	0.010	619853	0.013
TFD-22	640823	<0.002	640824	0.003		619870	0.006	619871	0.003
	640851	0.009	640852	0.011		619890	0.008	619891	0.004
	640879	0.002	640880	0.005		619906	0.023	619907	0.014
	640907	0.009	640908	0.007	TF-39	619931	0.007	619932	0.009
TFD-23	640920	0.153	640921	0.232		619947	0.006	619948	0.006
	640948	0.136	640949	0.201		619966	0.014	619967	0.009
	640976	<0.002	640977	<0.002		619987	0.006	619988	0.008
	646304	<0.002	646305	0.003	TF-40	726510	0.012	726511	0.012
	646332	<0.002	646333	<0.002		726528	0.404	726529	0.631
TFD-24	728103	0.02	728104	0.007		726548	0.004	726549	0.004
	728131	0.265	728132	0.135		726564	0.002	726565	0.003
	728159	0.004	728160	0.005	TF-41	726589	0.009	726590	0.005
	728187	<0.002	728188	<0.002		726605	0.004	726606	0.007
TFD-25	728005	0.005	728006	0.006		726624	0.013	726625	0.012
	728033	0.013	728034	0.011	TF-42	726885	0.077	726886	0.073
	728061	<0.002	728062	<0.002					
	728089	0.006	728090	0.004					
TFD-26	728285	0.017	728286	0.013					
	728313	0.123	728314	0.032					
	728341	<0.002	728342	0.006					
	728369	0.003	728370	0.007					
TFD-27	728194	0.059	728195	0.03					

Diamond Drilling					Reverse Circulation Drilling				
Drill Hole Number	Original Assay**		Duplicate Assay**		Drill Hole Number	Original Assay**		Duplicate Assay**	
	Sample No.	Gold (g/t)	Sample No.	Gold (g/t)		Sample No.	Gold (g/t)	Sample No.	Gold (g/t)
	728222	0.021	728223	0.032					
	728250	0.197	728251	0.505					
	728278	0.048	728279	0.054					
TFD-28	728376	0.013	728377	0.006					
	728404	0.306	728405	0.146					
	728431	0.006	728432	<0.002					

Table provided by Timmins

CERTIFICATE OF AUTHOR
William J. Lewis

As the co-author of this report for Timmins Gold Corp. entitled "NI 43-101 F1 Technical Report on the Preliminary Feasibility Study for the San Francisco Gold Project in Sonora, Mexico" dated March 31, 2008, I, William J. Lewis do hereby certify that:

1. I am employed by, and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, tel. (416) 362-5135, fax (416) 362-5763, e-mail wlewis@micon-international.com;
2. This certificate applies to the technical report titled "NI 43-101 F1 Technical Report on the Preliminary Feasibility Study for the San Francisco Gold Project, Sonora, Mexico", dated March 31, 2008;
3. I hold the following academic qualifications:

   B.Sc. (Geology) University of British Columbia 1985

4. I am a registered Professional Geoscientist with the Association of Professional Geoscientists of Manitoba (membership # 20480); as well, I am a member in good standing of several other technical associations and societies, including:

   Association of Professional Geoscientists of British Columbia (Membership # 20333)
   Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories (Membership # 1450)
   Professional Association of Geoscientists of Ontarion (Membership # 522
   The Geological Association of Canada (Associate Member # A5975)
   The Canadian Institute of Mining, Metallurgy and Petroleum (Member # 94758)

5. I have worked as a geologist in the minerals industry for 22 years;
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 4 years as an exploration geologist looking for gold and base metal deposits, more than 11 years as a mine geologist in underground mines and 5 years as a surficial geologist and consulting geologist on precious and base metals and industrial minerals;
7. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument;
8. I visited the San Francisco mine property on September 10, 2006 to review a portion of the 2006 drilling program and discuss the remaining program as well as the QA/QC program;
9. I have written the previous NI 43-101 for the mineral property that is the subject of this technical report;
10. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;
11. I am responsible for Sections 2 to 15 of this report, and portions of Sections 1, 17, 19, and 20 which I co-wrote;
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 this technical report not misleading;

Dated this 31st day of March, 2008

"William J. Lewis"

William J. Lewis, B.Sc., P.Geo.

CERTIFICATE OF AUTHOR
Michael G. Hester

As a co-author of this report for Timmins Gold Corp. entitled "NI 43-101 F1 Technical Report on the Preliminary Feasibility Study for the San Francisco Gold Project in Sonora, Mexico" dated March 31, 2008, I, Michael G. Hester do hereby certify that:

1. I am employed by, and carried out this assignment for Independent Mining Consultants, Inc. (IMC) of 3560 E. Gas Road, Tucson, Arizona, 85714, USA, phone number (520) 294-9861;
2. This certificate applies to the technical report titled "NI 43-101 F1 Technical Report on the Preliminary Feasibility Study for the San Francisco Gold Project, Sonora, Mexico", dated March 31, 2008;
3. I hold the following academic qualifications:

B.S. (Mining Engineering)	University of Arizona	1979
M.S. (Mining Engineering)	University of Arizona	1982

4. I am a Fellow of the Australian Institute of Mining and Metallurgy (FAusIMM #221108), a professional association as defined by NI 43-101. As well, I am a member in good standing of several other technical associations and societies, including:

   Society for Mining, Metallurgy, and Exploration, Inc. (SME Member #1423200)
   The Canadian Institute of Mining, Metallurgy and Petroleum (CIM Member # 100809)

5. I have worked in the minerals industry as an engineer continuously since 1979, a period of 29 years;
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. I am a founding partner, Vice President, and Principal Mining Engineer for Independent Mining Consultants, Inc. (IMC), a position I have held since 1983. I have also been employed as an Adjunct Lecturer at the University of Arizona (1997-1998) where I taught classes in open pit mine planning and mine economic analysis. I was also employed as a staff engineer for Pincock, Allen & Holt, Inc. from 1979 to 1983.
7. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument;
8. I am responsible for the mineral resource statement presented in this report and the development of the resource model on which it was based. I also developed the mine production schedule used as the basis for the mineral reserve. I visited the San Francisco mine property on February 14, 2007 to review conditions at the mine. I also visited the site on March 18, 1997 while the property was in commercial production under Geomaque. I performed mineral resource and mineral reserve estimation and mine planning work for Geomaque during 1997.
9. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;
10. I am responsible for Section 17.2 of this report;
11. 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 this technical report not misleading;

Dated this 31st day of March, 2008

"Michael G. Hester"

Michael G. Hester, FAusIMM

CERTIFICATE OF AUTHOR
Robert J. Leader

As a co-author of this report for Timmins Gold Corp. entitled "NI 43-101 F1 Technical Report on the Preliminary Feasibility Study for the San Francisco Gold Project in Sonora, Mexico" dated March 31, 2008, I, Robert J. Leader do hereby certify that:

1) I am employed by, and carried out this assignment for, Micon International Limited, Suite 205, 700 West Pender Street, Vancouver, BC, V6C 1G8, tel. (604) 647-6463, fax (604) 647-6455.

2) I hold the following academic qualifications:

   ACSM (First Class) Camborne School of Mines - 1974

   M.Sc. (Engineering) Queens University, Kingston, Ontario - 1981

3) I am a registered Professional Engineer with the Association of Professional Engineers and Geoscientists of British Columbia (Membership #13896), I am a member in good standing of other technical associations and societies, including:

   - The Canadian Institute of Mining, Metallurgy and Petroleum
   - The Institute of Materials, Minerals and Mining (IOM^3), UK

4) I have worked as a mining engineer in the minerals industry for 32 years;

5) I have read National Instrument NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 2 years working as a mining engineer in an open pit mine and over 15 years as a senior mining engineer and consultant carrying out reserves estimates, mine planning and design and cost estimating for diverse mining projects both underground and open pit;

6) In this report I am responsible for preparation of portions of sections 1; 18.1, 18.6, 18.7; and 19.

7) I visited the San Francisco mine site and reviewed the operation from September 25 to 28, 2007;

8) I have had no prior involvement with the mineral properties in question;

9) I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;

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 this report not misleading;

Dated this 31st day of March, 2008

"Robert J. Leader"

Robert J. Leader, M.Sc., P.Eng.

CERTIFICATE OF AUTHOR
Christopher A. Jacobs

As the co-author of this report on certain mineral properties of Timmins Gold Corp., in the state of Sonora, Mexico, I, Christopher A. Jacobs do hereby certify that:

1. I am employed by, and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, tel. (416) 362-5135, fax (416) 362-5763, e-mail cjacobs@micon-international.com;
2. This certificate applies to the technical report titled "NI 43-101 F1 Technical Report on the Preliminary Feasibility Study for the San Francisco Gold Project, Sonora, Mexico", dated March 31, 2008;
3. I hold the following academic qualifications:

B.Sc. (Hons) (Geochemistry)	University of Reading, UK	1980
M.B.A.	Gordon Inst. of Bus. Sci., University of Pretoria	2004

4. I am a Chartered Engineer, registered with the UK Engineering Council (registration number 369178), and a Professional Member of the Institute of Materials, Minerals and Mining (IOM[3], Membership number 46209); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum (Member No. 140649);
5. I have worked in the mining industry as an exploration and mining geologist (9 years), technical director and operations director of mining companies (11 years) and as a mineral economist (7 years); I have worked as an independent consultant with Micon since 2004.
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101;
7. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument;
8. I visited the San Francisco mine property on September 25-28, 2007 to review the technical and economic aspects of Timmins engineering work and to gather information for this study;
9. I have had no prior involvement with Timmins Gold Corp., the San Francisco project or the mineral property on which it is located;
10. I am independent of Timmins Gold Corp.;
11. I am responsible for Sections 17.3, 18.3 to 18.8 and portions of Sections 1, 19, and 20 of this report;
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 this technical report not misleading;

Dated this 31st day of March, 2008

"Christopher A. Jacobs"

Christopher A. Jacobs, CEng MIMMM

CERTIFICATE OF AUTHOR
Ian R. Ward

As a co-author of this report on certain mineral properties of Timmins Gold Corp., in the state of Sonora, Mexico I, Ian R. Ward, do hereby certify that:

1. I am employed as the President and Principal Metallurgist by, and carried out this assignment for; Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, Canada, tel (416) 362-5135; fax (416)362-5763;
2. This certificate applies to the technical report titled "NI 43-101 F1 Technical Report on the Preliminary Feasibility Study for the San Francisco Gold Project, Sonora, Mexico", dated March 31, 2008;
3. I hold the following academic qualifications:

   B.Sc. (Hons) Minerals Engineering, The University of Birmingham, U.K. 1968
4. I am a registered Professional Engineer of Ontario (membership number 48869010); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum;
5. I have worked as a metallurgist in the minerals industry and in consulting engineering companies for the last 39 years. My work experience includes processing plant operation, the management of technical and feasibility studies, management of metallurgical testwork programs and the design of numerous gold and base metal processing plants;
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101;
7. I have not visited the property which is the subject of this report;
8. I have supervised the review of metallurgical testwork and process plant production forecasts carried out by Mr Victor Bryant, IEng AIMMM presented in Sections 16 and 18.2 of this report and have taken responsibility for those sections of the report;
9. I am independent of the issuer under Section 1.4 of NI 43-101;
10. I have no prior involvement with the property or the issuer;
11. I have read NI 43-101 and I consider that this report has been prepared in compliance with the instrument;
12. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

Dated this 31st day of March, 2008

"Ian R. Ward"

Ian R. Ward, P.Eng.