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MINE DEVELOPMENT ASSOCIATES
MINE ENGINEERING SERVICES

TECHNICAL REPORT AND PRE-FEASIBILITY STUDY FOR THE CALLANQUITAS GOLD-SILVER DEPOSIT, IGOR PROJECT, REGIÓN DE LA LIBERTAD, PERÚ

Submitted to:

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CONTENTS

TABLES

FIGURES

1.0 SUMMARY (ITEM 1)

Mine Development Associates ("MDA") has prepared this technical report and Pre-Feasibility Study on the Igor project at the request of PPX Mining Corp. This report and the resource and reserve estimates have been prepared in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014. The effective date of this report is December 3, 2019.

1.1 Property Description and Ownership

The Igor project and the Callanquitas gold – silver deposit are located in northwestern Perú approximately 150 kilometers northeast of the city of Trujillo in the Huaranchal district of the Región de La Libertad. The property consists of four contiguous and overlapping mining concessions of approximately 1,300 hectares centered at approximately 781,746E and 9,153,629N UTM WGS84 Zone 17S. Land holding costs total approximately $42,600 per year.

PPX represents that there are no environmental liabilities of material importance within the Igor property and all of the necessary permits for exploration and mining activities are in place. Permitting for the proposed processing plant is in progress as of the effective date of this report.

1.2 Exploration and Mining History

Historical exploration and small-scale mining are believed to have begun during the 1500s. During the 1980s, mineralized mantos on the eastern side of Cerro Bola de Igor were exploited for gold and silver and there is a now-abandoned, 50tpd gold processing plant. PPX and its predecessor Sienna Gold Inc., began exploration of the property in 2005. Since drilling commenced in 2006, and through May of 2018, a total of 30,752.5 meters of drilling has been done in 136 core holes, primarily focused on the Callanquitas vein system. In October of 2016, PPX commenced underground test mining and bulk sampling of the Callanquitas Este vein. A total of 31,437 tonnes with an average gold grade of 8.56g Au/t were mined and toll processed at the off-site cyanide carbon-in-leach plant of Silver Cascas S.A.C. Approximately 7,028 ounces of gold have been produced for an average gold recovery of 81.2%.

1.3 Geology and Mineralization

The Igor property is centered on the southeast-plunging Igor anticline, a major fold and topographic high that largely consists of the Chimú Formation quartz arenite in its core, overlain by sandstone, siltstone, and thin black shale seams assigned to the Santa and Carhuaz Formations. Dacite porphyry dikes occur within the northwest-trending anticlinal axis and within north- and northeast-trending structural zones cutting the anticline. The entire sequence has been cut by northwest-, northeast- and north-trending faults.

The crest of the Igor anticline is transected by the approximately north-south trending, near-vertical Callanquitas fault and vein system that consists of multiple fault splays, fault breccia and gouge, hydrothermal breccia, discontinuous narrow veins, vein breccia, stockwork, and brecciated porphyritic dacite dikes. Two major splays have been named "Callanquitas Oeste" and "Callanquitas Este" veins, which are sub-vertical to steeply west dipping. Individual veins can be up to 1.5km long with thicknesses between 0.2 to 8.2m.

Gold and silver mineralization is found in breccias that originally contained variable quantities of pyrite, arsenopyrite, and minor chalcopyrite which have been strongly oxidized to depths of 200 to 300 meters below surface. Ag/Au ratios are generally low (about 10:1) in the oxide portion of the deposit, but can increase to over 100:1 at the oxide-sulfide interface.

1.4 Metallurgical Testing and Mineral Processing

Metallurgical tests carried out since 2014 indicate the Callanquitas mineralized material is amenable to cyanide heap leaching. Three stages of crushing followed by agglomeration with cement are planned, prior to stacking of the material on a re-useable leach pad. A Merrill-Crowe processing system will be constructed to produce gold doré for sale to off-site refiners or smelters. Average gold and silver recoveries are projected at about 80.0%, and 33.0%, respectively.

1.5 Mineral Resources

The stated resources, which are listed in Table 1.1, are fully diluted to 0.5 meter (width) x 2 meter (length) x 2.5 meter (height) blocks and are tabulated on a gold-equivalent ("AuEq") cutoff grade of 2.0g AuEq/t. Gold and silver grades were interpolated using inverse distance to the third power ("ID3"). Using the individual metal grades of each block, the AuEq grade was calculated based on prices of US$1,350.00 per ounce gold and US$18.00 per ounce silver, and a 3:1 recovery difference between gold and silver. The stated resources are inclusive of the reserves reported below and in Section 15.0 of this report.

Table 1.1 Callanquitas Reported Mineral Resources

Type	Class	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Oxide	Measured	109,000	5.319	19,000	61.7	215,000
	Indicated	1,361,000	4.668	204,000	113.4	4,962,000
	Meas. + Ind.	1,470,000	4.718	223,000	109.5	5,177,000
Oxide	Inferred	344,000	4.581	51,000	124.6	1,376,000
Sulfide	Inferred	269,000	2.970	26,000	159.3	1,378,000

- *CIM Definitions and Standards were followed for Mineral Resource estimates;*
- *Some apparent discrepancies are due to rounding.*

1.6 Mineral Reserves

Proven and Probable reserves have been estimated using only the Measured and Indicated resources. The reserves were calculated based on the proportion, the gold grades, and the silver grades of the Measured and Indicated resource blocks contained within each designed stope and a cutoff grade of 2.50g AuEq/t. Cutoff grade parameters were provided by PPX and were based on current actual costs, metallurgical studies, and contracts. Table 1.2 shows the fully diluted Proven and Probable reserves. The effective date of the estimated reserves is November 29, 2018. The reference point at which Mineral reserves are defined is at delivery to the crushing facility. The estimated Proven and Probable reserves are entirely within the estimated Measured and Indicated mineral resources.

Table 1.2 Igor Mine Proven and Probable Reserves

Type	Classification	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	g AuEq/t	K Ozs AuEq
Oxide	Proven	8	4.56	1	32.3	8	4.74	1
	Probable	1,022	4.09	134	104.6	3,437	4.66	153
	Proven & Probable	**1,030**	**4.10**	**136**	**104.1**	**3,445**	**4.66**	**154**

- *Proven and Probable reserves are based on Measured and Indicated resources;*
- *Reserves are reported as fully diluted and reflect a 95% extraction rate (5% ore loss);*
- *Reserves are reported based on $1,250 per ounce gold and $16.50 per ounce silver prices;*
- *Equivalent gold is reported based on metal prices and 80% recovery of gold and 33% recovery of silver;*
- *The point of reference is delivery to the crushing facility.*
- *Some apparent discrepancies are due to rounding.*

1.7 Mining Methods

The Igor mine will be accessed via two declines and a system of internal ramps. Two raises are included in the design to be used for ventilation, secondary egress, and ore passes. The planned mining method is a modified cut and fill.

1.8 Capital and Operating Costs

The total life-of-mine ("LOM") capital cost estimate is $11.7 million. This includes total initial capital estimated to be $7.1 million and sustaining costs of $4.6 million. Total operating costs are $77.3 million, or $632 per ounce of gold produced over the LOM as summarized in Table 1.3.

Table 1.3 Operating Cost Summary (USD)

	Life-of-Mine Cost K USD	Cost per Tonne Processed	Cost per Oz AuEq
Expensed Mine Development	$ 2,520	$ 2.45	$ 21
UG Mining Costs	$ 50,846	$ 49.38	$ 416
Process Costs	$ 17,640	$ 17.13	$ 144
G&A Costs	$ 2,493	$ 2.42	$ 20
Reclamation Costs	$ 3,804	$ 3.69	$ 31
Total Operating Costs	$ 77,303	$ 75.08	$ 632

The mining capital is minimized due to the use of a mining contractor that provides the required mining equipment. A contractor adjustment cost has been included in the amount of $2,374,000. The production schedule assumes processing of 350 TPD during the initial two years of mining and then ramping up to 500 TPD.

1.9 Economic Analysis

Estimated pre-tax and post-tax cash flows are shown in Table 1.4. The economic analysis is based on a gold price of $1,250 per ounce and a silver price of $16.50 per ounce. The analysis shows a strong after-tax internal rate of return ("IRR") of 59% and net present value ("NPV") of $30.1 million at 5% discount. This is reflective of the minimal continuing requirement for capital expenditure.

Table 1.4 Igor Project Cash-Flow Evaluation

Revenues	**Units**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Total**
Payable Gold (99%)	K Ozs Au	-	8	16	14	15	16	15	22	1	-	107
Payable Silver (99%)	K Ozs Ag	-	33	63	97	160	244	168	339	21	-	1,126
Equivalent Ounces Au	K Ozs AuEq	-	9	16	15	17	19	18	27	1	-	122
Gross Revenue - Au	K USD	$ -	$ 10,608	$ 19,508	$ 17,588	$ 18,274	$ 20,285	$ 19,274	$ 27,546	$ 1,168	$ -	$ 134,251
Gross Revenue - Ag	K USD	$ -	$ 543	$ 1,037	$ 1,604	$ 2,641	$ 4,023	$ 2,779	$ 5,589	$ 355	$ -	$ 18,572
Refining Cost - Au	K USD	$ -	$ (25)	$ (47)	$ (42)	$ (44)	$ (49)	$ (46)	$ (66)	$ (3)	$ -	$ (322)
Refining Cost - Ag	K USD	$ -	$ (16)	$ (31)	$ (49)	$ (80)	$ (122)	$ (84)	$ (169)	$ (11)	$ -	$ (563)
NPR Royalty	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Revenue	K USD	$ -	$ 11,109	$ 20,467	$ 19,102	$ 20,792	$ 24,138	$ 21,922	$ 32,900	$ 1,509	$ -	$ 151,938
Operating Costs												
Expensed Mine Development	K USD	$ -	$ 628	$ 492	$ 602	$ 548	$ 249	$ -	$ -	$ -	$ -	$ 2,520
UG Mining Costs	K USD	$ -	$ 5,644	$ 7,554	$ 7,705	$ 7,331	$ 8,003	$ 7,060	$ 7,550	$ -	$ -	$ 50,846
Surface Rehandle	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Costs	K USD	$ -	$ 2,016	$ 2,506	$ 2,683	$ 2,581	$ 2,740	$ 2,505	$ 2,608	$ -	$ -	$ 17,640
G&A Costs	K USD	$ -	$ 340	$ 340	$ 340	$ 340	$ 340	$ 340	$ 340	$ 113	$ -	$ 2,493
Other Costs	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation Costs	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,804	$ -	$ 3,804
Total Operating Costs	K USD	$ -	$ 8,628	$ 10,892	$ 11,331	$ 10,801	$ 11,332	$ 9,905	$ 10,498	$ 3,917	$ -	$ 77,303
Stream Equivalent Ounces	K Ozs AuEq	-	0.9	1.6	1.5	1.7	1.9	1.8	2.7	0.1	-	12.2
RIVI Finance Payments	K USD	$ -	$ 758	$ 1,397	$ 1,305	$ 1,422	$ 1,653	$ 1,500	$ 2,253	$ 104	$ -	$ 10,392
Net Operating Cash Flow After RIVI	K USD	$ -	$ 9,386	$ 12,289	$ 12,636	$ 12,223	$ 12,985	$ 11,405	$ 12,751	$ 4,021	$ -	$ 87,695
Net Operating Cash Flow	K USD	$ -	$ 1,723	$ 8,178	$ 6,466	$ 8,569	$ 11,153	$ 10,517	$ 20,149	$ (2,512)	$ -	$ 64,243
Capital Costs												
Plant & Infrastucture Direct Costs	K USD	$ 2,858	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,858
Plant & Infrastucture Indirect Costs	K USD	$ 357	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 357
Contingency - Plant & Infrastucture	K USD	$ 428	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 428
Capitalized Development	K USD	$ 629	$ 1,116	$ 1,303	$ 1,670	$ 1,287	$ 39	$ -	$ -	$ -	$ -	$ 6,045
Other Mining Capital	K USD	$ 482	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 90	$ -	$ 572
Contractor Adjustments	K USD	$ 2,374	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,374
Other Owner's Capital	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Salage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (908)	$ -	$ (908)
Total Capital Cost	K USD	$ 7,128	$ 1,116	$ 1,303	$ 1,670	$ 1,287	$ 39	$ -	$ -	$ (818)	$ -	$ 11,726
Pre-Tax Cash Flow	K USD	$ (7,128)	$ 606	$ 6,875	$ 4,796	$ 7,281	$ 11,113	$ 10,517	$ 20,149	$ (1,694)	$ -	$ 52,516
Cumulative Pre-Tax Cash Flow	K USD	$ (7,128)	$ (6,522)	$ 353	$ 5,149	$ 12,431	$ 23,544	$ 34,061	$ 54,211	$ 52,516	$ -	
Pre-Tax IRR	%	64%										
Pre-Tax NPV (5%)	K USD	$ 37,664										
Pre-Tax NPV (8%)	K USD	$ 31,037										
Pre-Tax NPV (10%)	K USD	$ 27,335										
Tax Considerations												
Net Operating Cash Flow	K USD	$ -	$ 1,723	$ 8,178	$ 6,466	$ 8,569	$ 11,153	$ 10,517	$ 20,149	$ (2,512)	$ -	$ 64,243
Use of Tax Pools												
Exploration expenses	K USD	$ -	$ -	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ -	$ (20,705)
Capex	K USD	$ -	$ -	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ -	$ (6,499)
Contractor Adjustment	K USD			$ (339)	$ (339)	$ (339)	$ (339)	$ (339)	$ (339)	$ (339)		$ (2,374)
Loss Carried Forward	K USD		$ (1,723)	$ (1,266)	$ (1,162)	$ (2,103)	$ (15)	$ (400)	$ (400)			$ (7,068)
Mine site development costs	K USD	$ -	$ -	$ (2,687)	$ (1,079)	$ (2,240)	$ (39)	$ -	$ -			$ (6,045)
Net Taxable Income	K USD	$ -	$ -	$ -	$ -	$ -	$ 6,874	$ 5,892	$ 15,524			$ 28,290
Tax Payable (29.5%)	K USD	$ -	$ -	$ -	$ -	$ -	$ 2,028	$ 1,738	$ 4,580	$ -	$ -	$ 8,345
Peruvian Royalty Tax Payable	K USD	$ -	$ 7	$ 196	$ 106	$ 216	$ 428	$ 413	$ 1,030	$ -	$ -	$ 2,395
Net After-Tax Cash Flow	K USD	$ (7,128)	$ 599	$ 6,679	$ 4,690	$ 7,066	$ 8,658	$ 8,366	$ 14,540	$ (1,694)	$ -	$ 41,776
Cumulative After-Tax Cash Flow	K USD	$ (7,128)	$ (6,529)	$ 150	$ 4,840	$ 11,905	$ 20,563	$ 28,930	$ 43,470	$ 41,776	$ -	
After-Tax IRR	%	59%										
After-Tax NPV (5%)	K USD	$ 30,075										
After-Tax NPV (8%)	K USD	$ 24,817										
After-Tax NPV (10%)	K USD	$ 21,869										
After-tax Payback	Years	1.98										

Sensitivity analysis was completed for metal prices, operating costs, and capital. These were done for both pre- and after-tax scenarios. The metal price sensitivity used a constant ratio between gold and silver prices of $1,250 gold to $16.50 silver. Table 1.5 shows the pre-tax sensitivity of the NPV and IRR to changes in gold prices from $1,100 to $1,400 in $25 increments. Table 1.6 shows the after-tax sensitivities to metal prices in the same range of gold prices.

Table 1.5 Pre-Tax Gold Price Sensitivity

Au Price	NPV (5%)	NPV (8%)	NPV (10%)	IRR
$ 1,100	$ 25,000	$ 20,122	$ 17,411	45%
$ 1,125	$ 27,111	$ 21,941	$ 19,065	48%
$ 1,150	$ 29,221	$ 23,760	$ 20,719	51%
$ 1,175	$ 31,332	$ 25,579	$ 22,373	55%
$ 1,200	$ 33,443	$ 27,399	$ 24,027	58%
$ 1,225	$ 35,554	$ 29,218	$ 25,681	61%
$ 1,250	$ 37,664	$ 31,037	$ 27,335	64%
$ 1,275	$ 39,775	$ 32,856	$ 28,989	67%
$ 1,300	$ 41,886	$ 34,675	$ 30,643	70%
$ 1,325	$ 43,996	$ 36,494	$ 32,297	73%
$ 1,350	$ 46,107	$ 38,313	$ 33,951	76%
$ 1,375	$ 48,218	$ 40,132	$ 35,605	79%
$ 1,400	$ 50,328	$ 41,951	$ 37,259	82%

Table 1.6 After-Tax Gold Price Sensitivity

Au Price	NPV (5%)	NPV (8%)	NPV (10%)	IRR
$ 1,100	$ 21,516	$ 17,326	$ 14,987	43%
$ 1,125	$ 22,950	$ 18,582	$ 16,141	46%
$ 1,150	$ 24,381	$ 19,834	$ 17,291	48%
$ 1,175	$ 25,808	$ 21,083	$ 18,439	51%
$ 1,200	$ 27,232	$ 22,330	$ 19,584	54%
$ 1,225	$ 28,655	$ 23,575	$ 20,727	57%
$ 1,250	$ 30,075	$ 24,817	$ 21,869	59%
$ 1,275	$ 31,492	$ 26,058	$ 23,008	62%
$ 1,300	$ 32,907	$ 27,297	$ 24,146	65%
$ 1,325	$ 34,319	$ 28,532	$ 25,281	68%
$ 1,350	$ 35,730	$ 29,767	$ 26,415	71%
$ 1,375	$ 37,139	$ 31,000	$ 27,548	73%
$ 1,400	$ 38,546	$ 32,232	$ 28,679	76%

Pre-tax and after-tax revenue, operating and capital cost sensitivities are shown graphically in Figure 1.1 and Figure 1.2 respectively.

Figure 1.1 Pre-Tax Sensitivity Chart

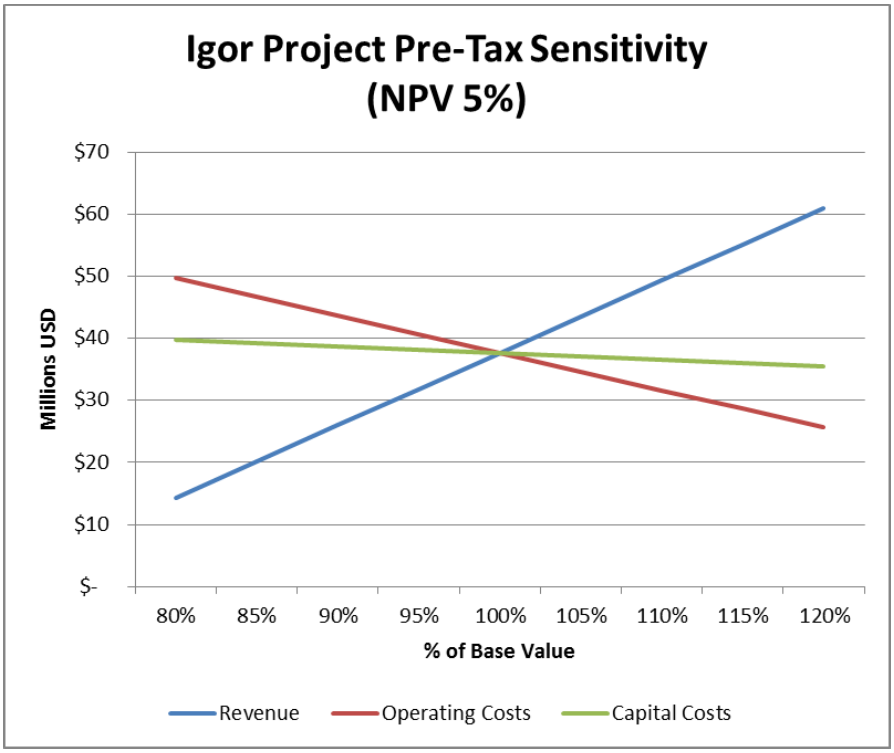
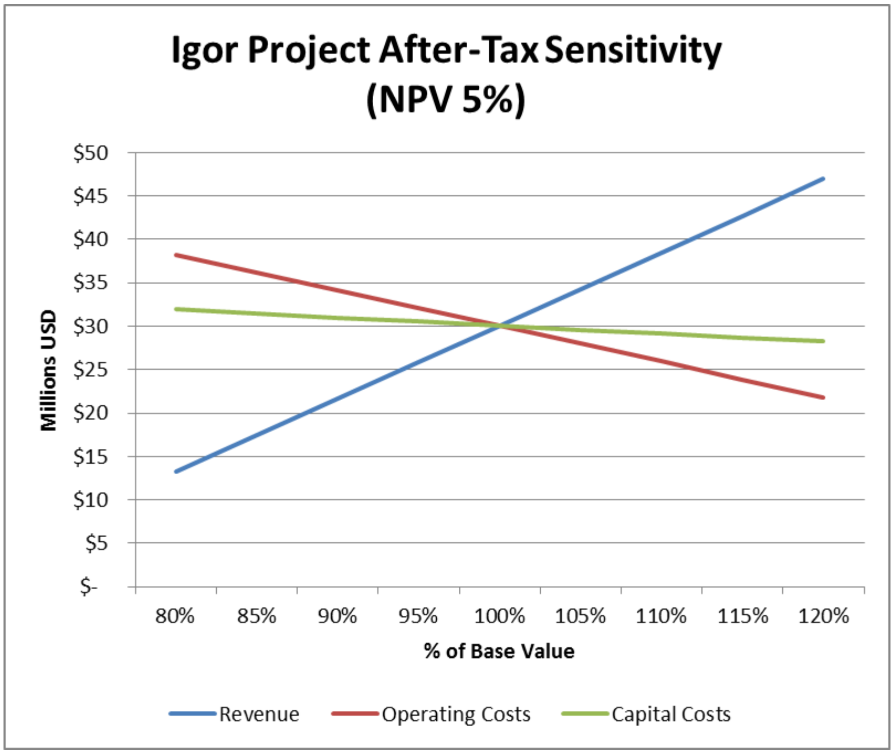


Figure 1.2 After-Tax Sensitivity Chart



1.10 Conclusions and Recommendations

The authors have reviewed the project data, including the Igor drill-hole database and metallurgical test results. The authors believe that the data provided by PPX, as well as the geological interpretations PPX has derived from the data, are generally an accurate and reasonable representation of the Igor project, and supports the use of these data in mineral resource and mineral reserve estimation.

The PFS indicates the Igor project is a viable mining and heap-leach processing operation, and work should continue on advancing the project to a production decision. The authors recommend a work program of $3.3 million as summarized in Table 1.7 and outlined below.

Table 1.7 PPX Cost Estimate for the Recommended Program

Item	Est. Cost USD
Milling and CIL Scoping Study	$ 100,000
Expansion and Exploration Drilling	$ 3,000,000
Metallurgical Testing	$ 100,000
Resource Update	$ 100,000
Total:	**$ 3,300,000**

PPX should evaluate the possible addition of a conventional milling circuit to augment or replace the heap leach facility later in the mine life. This could potentially include a capacity expansion should resource and reserve drilling identify additional areas for mining to support higher capacity at the processing facility.

Given the open nature of the Callanquitas deposit and the possible extension of the deposit to the south and at depth, additional expansion and exploration core drilling, from the surface as well as underground, should be conducted to determine the ultimate dimensions of the Callanquitas deposit and its intersection with the Portachuelos target area. Although results from drilling in the Portachuelos area were not considered in this PFS, the intersection of these mineralized structures presents an attractive exploration target that should be investigated by additional exploration drilling.

Additional metallurgical testing to better understand the deportment of silver is recommended. Sulfide-bearing gold and silver mineralization has been encountered both at depth within the Callanquitas deposit and at other target areas within the Igor Project. Should unoxidized mineralization prove to be voluminous, metallurgical testing will be required to determine an appropriate flow sheet, potential gold and silver recoveries, cost parameters, etc. to determine the economic potential of sulfide-bearing gold and silver mineralization.

The authors believe that the Callanquitas mine and Igor project is a project of merit and warrants the proposed program and level of expenditures outlined above.

2.0 INTRODUCTION AND TERMS OF REFERENCE (ITEM 2)

Mine Development Associates ("MDA") has prepared this technical report and Pre-Feasibility Study ("PFS") on the Callanquitas gold – silver deposit and Igor project, located in the Región de La Libertad, Provincia de Otuzco in northern Perú, at the request of PPX Mining Corp. ("PPX"), a Canadian company (TSX.V: PPX.V, SSE: PPX, BVL: PPX) based in Vancouver, British Columbia. This report has been prepared in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as amended.

2.1 Project Scope and Terms of Reference

The purpose of this report is to provide a technical summary and PFS for the Callanquitas gold – silver deposit and the Igor project, including an estimate of mineral reserves and an updated estimate of the mineral resources. This report incorporates results from drilling during 2017 through May of 2018, as well as information from underground exploration and test mining activities, all of which post-date the most recent technical report for the project by Davis and Sim (2013).

The mineral resources were estimated and classified under the supervision of Mr. Paul Tietz, C.P.G. and Senior Geologist for MDA. Mineral reserves were estimated and classified by Mr. Joseph Seamons, P.E. and Senior Mining Engineer for MDA. The economic analysis for the PFS was carried out by Mr. Thomas L. Dyer, P.E. and Senior Mining Engineer for MDA. Mr. Tietz, Mr. Seamons and Mr. Dyer are qualified persons under NI 43-101 and have no affiliations with PPX or their subsidiaries except that of independent consultant/client relationships. Section 13 on Mineral Processing and Metallurgical Testing, and Section 17 Recovery Methods, were prepared by Dr. Diogenes A. Uceda, P.E. Section 18 was prepared under the supervision of Mr. Ernie Burga P.E. Dr. Uceda and Mr. Burga are qualified persons under NI 43-101 and have no affiliation with PPX or their subsidiaries except that of and independent consultant/client relationship. Mineral resource estimation described in this section follows CIM standards and the disclosure and reporting requirements set forth in NI 43-101.

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

Mr. Tietz visited the Igor project on January 12 – 18, 2018. This site visit included discussing the project status, evaluating the current drilling and sampling program, conducting field verification of the drilling, investigating representative exposures in road cuts and outcrops, and reviewing the project geology. Mr. Hector Paredes (PPX Operations Manager) and Mr. Marcial Gutierrez (PPX project geologist) accompanied Tietz during the project visit. Cesar Sullca (PPX mine geologist) conducted a tour of the underground mining operation and provided excellent insights into the vein structure and morphology.

Mr. Seamons visited the Igor project on June 17 to 22, 2018 accompanied by Mr. Paredes and Mr. Grigory Pleshcheev (PPX mining engineering consultant). This site visit included touring the Callanquitas underground mine operations, surface mine facilities, plant site for future construction, and the PPX Lima office.

Site visits were not conducted by Dr. Uceda, Mr. Burga, and Mr. Dyer.

The effective date of the pre-feasibility study stated within this technical report is December 3, 2018. The date of completion of the mineral resources stated within this technical report is July 28, 2018. The database used to support the mineral resources is dated at June 29, 2018.

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

In this report, measurements are generally reported in metric units. Where information was originally reported in Imperial units, the authors have made the conversions as shown below. In cases of metallurgical tests, if the original data was reported in Imperial units, those are retained to avoid changes in precision and rounding.

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

Linear Measure

1 centimeter = 0.3937 inch

1 meter = 3.2808 feet = 1.0936 yard

1 kilometer = 0.6214 mile

Area Measure

1 hectare = 2.471 acres = 0.0039 square mile

Capacity Measure (liquid)

1 liter = 0.2642 US gallons

Weight

1 tonne = 1.1023 short tons = 2,205 pounds

1 kilogram = 2.205 pounds

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

Frequently used acronyms and abbreviations

AA	atomic absorption spectrometry
Ag	silver
Au	gold
cm	centimeters
core	diamond core-drilling method
ºC	degrees centigrade
°F	degrees Fahrenheit
ft	foot or feet
g/t	grams per tonne
ha	hectares
ICP	inductively coupled plasma analytical method
in	inch or inches
kg	kilograms
km	kilometers
kW	kilowatts
kWH	kilowatt-hours
l or L	liter
lbs	pounds
µm	micron
m	meters
m^2	square meters
m^3	cubic meters
Ma	million years old
mi	mile or miles
mm	millimeters
NSR	net smelter return
oz	ounce
ppm	parts per million
ppb	parts per billion
QA/QC	quality assurance and quality control
RC	reverse-circulation drilling method
RQD	rock-quality designation
t	metric tonne or tonnes
tpa	metric tonnes per annum
tpd	metric tonnes per day
TPD	metric tonnes per day
tph	metric tonnes per hour
tpy	metric tonnes per year

3.0 RELIANCE ON OTHER EXPERTS (ITEM 3)

The authors are not experts in legal matters, such as the assessment of the validity of mining claims, mineral rights, and property agreements in Perú or elsewhere. The authors did not conduct any investigations of the environmental, tax, or political issues associated with the Igor project, and are not experts with respect to these matters. The authors have therefore relied fully upon information and opinions provided by PPX with regards to the following:

- Section 4.2, which pertains to mineral and surface land tenure; and
- Section 4.3, which pertains to legal agreements and encumbrances;

The information in Subsections 4.2 and 4.3 are based on the *Limited Title Report on the Predios de PPX en el Perú* dated November 10, 2018 and prepared by Mrs. Liliana Garfias, Mineral Land Consultant for PPX.

The authors have relied fully upon information and opinions provided by Mr. Hector Parede, PPX's project manager, in various project 2018 email communications, for Sections 4.4, 4.5, and Section 20 which pertain to environmental permits and liabilities. .

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

4.0 PROPERTY DESCRIPTION AND LOCATION (ITEM 4)

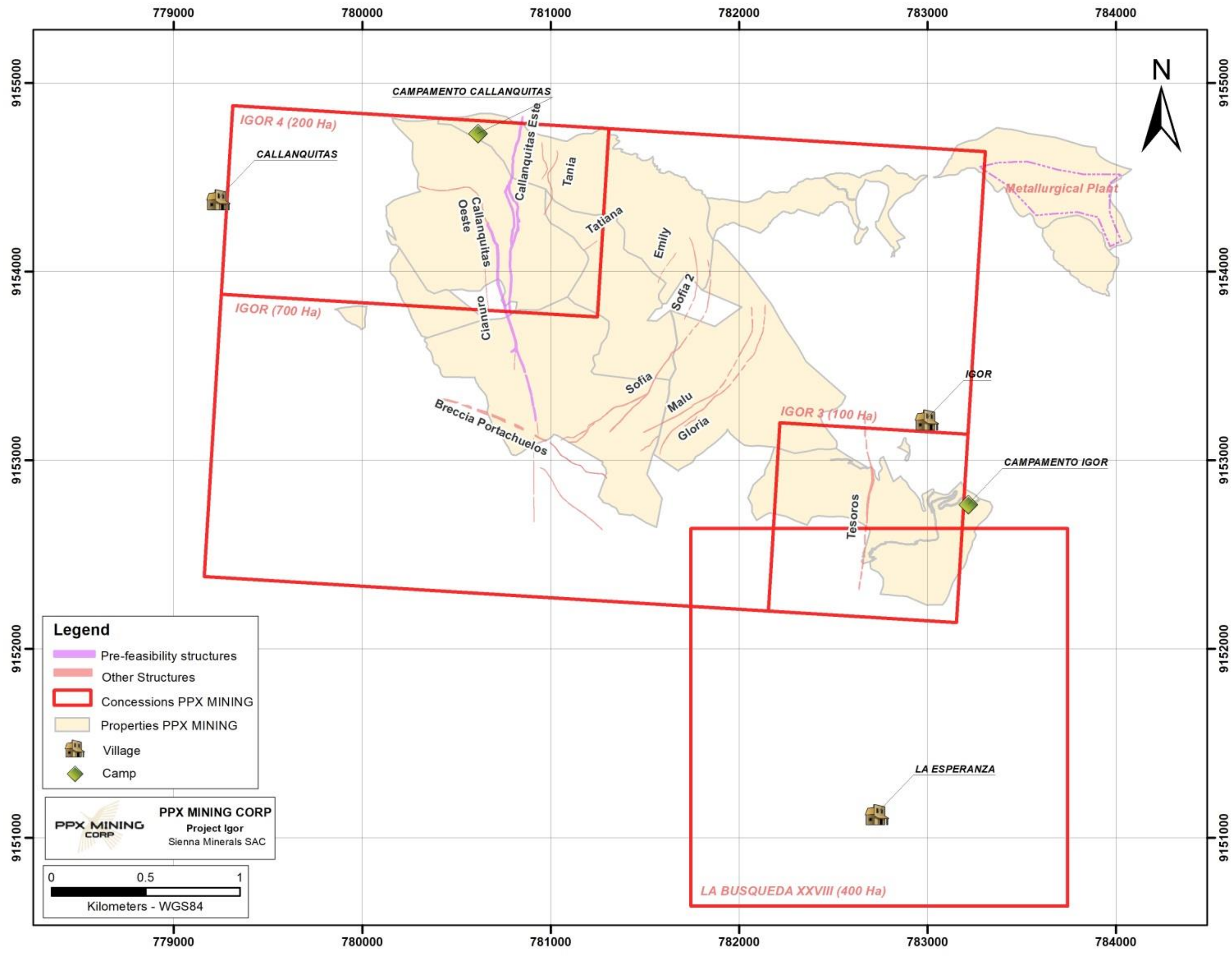
The authors are not experts in land, legal, environmental, and permitting matters and express no opinion regarding these topics as they pertain to the Igor project. . The authors do not know of any significant factors and risks that may affect access, title, or the right or ability to perform work on the property, beyond what is described in this report.

4.1 Location

The Igor project and the Callanquitas gold–silver deposit are located in northwestern Perú approximately 150 kilometers northeast of the city of Trujillo in the Huaranchal district of the Región de La Libertad (Figure 4.1), which until 2002 was known as the Departamento de La Libertad. The project is located within a mining property that consists of four contiguous and overlapping mining concessions centered at approximately 781,746E and 9,153,629N UTM WGS84 Zone 17S as shown in Figure 4.2.

Figure 4.1 Location of the Igor Project, Perú

Figure 4.2 Igor Project Property Map



Note: "Predios" shown with tan fill (Properties PPX Mining)

4.2 Land Area

The four concessions of the Igor property are summarized in Table 4.1. Accounting for their overlapping configuration (Figure 4.2), the four concessions occupy approximately 1,300 hectares.

Table 4.1 Mining Concessions of the Igor Property

Concession	Area (ha)	Recording Date	Recording Number	Owner
Igor	700	12/9/1979	20000192	Sienna Minerals
Igor 3	100	12/9/1979	11313877	Sienna Minerals
Igor 4	200	12/9/1979	11313881	Sienna Minerals
La Busqueda XVIII	400	28/11/2002	11024783	Sienna Minerals

PPX holds 100% ownership of the Igor mining concessions, which are of the metallic type, through its wholly owned subsidiary Sienna Minerals S.A.C. The mineral rights to these concessions are subject to the paramount title of the Republic of Perú, administered by the Instituto Geologico Minero Metalurgico de Perú ("INGEMMET"). Under the General Mining Law of Perú, the title holder has the right to explore, develop, and mine metallic minerals. The concessions do not expire as long as the requirements to maintain the concessions under the General Mining Law are met. Mining concessions are granted without a term or time limit, and can be held indefinitely, provided that the obligations thereunder are discharged by the mining titleholder. For the Igor property, these obligations include the payment of annual license fees that total $3,899 and have been paid through June 2019.

Holders of valid mining concessions are also obligated to invest in the exploration and exploitation of minerals and must reach certain production levels no later than by the expiration of the tenth year, calculated from the following year in which the title was granted. If minimum production levels are not reached, a series of escalating penalties are imposed. The title holder of the property can avoid the application of these penalties by annual investment of no less than 10 times the amount of the penalties. In the case of the Igor property, PPX is required to produce minerals with a value of $1,635,00 in 2018, or invest $327,000 in 2018 to avoid the application of the penalties and maintain the property in good standing.

If the minimum production is not reached at the end of 30 years since the following year in which the title concession was granted, the termination of the mining concession must be declared by the Mining and Metallurgical Geological Institute of Perú ("INGEMMET"). The thirty years are completed in 2038, according to internal calculations of INGEMMET.

Surface rights granting access to and disturbance of the surface of the Igor property for exploration and mining activities are held by PPX through their subsidiaries Agraria Huaranchal S.A.C., which collectively own 428.6 hectares of property known as "predios" within and adjacent to the Igor concessions (Figure 4.2). PPX represents that these predios provide adequate access and surface rights for the potential exploitation of the Callanquitas deposit as envisioned in the PFS presented in this report. Municipal taxes must be paid to maintain ownership of the predios. These taxes are estimated to be approximately $6,003 per year.

4.3 Agreements and Encumbrances

There are no royalties on mineral production from the Igor property, payments required for surface access rights, or other underlying agreements relating to any of the claims other than the municipal property taxes and annual concession fees summarized above. The total estimated annual concession and land holding costs are $42,602 (maximum amount with no activity in the concessions) as summarized in Table 4.2.

Table 4.2 Summary of Annual Concession and Land Holding Costs

Item	Estimated Annual Cost ($)
Annual Concession Fees	$ 3,899
Investment in Lieu of Required Minimum Production Values	$ 32,700
Municipal Property Tax	$ 6,003
Total	***$ 42,602***

4.4 Environmental Liabilities

PPX represents there are no environmental liabilities of material importance associated with the disturbances caused by exploration and mining activities within the Igor property.

4.5 Environmental Permitting

Several environmental permits are required and have been obtained for exploration at the Igor project as follows:

- Estudio de Impacto Ambiental Semidetallado (“EIAsd”) which is valid until the end of 2019 and is regulated by the Dirección General de Asuntos Ambientales Mineros (“DGAM”);
- Aprobación de la Modificación del EIAsd de Exploración del Proyecto IGOR (2011);
- Aprobación del ITS de la Modificación del EIAsd de Exploración del Proyecto IGOR (2014);
- Aprobación del Declaracíon de Impacto Ambiental (“DIA”) for exploration within the Igor 4 concession (2017);
- Permiso de Uso de Agua for exploration activities, regulated by the Autoridad Nacional del Agua (“ANA”) (2017-2019);
- Certificado de tratamiento de aguas servidas from the Dirección General de Salud Ambiental e Inocuidad Alimentaria (“DIGESA”) (2010); and
- Certificado de Identificación de Restos Arqueológicos (“CIRA”), regulated by the Ministerio de Cultura.

For the underground mine at the Callanquitas deposit, the following environmental permits are required and have been obtained:

- Aprobación de la DIA (2015);
- Aprobación del CIRA (2016);
- Aprobación del Plan de Minado e inicio de actividades (2016);
- Aprobación del Certificado de Operación Minera (“COM”) (2016);

- Aprobación del Permiso de uso de explosivos (2016); and
- Aprobación del Licencia de uso de aguas para una actividad industrial de explotación (2017).

Permitting for the proposed processing plant, which is to be constructed in phases, is in progress as of the effective date of this report. Table 4.3 summarizes the status of the permitting process.

Table 4.3 Awarded Permits

Ref	Description	Awarded Date
1	Approval of Terms of Reference for Processing Plant Semi-detailed Environmental Impact Assessment (EIAsd), issued by MEM, Ministry of Mines and Energy.	24-10-2017
2	CIRA certificate (Issued by INC, National Cultural Institute), this confirms the non-existence of archeological remains within the proposed plant area.	21-09-2018
3	Water availability permit (issued by ANA, National Water Authority). Allocating the water required to run Processing Plant (120 m^3 / day).	25-10-2018
4	EIAsd approval (Issued by DREM, Regional Directorate of Mines and Energy-Trujillo). Confirming compliance to all Peruvian environmental laws applied to Processing Plants.	19-11-2018
5	PMA approval (Issued by INC), this is an Environmental Monitoring Plan (Plan Monitoreo Ambiental - PAM) that Processing Plant can use to start work within the CIRA approved area. The PMA also assigns an (approved) archeologist to supervise the work.	07-11-2018
6	Community agreement, Caserio Igor - Processing Plant, covering the life of the project (Processing Plant), initially estimated for 8 years.	20-11-2018

Permits pending approval are summarized in Table 4.4.

Table 4.4 Pending Approval

Ref	Description	Expected Date
A	Permit to build the Water system, intake and distribution (issued by ANA), based on awarded Water Availability (Ref 3 above).	04-01-2019
B	Permit to use water during construction (issued by ANA), as from the start date of A.	16-01-2019
C	Permit to build and initiate operations of Process Plant, referred as F2 and issued by DREM Trujillo.	08-01-2019

PPX represents that it has obtained the required environmental and other permits needed for the exploration and mining activities proposed in the PFS of this report. Permitting for the processing plant is underway, but not yet completed as of the effective date of this report.

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

The information summarized in this section is drawn from Davis and Sim (2013), and derived from publicly available sources, as cited. The authors have reviewed this information and believe this summary is materially accurate.

5.1 Access to Property

From the city of Trujillo there are two land routes for access to the property as shown in Figure 5.1. Both routes consist of a combination of paved and unpaved roads. The northern route through the towns of Chicama, Sausal, 9 de Octubre and Lucma totals approximately 145 kilometers. The route through Otuzco and Huaranchal is approximately 157 kilometers and requires about two more hours of travel time (Davis and Sim, 2013). The city of Trujillo has commercial airline service to the cities of Lima and Cuzco, Perú, and may be reached in approximately eight hours driving time from Lima via the Pan-American Highway.

Figure 5.1 Access Map for the Igor Project, Perú

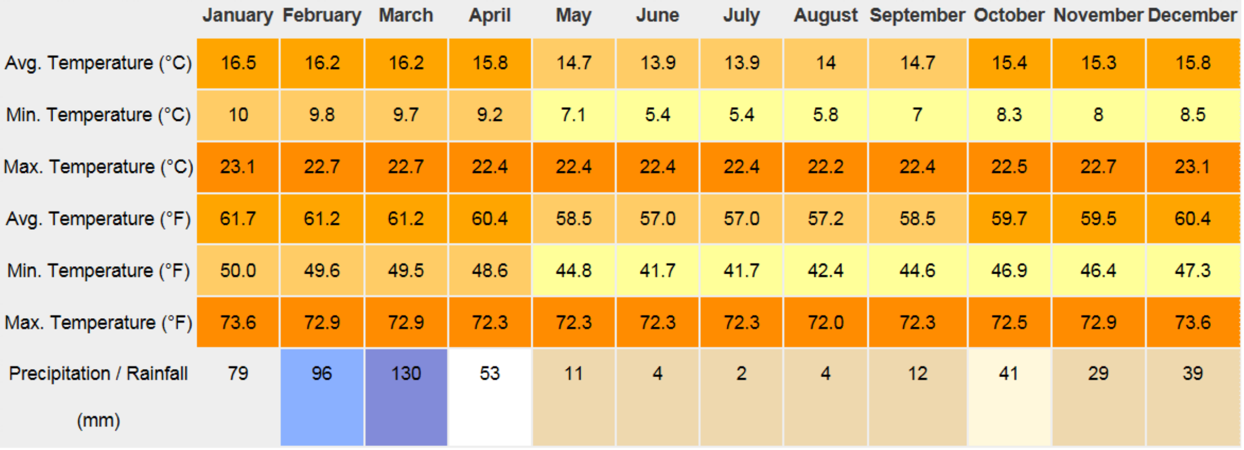
The surface rights as described in Section 4.3 are sufficient for the mining and exploration activities proposed in this report.

5.2 Climate

Temperature and precipitation data for the town of Huaranchal, 5.5 kilometers south of the Igor property, are summarized in Table 5.1. The area is characterized by cool dry winters and temperate, rainy summers. Because the Igor property is approximately 700 meters to 1,700 meters higher in elevation than Huaranchal, minimum temperatures during the Andean winter can reach below 0°C, and maximum temperatures are generally lower than those shown in Table 5.1.

Mining and exploration can be conducted year-round.

Table 5.1 Average Climate Conditions at Huaranchal, Perú
(from https://en.climate-data.org/location/875088/)



	January	February	March	April	May	June	July	August	September	October	November	December
Avg. Temperature (°C)	16.5	16.2	16.2	15.8	14.7	13.9	13.9	14	14.7	15.4	15.3	15.8
Min. Temperature (°C)	10	9.8	9.7	9.2	7.1	5.4	5.4	5.8	7	8.3	8	8.5
Max. Temperature (°C)	23.1	22.7	22.7	22.4	22.4	22.4	22.4	22.2	22.4	22.5	22.7	23.1
Avg. Temperature (°F)	61.7	61.2	61.2	60.4	58.5	57.0	57.0	57.2	58.5	59.7	59.5	60.4
Min. Temperature (°F)	50.0	49.6	49.5	48.6	44.8	41.7	41.7	42.4	44.6	46.9	46.4	47.3
Max. Temperature (°F)	73.6	72.9	72.9	72.3	72.3	72.3	72.3	72.0	72.3	72.5	72.9	73.6
Precipitation / Rainfall (mm)	79	96	130	53	11	4	2	4	12	41	29	39

5.3 Physiography and Vegetation

The property includes extremely rugged to rolling mountainous terrain. Elevations in the area range between 2,800 meters and 3,800 meters, with abrupt topographic transitions. Slopes on the southern parts of the Igor concession and the La Busqueda XVIII concession are less severe. There are sites within the concessions and surface holdings potentially suitable for mine infrastructure such as waste rock and tailings storage, heap leach pads, and a process plant area.

At lower elevations of the property, vegetation consists of mixed brush, grasses and trees. The upper elevations are mainly rocky alpine grasslands with sparse brush, such as at Cerro Igor (Figure 5.2).

5.4 Local Resources and Infrastructure

Electrical power from the regional system is available within the project area, but is insufficient for the project's proposed requirements. Diesel-powered generators are presently in place and supply electrical power for the underground workings at the Callanquitas mine. As discussed in Section 4.2, PPX controls sufficient surface rights for access, exploration and mining operations.

Figure 5.2 View of Cerro Igor and Adjacent Ridge

The region includes several towns and small cities with an adequate supply of skilled and unskilled labor suitable for a mining operation. The villages of Callanquitas and Igor provide a source for manual labor; this local resource complements PPX's community social programs. Engineering and industrial equipment and services, fuel, banking, telecommunications and transportation services are available in the city of Trujillo.

6.0 HISTORY (ITEM 6)

The information in this section is summarized from Davis and Sim (2013), as well as other sources as cited. Mr. Tietz has reviewed this information and believes this summary is materially accurate and assumes responsibility for the text.

6.1 Exploration History

The Igor project's name derives from the son of a Russian immigrant who explored the area in the 1880s, but there are shallow underground workings believed to date from as early as the 1500s. The Igor concessions were recorded in 1979. In the 1980s, mineralized mantos on the eastern side of Cerro Bola de Igor were exploited for gold and silver and there is a now-abandoned, 50tpd gold processing plant located close to the village of Igor.

At some point the Igor concessions were acquired by Sr. Samuel Guia. In 1998, Sr. Guia optioned the property to Rio Amarillo Gold Ltd., but the option was not exercised and ownership of the property reverted to Mr. Guia. In 2001, Matrix Gold Mining Investment S.A. ("Matrix") entered into a purchase option agreement with Mr. Guia for the property. In June of 2005, Sienna Minerals S.A.C. ("Sienna"), the Peruvian, wholly-owned subsidiary of Sienna Gold Inc., purchased a 60% interest in the Igor concession from Matrix. Sienna purchased the remaining 40% interest from Matrix in 2006 and purchased the Busqueda XVIII concession from Vena Resources in August of 2010. Sienna Gold Inc. changed its name to Peruvian Precious Metals Corp ("Peruvian") in July of 2013. In August of 2016, Peruvian changed its name to PPX Mining Corp, retaining 100% ownership of the property through its subsidiary Sienna. Exploration conducted within the Igor property by Sienna Gold Inc., Peruvian and PPX are discussed in Section 9.0 of this report.

6.2 Historic Mineral Resource Estimates

There have been no historical resource estimates completed on the Callanquitas property.

6.3 Past Production

Mr. Tietz is not aware of records of mineral production from the Igor property during Spanish colonial and more recent times, prior to the ownership of PPX. There currently is small-scale artisanal mining at Domo and Tesoros though Mr. Tietz is not aware of any production records associated with this mining activity. Underground test mining and bulk sampling by PPX are discussed in Section 9.4.

7.0 GEOLOGIC SETTING AND MINERALIZATION (ITEM 7)

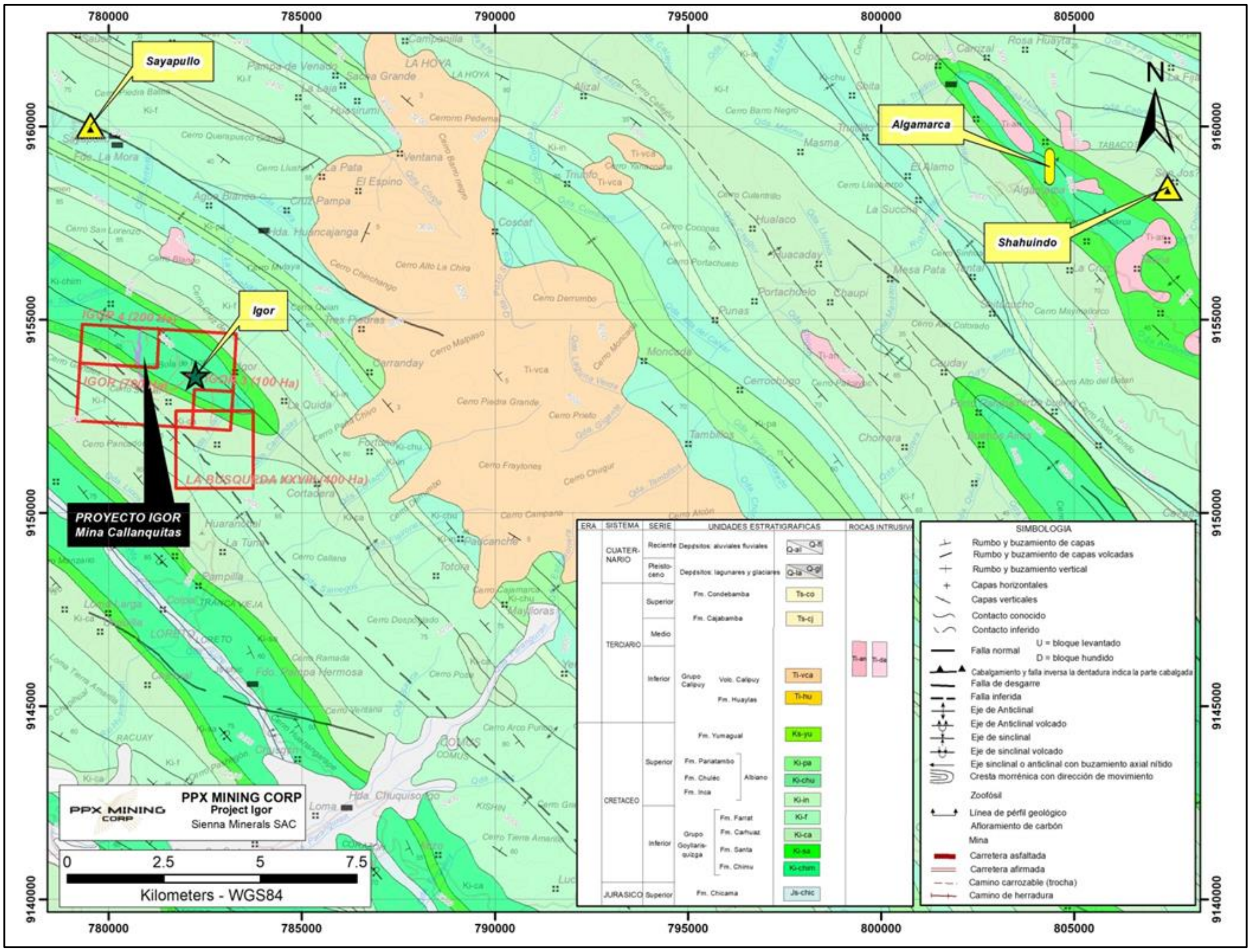
The information presented in this section of the report is derived from discussions with PPX personnel with further details taken primarily from Davis and Sim (2013), and other sources, as cited. Mr. Tietz believes this summary accurately represents the Igor project geology and mineralization as it is presently understood.

7.1 Regional Geologic Setting

The Igor project is situated in the Cordillera Occidental of the Peruvian Andes, which is part of the Andean fold and thrust-fault belt and has undergone multiple episodes of east-northeast vergent compressional deformation related to Cenozoic convergence of the Nazca and South American plates. The regional geology is characterized by multiple northwest-southeast striking anticlines and synclines that are largely upright and have been cut by southwest-dipping thrust faults. Marine shale and intercalated sandstone and quartzite sequences of the Upper Jurassic Chicama Formation are the oldest rocks exposed in the cores of the anticlines (Figure 7.1).

The Chicama Formation is overlain by the Lower Cretaceous Chimú Formation, consisting of quartz arenite with intercalated thin black shale and local coal seams, which in turn is overlain by several marine sedimentary units of lower Cretaceous age (Figure 7.1). Largely subaerial volcanic rocks of the mid-Eocene to Miocene Calipuy Group unconformably overly the Mesozoic sedimentary rocks and a number of Paleogene and Neogene stocks and dikes of felsic to intermediate compositions have intruded the major folds.

Figure 7.1 Regional Geology of the Igor Project Area
(from PPX, 2018)

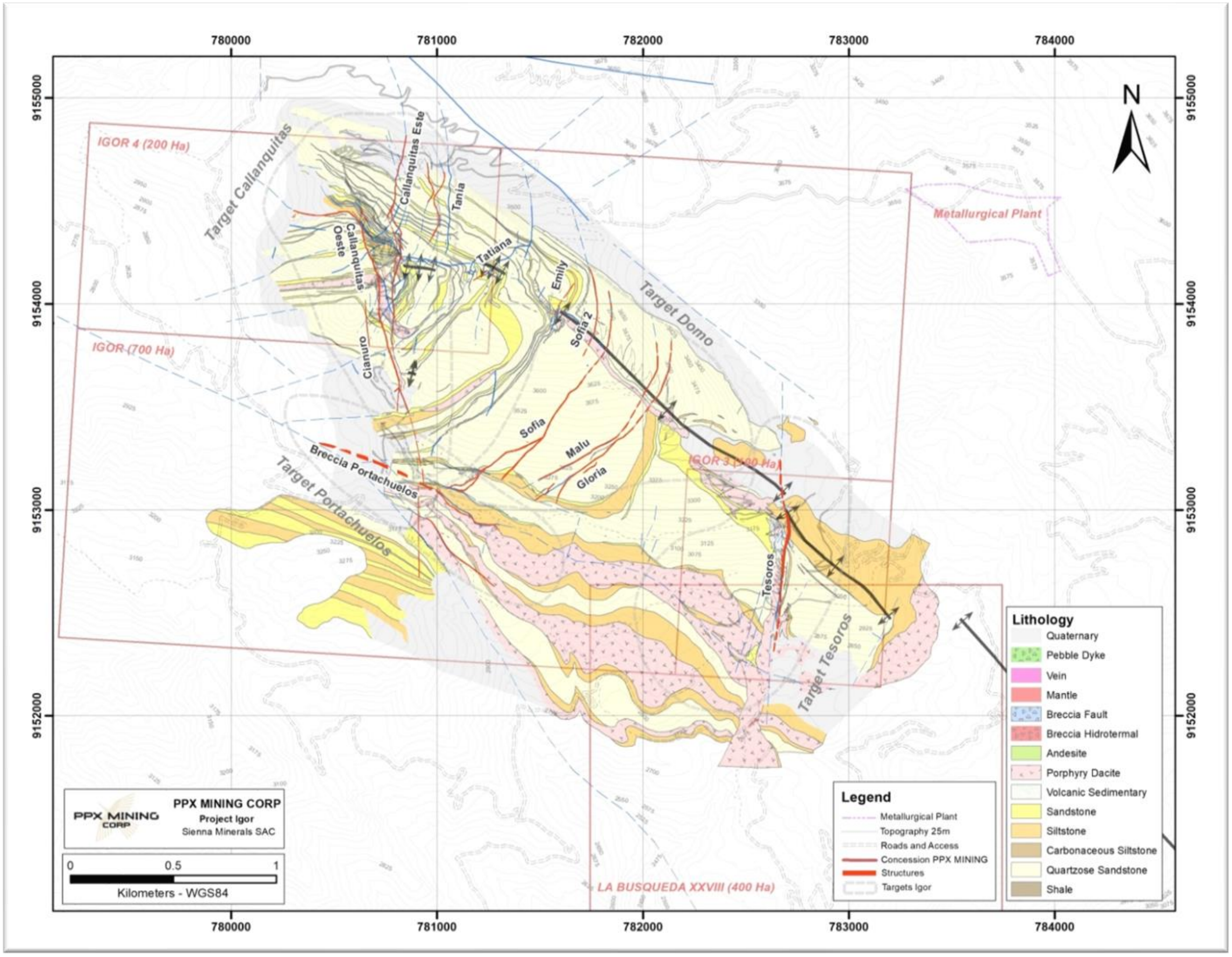


7.2 Property Geology

The Igor project is centered on the informally named, northwest-striking, southeast-plunging Igor anticline, a major fold and topographic high that largely consists of the Chimú Formation quartz arenite in its core. Overlying the Chimú, and occurring on the limbs of the anticline, are a clastic sedimentary sequence of interlayered, lensing sandstone, siltstone, and thin black shale seams, that is assigned to the Santa and Carhuaz Formations.

A dacite porphyry stock (locally named the Huevo de Condor stock) and sills crop out within the southwestern limb of the anticline. Dacite porphyry dikes, thought to be emanating from the stock, occur within the northwest-trending anticlinal axis and within north- and northeast-trending structural zones cutting the anticline. The dacite intrusions are pre-mineral although there appears to be a spatially association with mineralization. It is likely that the same structural zones which were favorable for intrusive emplacement were amenable to later hydrothermal fluid movement and subsequent mineralization. Minor granodiorite is present in some of the deep drill holes. The entire sequence has been cut by a variety of northwest-, northeast- and north-trending faults (Figure 7.2).

Figure 7.2 Igor Property Geologic Map
(from PPX, 2018)



The crest of the Igor anticline is transected obliquely by the approximately north-south trending, near-vertical Callanquitas fault and vein system (Figure 7.2) that is at least 2.4 kilometers in length and as much as 8.2 meters in width. It has been referred to as the "Callanquitas Structure" by Davis and Sim (2013) and consists of multiple fault splays, fault breccia and gouge, hydrothermal breccia, discontinuous narrow veins, vein breccia, stockwork, and brecciated porphyritic dacite dikes. Two major splays have been named "Callanquitas Oeste" and "Callanquitas Este" (Figure 7.2). Because the individual Callanquitas mineralized zones have an over-all vein-like morphology, and gold-silver mineralization within it is not as wide as the entire host fault zone, Mr. Tietz prefers the term "Callanquitas vein" when referring to the gold-silver deposit.

7.3 Mineralization

Igor project mineralization occurs within four primary target areas: Callanquitas, Domo, Tesoros, and Portachuelos (see Figure 7.2). Mineralization within the property often occurs at the transition zone from Chimú up into the Santa and Carhuez rocks. There is a spatial association of dacite porphyry dikes emplaced within structural breccias with both structural and manto-style mineralization.

The Callanquitas area, which is on the northwest side of the Igor anticline, is the focus of the current resource estimate and pre-feasibility technical analyses. Domo and Tesoros, within the east and southeast sides of the anticline, respectively, have had limited drilling and both are areas of minor artisanal mining activity. Portachuelos is an exploration target on the southwest limb of the anticline on trend south of Callanquitas.

7.3.1 Callanquitas Veins

The Callanquitas veins (Callanquitas Este, Callanquitas Oeste and Cianuro) occur within a north-south striking zone of multiple, anastomosing to divergent faults that extends over a length of 2,400 meters. Individual veins can be up to 1.5km long with thicknesses between 0.2 to 8.2m. The Callanquitas structures are open along strike to the south and north and also at depth.

The Callanquitas mineralization is considered an intermediate-sulfidation type, consisting of mineralized structural breccias that originally contained variable quantities of pyrite, arsenopyrite, and minor chalcopyrite. These sulfide minerals were subsequently strongly oxidized to significant depths. The primary mineralized structures (Callanquitas Este and Oeste structures), are nearly vertical and steep westerly dips are common.

The structures cut through a sequence of sandstone, siltstone and quartzite. The silicified sandstone and quartzite units were more amenable to brecciation and therefore more favorable for migrating hydrothermal fluids and mineralization. The dacite intrusive rocks are generally argillically-altered and although the mineralized structures cut through the dacite, the general lack of brecciation and development of open-space renders the intrusions a poor host for mineralization.

Detailed underground mapping by PPX, along with a re-logging of early core holes, indicates multiple stages of veining, brecciation, alteration, and mineralization. The initial event appears to be emplacement of quartz veins within faults, followed by multiple stages of hydrothermal brecciation (“HBx”) with associated pervasive silicification, which was then cut by sulfide veinlets. The last stage is a fault gouge or tectonic breccia (“TBx”) usually occurring adjacent or cross-cutting the HBx. The highest gold grades are associated with the early quartz and HBx while increased silver occurs with the late-stage sulfide veinlets. The TBx is argillized and appears to be mostly post-mineral in the paragenesis, though it often contains low-grade gold in the range of 0.5 to 1.0g Au/t. Gold within the TBx is believed to be due to the presence of mineralized clasts within the argillized fault gouge, though there are exposures underground of TBx with oxidized veinlets that can assay up to multiple grams of gold. It is possible that there are multiple TBx events with early-stage faulting prior to the formation of silica-sulfide veinlets.

Drilling indicates that mineralization in the Callanquitas Este and Oeste veins occurs over a 1,400 meters north-south strike length and extends to depths of up to 500 meters below the topographic surface. Oxidation within the mineralized structures occurs to depths of 200 to 300 meters below surface. Ag/Au ratios are generally low (about 10:1) in the oxide portion of the deposit, but can increase to over 100:1 at the oxide-sulfide interface. The change in Ag/Au ratios is not just related to supergene enrichment, but also represents the occurrence of late-stage, silver-rich, silica-sulfide veinlets which cut the more gold-rich hydrothermal breccias.

7.3.2 Domo

Domo mineralization occurs within a zone of bedding-parallel mantos and perpendicular veins along the crest of the southeast end of the Igor Anticline. Gold and silver mineralization is controlled by a series of northeast-southwest trending faults and occurs as oxidized pyritic breccias with variable silicification.

7.3.3 Tesoros

Tesoros mineralization consists of gold-bearing veins, breccias, and mantos along a major north-northeast striking fault that has localized the emplacement of dacitic porphyries and pebble dikes. Note: The gold and silver mineralization at Domo and Tesoros is described in NI 43-101 Technical Report *Updated Technical Report and Resource Estimate of the Igor Miner Project* (Henkle and Lytle, 2008).

7.3.4 Portachuelos

Portachuelos lies along the southern extension of the Callanquitas structure zone in an area where the north-south Callanquitas structures, and also the northeast-trending Domo structures, trend to the south and southwest into the western portion of the Huevo de Condor dacite porphyry stock. It is an early stage exploration target which is currently the focus of PPX exploration drilling.

8.0 DEPOSIT TYPES (ITEM 8)

Field evidence suggests that the Igor project mineralization belongs to the epithermal type of gold-silver deposit (Guilbert and Park, 1986). In general, epithermal gold-silver deposits are composed of structurally or stratigraphically controlled disseminations or veins that form in a shallow environment (less than or about 1.5 km) and are hosted by volcanic or sedimentary rocks. The mineralization is dominated by gold and silver but can contain variable amounts of copper, lead, and zinc.

Epithermal gold deposits can be placed on a continuum between:

- High sulfidation, characterized by quartz-kaolinite-alunite alteration, enargite and other sulfo-salt minerals, or high sulfur oxidation state (Ashley, 1982; Hedenquist, 1987; Bonham, 1988); and
- Low sulfidation, characterized by adularia-sericite alteration.

The setting, alteration, and mineralization characteristics of the Callanquitas deposit are consistent with an intermediate- or high-sulfidation epithermal system (Hedenquist et al., 2000). The deposit lacks the characteristic enargite, vuggy residual quartz and alunite of the high-sulfidation systems and does not have the adularia diagnostic of the low-sulfidation systems.

The Igor project mineralization occurs within mineralized structural breccias that originally contained variable quantities of pyrite, arsenopyrite, and minor chalcopyrite. These sulfide minerals were subsequently strongly oxidized to significant depths.

The Igor project lies in a similar geologic setting, and has a similar intermediate-sulfidation mineral style, as Tahoe Resources’ Shahuindo gold-silver mine located about 25 kilometers east of Igor (see Shahuindo location on Figure 7.2). Mineralization at both properties occur within the Lower Cretaceous sedimentary package, and while the Shahuindo mineralization is being exploited as a bulk-tonnage, open-pit deposit, the Shahuindo mineralization does exhibit strong structural control within the siltstone/sandstone host rocks. Dacite porphyry stocks and sills occur at both properties and these appear to be spatially, if not genetically, related to the gold-silver mineralization.

9.0 EXPLORATION (ITEM9)

The information in this section is derived from multiple sources as cited. Mr. Tietz has reviewed this information and believes it materially reflects the work done on the property by the issuer of this report.

PPX and its predecessors Peruvian and Sienna have carried out surface and underground geologic mapping and sampling, geophysical surveys, and drilling since 2005. Drilling is summarized in Section 10.0. Underground test mining and bulk sampling began in 2017 and is summarized in Section 9.4.

9.1 Geologic Mapping

PPX geologists and consultants mapped the surface geology of the Igor concession and portions of the Busqueda XVIII concession at a scale of 1:5,000 during 2006 through 2018. The central portion of the Igor concession was also mapped at a scale of 1:1,000 during that time. A total of approximately 3,240 meters of underground workings also have been mapped within the Callanquitas mine and at Tesoros. The underground mapping has been at a minimum scale of 1:250.

9.2 Geochemical Sampling

Henkle and Lytle (2008) reported that in 2005 through 2007 a total of approximately 3,100 surface geochemical samples "of all kinds" were collected from the property and analyzed for gold, silver and other elements. By the middle of 2013, a total of approximately 4,200 surface samples were collected (Davis and Sim, 2013), including outcrop, trench and "old working samples".

Maps compiled by PPX in May of 2018 indicate there are geochemical data from 252 soil samples, 1,911 surface rock samples including trench and channel samples, and 2,254 underground samples which are mainly channel samples. The results for gold and silver are summarized with the thematic maps shown in Figure 9.1 and Figure 9.2.

Figure 9.1 Gold in PPX Surface and Underground Samples

(from PPX, 2018)

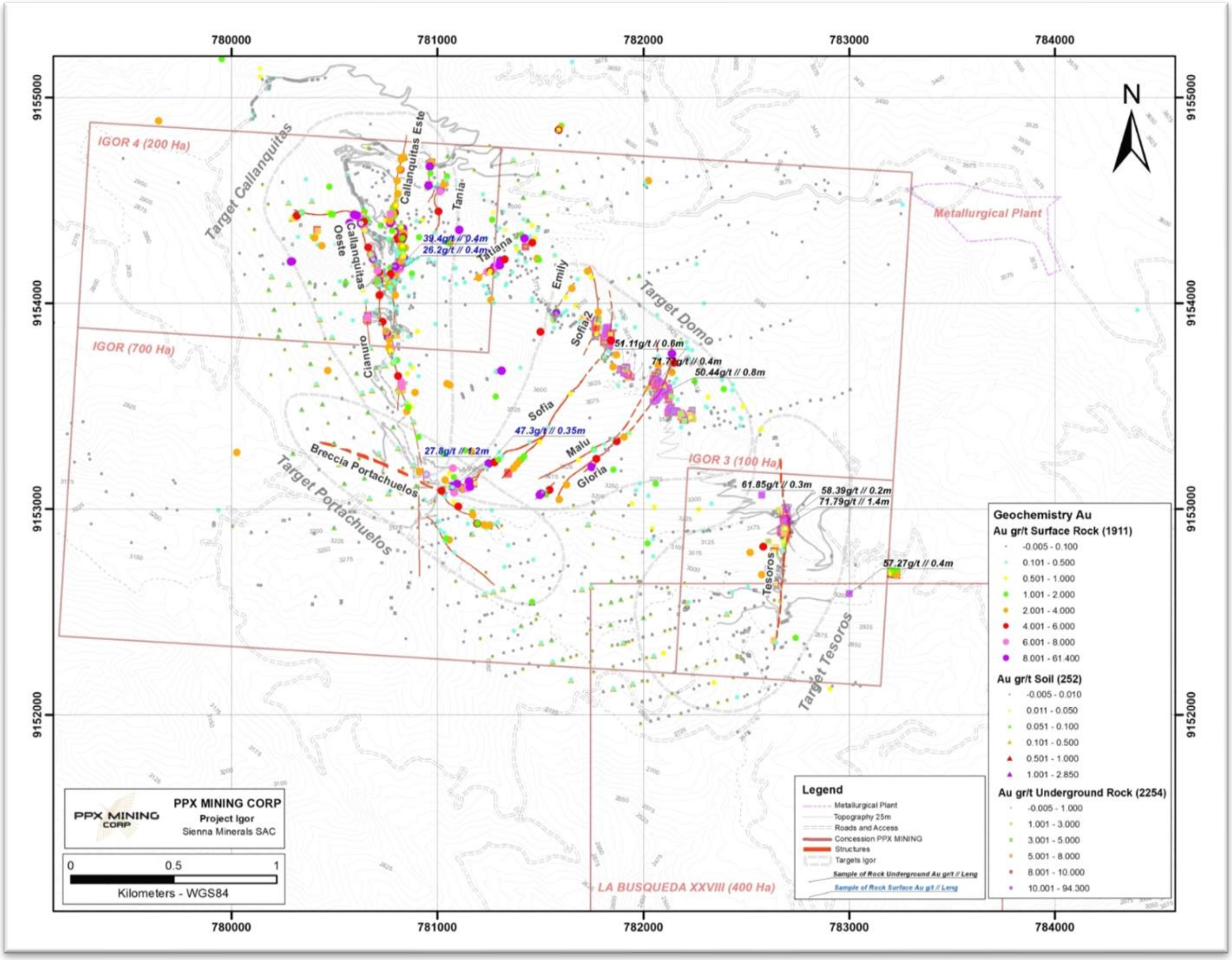
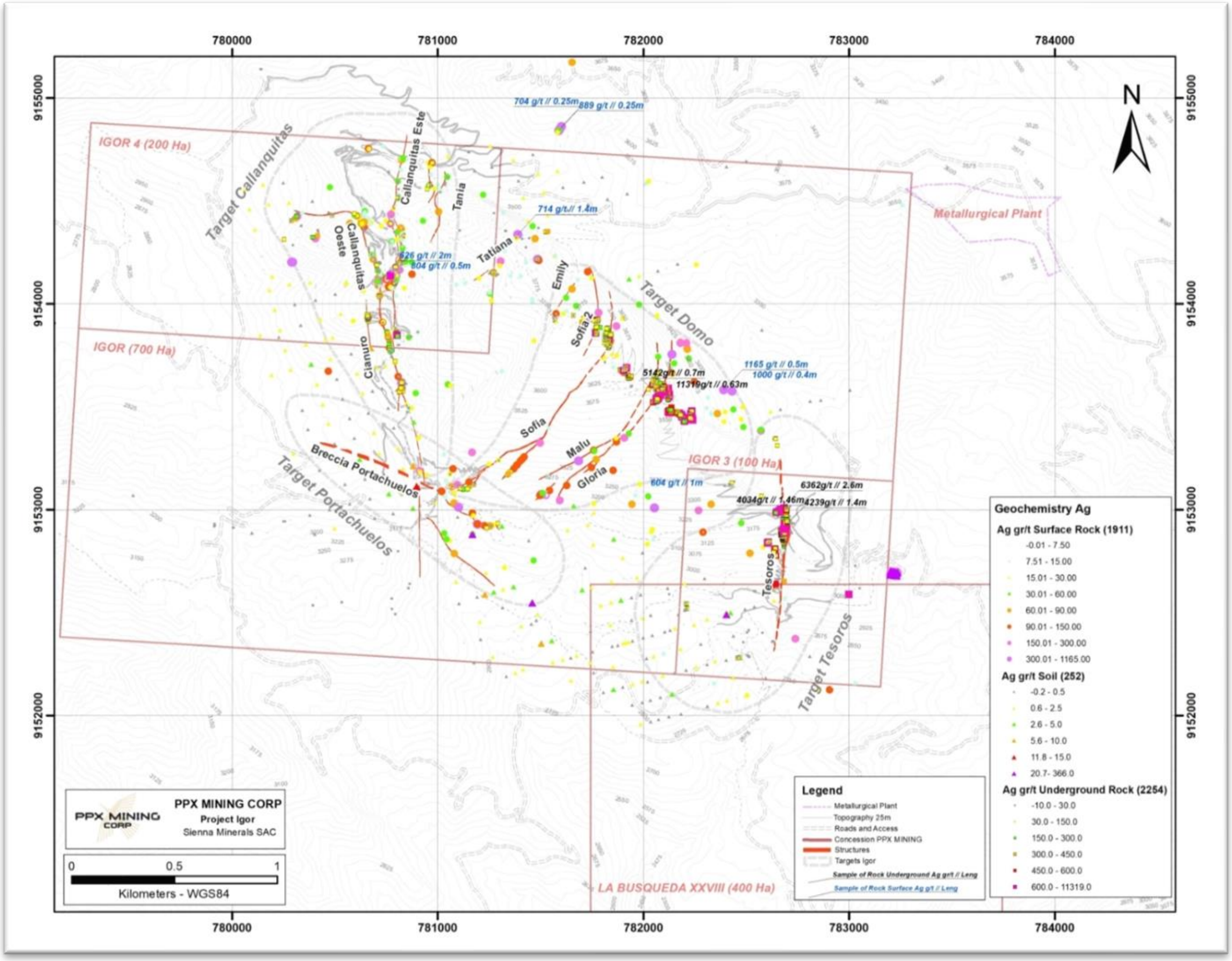


Figure 9.2 Silver in PPX Surface and Underground Samples
(from PPX, 2018)



9.3 Geophysical Surveys

Two geophysical surveys have been conducted at the Igor project. An induced potential and resistivity (“IP/Res”) survey was carried out in 2006 by Geofisica Consultores SRL over the central part of the property. The IP/Res survey totaled 14.3 line-kilometers on nine lines with an azimuth of N75°E. Figure 9.3 and Figure 9.4 show the results. A major area of elevated chargeability and low resistivity corresponds to the altered porphyritic dacite that crops out within the southwest part of the Igor anticline. The chargeability “high” and resistivity “low” extends to the northwest to the Portachuelos area and north along the Callanquitas fault zone. These geophysical trends mimic the location of dacite dikes and sills encountered by drilling beneath cover and at depth along the Callanquitas zone.

In 2007, a ground magnetic survey was performed by Fugro Ground Geophysics Pty Ltd. A total of 55.7 line-kilometers were surveyed along an azimuth of N75°E.

Figure 9.5 shows the results. A modest magnetic high corresponds to the dacite intrusion.

Figure 9.3 Chargeability Map, 2006 IP/Resistivity Survey
(from PPX, 2018)

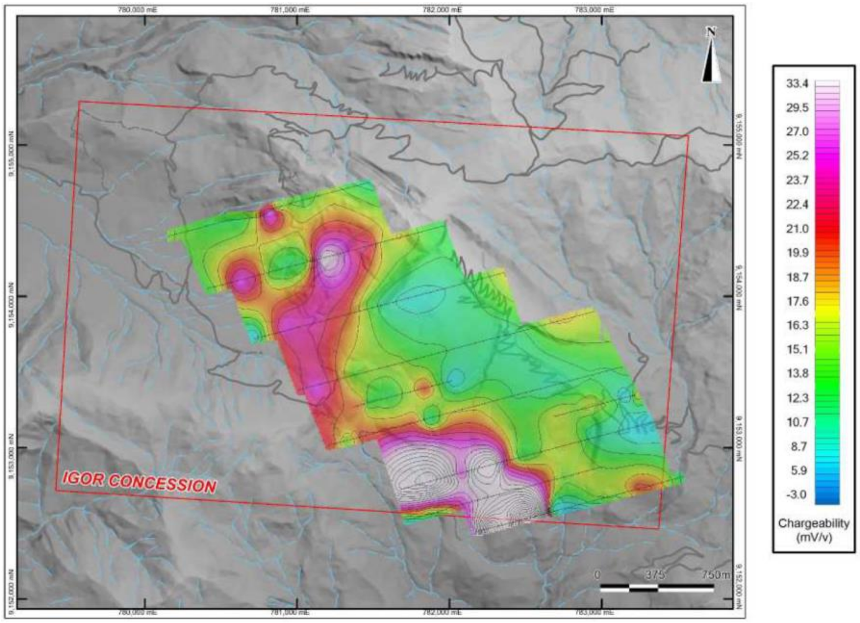


Figure 9.4 Resistivity Map, 2006 IP/Resistivity Survey
(from PPX, 2018)

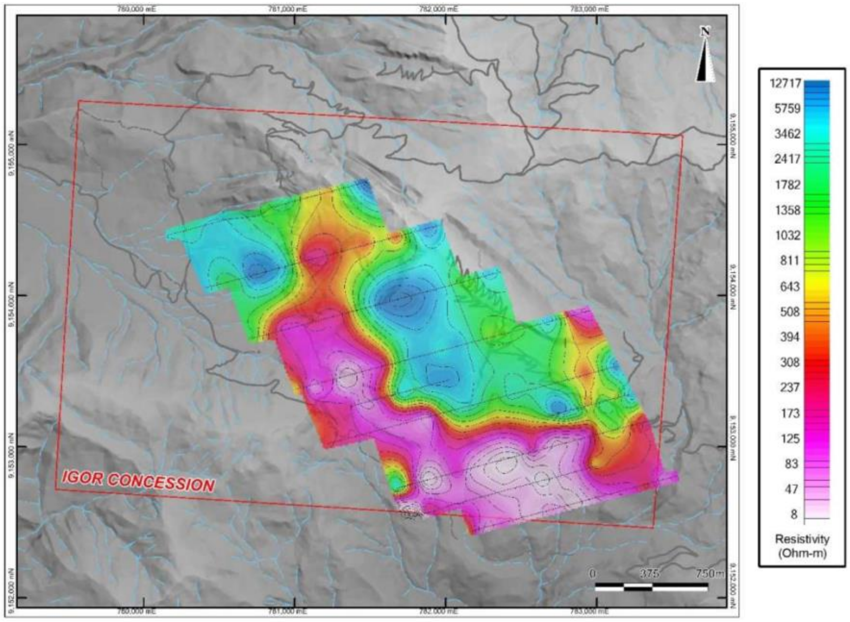
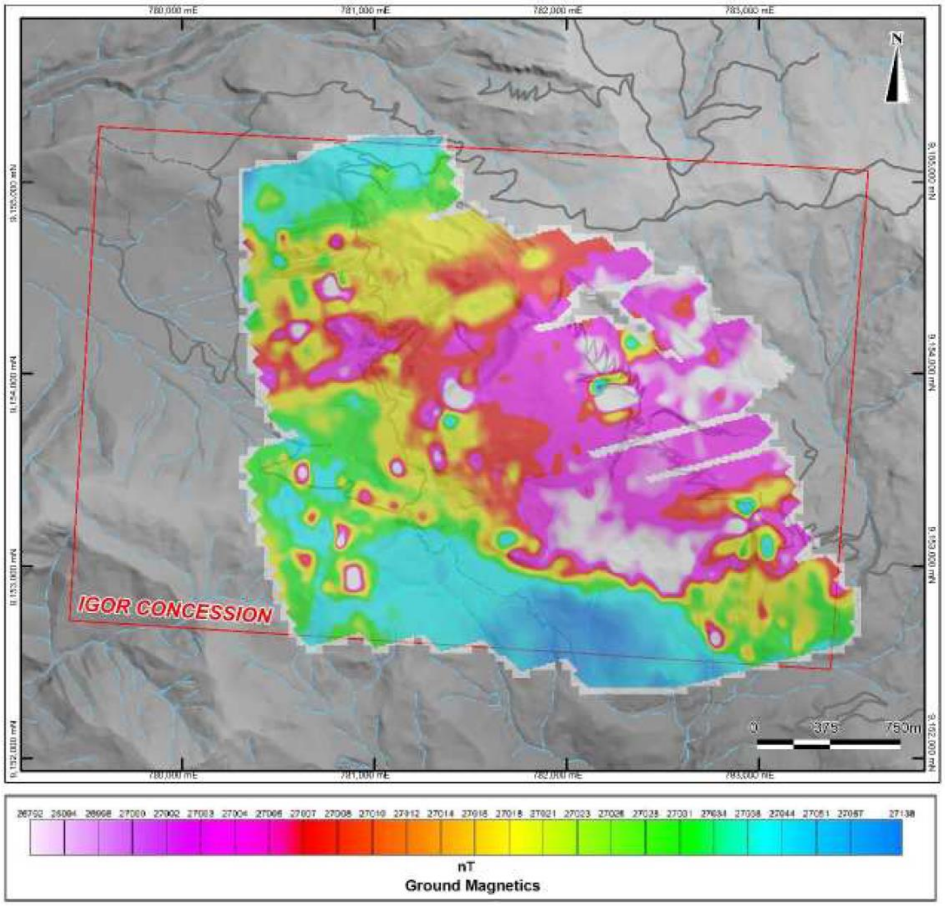


Figure 9.5 Ground Magnetic Survey, 2007
(from PPX, 2018)



9.4 Underground Test Mining and Bulk Sampling 2016 - 2018

PPX commenced underground test mining and bulk sampling of the Callanquitas deposit with their underground mining partner, Proyectos La Patagonia S.A.C. ("PLP") in October of 2016. As of the effective date of this report, a total of 40,811 tonnes have been mined. A total of 31,437 tonnes at an average grade of 8.56g Au/t have been toll processed at the off-site Malin plant of Silver Cascas S.A.C. ("Silver Cascas") and the remainder has been stockpiled on-site. Approximately 7,028 ounces of gold have been produced, with an average gold recovery rate of 81.2%. There was no recovery of silver from this material. PPX has reported "direct cash costs" of $262/tonne processed.

9.4.1 Underground Channel Sampling 2017 - 2018

PPX and PLP completed underground channel sampling as part of their test mining program. A total of 1,126 samples were collected from the Callanquitas Este vein. The channel samples, which range from 0.2 to 3 meters with a median length of 0.9 meters, are face samples collected across the width of the mineralized structure. Very often a single ~1-meter channel sample covered the full width of the mineralized structure. Where the breccia/vein structure widens, two to three contiguous channel samples characterized the mineral zone. The center sample usually covered the high-grade vein/hydrothermal breccia and the two surrounding samples comprised lower-grade wallrock or weakly mineralized tectonic breccia.

The channel samples were included in the sample data used in the current resource estimate, as described in Section 14.2, with the descriptive statistics for the channel samples in Section 14.8.2.

9.5 Author's Statement

The underground channel-sample results, along with the drilling, form the basis for the current mineral resource estimate described in Section 14.0. Drilling is described in Section 10.0 while the audit and quality control review of the underground sample and drilling data is described in Section 12.0. While Mr. Tietz has not analyzed the sampling methods or reviewed quality control on surface sampling at the Igor property, the surface work completed by PPX appears typical of projects at this intermediate stage of development and can be used to help guide further exploration. The surface sampling data were not used in the determination of the current mineral resource estimate described in Section 14.0.

10.0 DRILLING (ITEM 10)

All of the drilling summarized in this section was completed by PPX and its predecessors Peruvian and Sienna. The information presented in this section of the report is derived from multiple sources, as cited. Mr. Tietz has reviewed this information and believes this summary accurately represents drilling done at the Igor project.

10.1 Summary

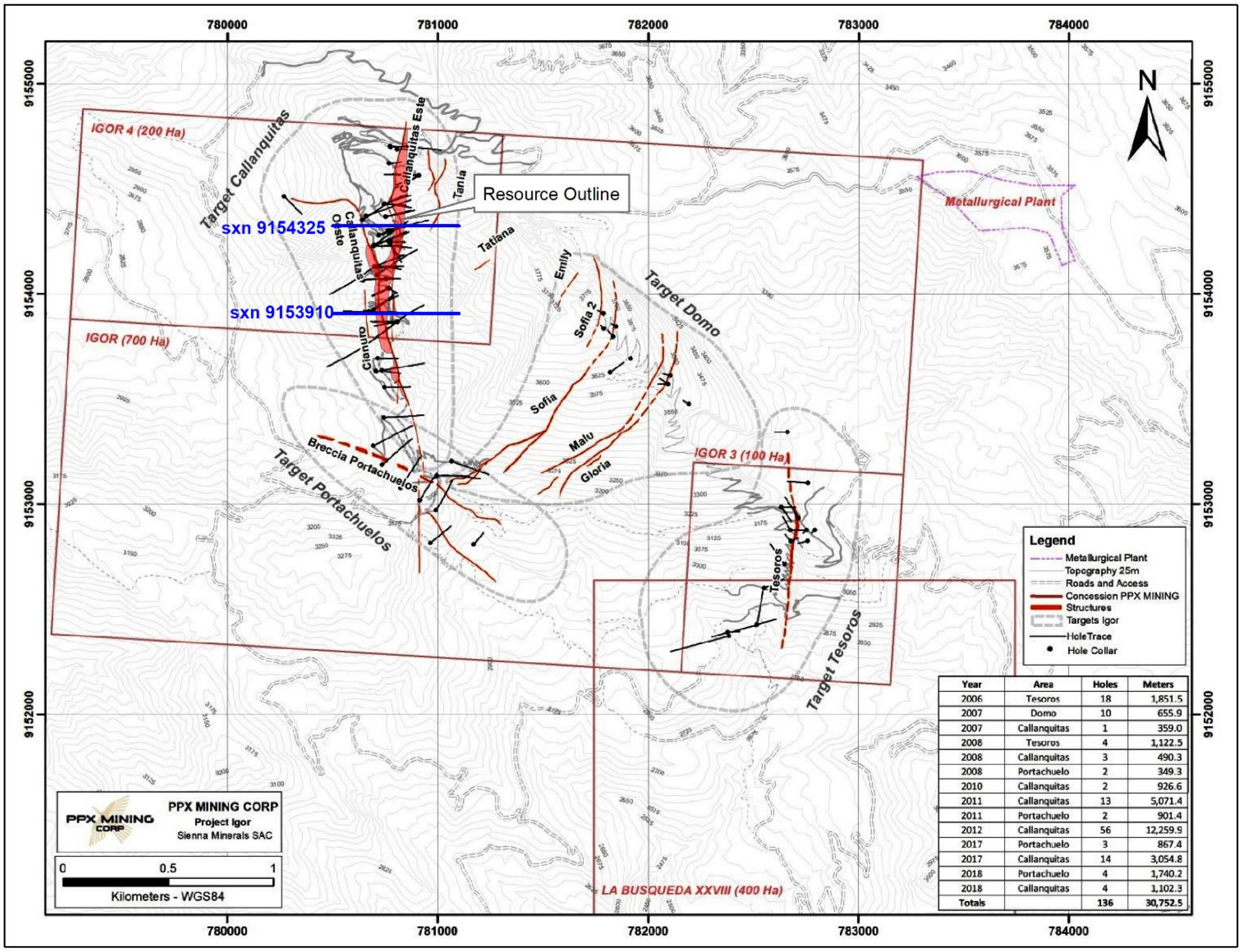
Mr. Tietz constructed a drilling database with information provided by PPX. The drilling database contains records for a total of 30,752.5 meters of drilling done in 136 drill holes from 2006 through May of 2018 (Table 10.1). Mr. Tietz is not aware of any other holes drilled within the property during that period. The most recent drill hole in the current database is hole CA-18-99. All of the drilling has been done with diamond-core wireline drilling methods. Approximately 68% of the holes and 76% of the meters have been drilled in the Callanquitas deposit. All but two of the holes were drilled with dips shallower than -85°.

Table 10.1 Summary of Igor Project Diamond-Core Drilling

Year	Area	Holes	Meters
2006	Tesoros	18	1,851.5
2007	Domo	10	655.9
2007	Callanquitas	1	359.0
2008	Tesoros	4	1,122.5
2008	Callanquitas	3	490.3
2008	Portachuelos	2	349.3
2010	Callanquitas	2	926.6
2011	Callanquitas	13	5,071.4
2011	Portachuelos	2	901.4
2012	Callanquitas	56	12,259.9
2017	Portachuelos	3	867.4
2017	Callanquitas	14	3,054.8
2018	Portachuelos	4	1,740.2
2018	Callanquitas	4	1,102.3
Totals		***136***	***30,752.5***

The distribution of drill-hole collar locations within the Igor project is shown in Figure 10.1. Representative cross-sections showing drill results are presented in Section 14.0

Figure 10.1 Map of Igor Project Drill Holes



Year	Area	Holes	Meters
2006	Tesoros	18	1,851.5
2007	Domo	10	655.9
2007	Callanquitas	1	359.0
2008	Tesoros	4	1,122.5
2008	Callanquitas	3	490.3
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2010	Callanquitas	2	926.6
2011	Callanquitas	13	5,071.4
2011	Portachuelo	2	901.4
2012	Callanquitas	56	12,259.9
2017	Portachuelo	3	867.4
2017	Callanquitas	14	3,054.8
2018	Portachuelo	4	1,740.2
2018	Callanquitas	4	1,102.3
Totals		136	30,752.5

10.2 PPX Drilling 2006 - 2007

PPX's initial drilling program consisted of 28 holes drilled in the Tesoros and Domo areas. One hole was also drilled in the Callanquitas zone. BQ-, NQ- and HQ-diameter core sizes were recovered (Davis and Sim (2013). The drilling contractor was Mining Drilling Service and MDH of Lima. Results from this drilling were used by Henkle and Lytle (2008) to estimate the historical "geological resources" at Tesoros and Domo as summarized in Section 6.2, all of which were reportedly within 40 meters of the surface.

The 2006 drilling was done with a skid-mounted Diamec 262 core drill. It has been reported that in 2007 drilling was done by the drilling contractor Energold Drilling using a Hydrocore Series One core drill (Davis and Sim, 2013).

10.3 PPX Drilling 2008

Four additional core holes were drilled at Tesoros, three holes were drilled at Callanquitas and two holes were drilled in the Portachuelos area in 2008. The best results were in Callanquitas hole CA-08-1B, which intersected 218.4 meters at average grades of 1.44g Au/t and 22.3g Ag/t, but that hole was lost while still in mineralization. Energold Drilling was the drilling contractor and a Hydrocore Series One core drill was used (Davis and Sim, 2013).

10.4 PPX Drilling 2010 – 2012

Drilling in 2010 through 2012 was mainly performed by AK Drilling International of Lima, Perú, with a track-mounted Sandvik DE-710 core drill. Some of the drilling was done by Bradley-MDH of Lima, Perú, with a Diamec 250 core drill (Davis and Sim, 2013). HQ-diameter was the core size for the vast majority of drilling during this period; much smaller amounts of NQ- and BQ-diameter core were also recovered.

A total of 15 holes were drilled at the Callanquitas area in 2010 and 2011 to test the near-surface fault zone mineralization in addition to the potential for porphyry-style mineralization at depth. Two holes were drilled at the Portachuelos area. Good results at Callanquitas confirmed the presence of a principal mineralized north-south zone with a length of at least 900 meters.

A total of 56 holes were drilled at the Callanquitas area in 2012 with the objective of expanding mineralization and defining mineral resources. Drilling focused primarily on the Callanquitas Este structure but also targeted and defined the sub-parallel Callanquitas Oeste structure. Potentially economic mineralization on the Callanquitas structures was defined over a 1,000-meter strike length and to depths of over 500 meters.

10.5 PPX Drilling 2017 - 2018

Drilling resumed in 2017 and by May of 2018 totaled about 6,765 meters in 25 holes, 18 of which were drilled in the Callanquitas area. Four of the holes were drilled in the Portachuelos area. The drilling contractor was MDH of Lima, Perú. Two LD-250 man-portable core rigs were used to recover HQ and NQ sizes of core.

The primary objective of the Callanquitas drilling was to infill gaps in the previous drill pattern and upgrade resource classification. The results increase confidence in mineral continuity within the structures. All 2017 to 2018 drilling has been included within the current resource estimate.

10.6 Drill-Hole Collar Surveys

PPX geologists used a hand-held GPS to survey the locations of the drill-hole collars in 2006 through 2008. For the 2010 through 2012 and the 2017 through 2018 drilling campaigns, the drilling sites were initially located by a contract surveyor (Ingenieria y Construcciones Inka S.A.C.) using coordinates provided by the PPX geology group. The collars were re-surveyed again after drill-hole completion.

Upon drill-hole completion, a cement plug was put within the upper meter of the hole. A 0.75-meter square concrete monument was built over the collar with a PVC pipe with metal re-bar marking the precise hole location. The drill hole specifics, including northing, easting, elevation, drill depth, and orientation, were etched within the concrete monument.

The 2017 and 2018 drill hole locations are evident and undisturbed while many of the pre-2017 drill hole locations are no longer visible due to post-drilling surface disturbance, primarily new road construction to access further drill sites and truck traffic to access these sites. PPX has provided photos of all historic drill sites which show the concrete monuments marking these now mostly covered drill locations.

10.7 Down-Hole Surveys

During the 2010 through 2012 drilling, all holes were surveyed with a REFLEX EZ-Trac survey tool at 50-meter intervals. During 2017 - 2018, Callanquitas drill holes were surveyed with the REFLEX tool at 25-meter intervals and the drill holes at Portachuelos were surveyed at 50-meter intervals.

10.8 Summary Statement

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

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

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

11.1 Core Handling, Logging, and Sampling

The following information on the 2006 through 2008 core handling and sampling procedures is from Henkle and Lytle (2008). Information on the 2010 through 2018 core handling, logging, and sampling procedures is from Davis and Sim (2013), and/or noted by Mr. Tietz during his January 2018 site visit.

Core handling and sampling for PPX's 2006 through 2008 drilling consisted of transporting the core from the drill rig to a secure, rented facility in Trujillo. After the core was logged, the core was cut in half longitudinally and separated into 2-meter intervals, with one half placed back into the core box for storage and the other half placed into tagged and then sealed plastic bags for shipment to the laboratory.

For the 2010 through 2018 drilling, the full heavy-duty, plastic core boxes were collected from the drill rig and moved to the core facilities, which include separate core logging and cutting shacks where geotechnical and then geologic logging were performed by PPX geologists. The core facilities were located on-site for the 2010 through 2012 drilling, but were moved into the town of Huaranchal, located one-hour drive south of the project site, for the 2017 and 2018 drilling. For the latter drill program, the core has been collected at the drill site and brought into Huaranchal about once per day by PPX technicians or the drillers.

Geotechnical information was recorded on paper logging forms and later captured in electronic spreadsheets. This included recovery, RQD, and fracture frequency measurements, and alteration and joint classifications. Geologic information included lithology, alteration, mineralization, and structural variations all recorded on paper logging forms and later captured in electronic spreadsheets.

Commencing in 2010, after logging completion, sample intervals based on geologic characteristics were marked on the core boxes with sample lengths ranging from 0.5 meters to 2.0 meters. In the case of poor recovery, sample lengths >2.0 meters were permitted, but a zone with poor recovery is never mixed with a zone of good recovery. Samples were assigned a blind numbered sequence and the sample locations were marked in the plastic core box and dividers. Quality assurance and quality control ("QA/QC") samples were inserted every tenth sample and the QA/QC sample locations were noted on the core boxes. The core was marked with a control cut line, perpendicular to the principal structural/bedding fabric, so that two symmetrical halves could be produced when cut. The core was then photographed and moved to the cutting shack and laid out in order.

The marked-up core was cut along the control lines and the two core halves were replaced in the core boxes. A diamond saw was used to cut the core, though in zones of highly broken core or clay-rich intervals, a spatula was used to separate the two halves and sample the core. Field duplicates were obtained by taking the halved core and cutting it again to produce two quarter pieces of core. In 2010 through 2012, one quarter-core split was used as the original and the other quarter-core split was used as the field duplicate. In 2017 and 2018, the original sample has been a full half-core and the duplicate is a quarter-core split of the remaining half core.

The samples were placed in polythene bags along with a pre-numbered sample tag and closed using a zip tie. The sample number was also written on the outside of the sample bag with a permanent marker. The samples were then collected and placed in large (rice) sacks and the sample numbers were written on the outside of the sacks and closed with a metal tie.

The samples were trucked by PPX personnel to Trujillo, a major city located on the coast 100km west of the project site, where they were picked up by SGS Laboratories ("SGS") personnel and transported to the SGS laboratory facility in Lima for analyses. The chain of custody for all transfers of the samples from Huaranchal, to Trujillo, and then Lima, was recorded and kept in PPX's Huaranchal office.

11.2 Sample Preparation and Analysis

The 2006 through 2008 drill samples were analyzed at ALS Chemex Laboratories ("ALS") in Lima, Peru. After crushing the total sample and then pulverizing a 250g split to 85% of the sample passing 70 microns, a 30-gram aliquot was assayed for gold by fire-assay fusion with an atomic absorption ("AA") finish (ALS code Au-AA23). A second 0.5-gram aliquot was analyzed for 34 elements, including silver, by inductively-coupled plasma atomic-emission spectroscopy ("ICP") after aqua regia digestion (ALS code ME-ICP41). Samples which exceeded 10 ppm Au in the initial gold analyses were re-assayed by fire-assay fusion with a gravimetric finish (ALS code Au-GRA21). Samples with silver greater than the ICP upper detection limit of 100 ppm Ag were re-assayed by AA following aqua regia digestion (ALS code Ag-AA46). Samples with very high-grade silver values (>1,500 ppm Ag) were re-assayed by fire assay with gravimetric finish (ALS code AG-GRA21).

The 2010 through 2018 drill and underground samples were analyzed at SGS Laboratories ("SGS") in Lima, Peru. After crushing the total sample and then pulverizing a 250g split to 85% of the sample passing 70 microns, a 30-gram aliquot was assayed for gold by fire-assay fusion with an AA finish (SGS code FAA313). A second 0.5-gram aliquot was analyzed for 39 elements, including silver, by ICP following digestion by aqua regia (SGS code ICP12B). Samples which exceeded the upper limits of the initial gold analyses (>5 ppm Au) were re-assayed by fire assay with a gravimetric finish (SGS code FAG303). Samples with silver greater than the ICP upper limit of 100 ppm Ag were re-assayed by AA following aqua regia digestion (SGS code AAS11B).

11.3 Sample Security

The core and core samples in all drill campaigns have remained under the supervision of PPX personnel and kept in secure, locked facilities through the logging and sampling process. There is an acceptable chain of custody protocol for the shipping and transferring of samples to the analytical laboratories. Once a drill hole is logged and sampled, the sampled core is taken by truck from Huaranchal to PPX's secure core storage facility in Trujillo. It is Mr. Tietz's opinion that the sampling and chain of custody protocols used by PPX are adequate.

11.4 Quality Assurance/Quality Control ("QA/QC")

QA/QC samples were inserted into the sample stream sent to the laboratories on a regular basis in all drill campaigns. For the 2006 through 2008 drilling, QA/QC samples consisted of preparation blanks and certified reference materials ("CRMs") inserted at an approximate rate of one QA/QC sample for every 50 drill samples sent to ALS. For the four Callanquitas holes, the insertion rate was increased to approximately one QA/QC sample (a blank or CRM) for every nine drill samples. For all later drill campaigns (2010 through 2018), field duplicates were collected and included in the sample stream sent to ALS and SGS, while keeping the same QA/QC insertion rate; i.e. one blank, one standard, or one field duplicate was included with every nine samples sent to the lab. The QA/QC samples have been inserted on a regular pattern so approximately equal number of QA/QC sample types have been inserted into the sample stream.

Preparation "coarse" blanks were used in all drill campaigns while analytical "fine" blanks were inserted into the sample stream in the 2017 through 2018 drill campaign. The preparation blanks are a quartzite gravel collected near Lima, Peru and purchased from an outside vendor. The analytical blanks are pulp material purchased from COMACSA (Agregados Calcareos S.A.). Neither of these sample types can be considered "blind" to the analytical laboratories when compared against the original core samples. A total of 834 preparation blanks and 85 analytical blanks are within the current project database.

Eleven different CRMs have been used and these were purchased as pulp material from commercial laboratories Smee and Associates (Vancouver, Canada) and ROCKLABS (Auckland, New Zealand). A total of 729 CRM samples are within the current project database. The CRMs are not blind to the analytical laboratory.

The field duplicates are quarter-core samples. In the 2010 through 2012 drill program, the sampling procedure was to take both the original and duplicate as quarter-core samples from the same half-cut core piece. The sampling procedure changed during the 2017 through 2018 drill program such that the original sample was a full half-core and the duplicate was a quarter-core split of the remaining half core. A total of 556 field duplicate samples are within the current project database.

The QA/QC program also included sending original pulps to a second laboratory as same-pulp duplicates. Two hundred same-pulp duplicates were sent to Inspectorate (Lima, Peru) in 2012 for check assays and 342 same-pulp duplicates were sent to Certimin (Lima, Peru) in 2018. The same-pulp duplicates were selected from mineralized intervals and included a full range of gold values. The same-pulp duplicates used similar analytical methods (fire assay with atomic absorption finish for gold and aqua regia with an ICP finish for silver) as the original laboratories.

The QA/QC analyses and a discussion of the results are presented in Section 12.6.

11.5 Summary Statement

Mr. Tietz believes the sample preparation, security, and analytical procedures used by PPX were acceptable procedures and the resulting analytical data are of sufficient quality for use in the resource estimation.

12.0 DATA VERIFICATION (ITEM 12)

Data verification, as defined in NI 43-101, is the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used. There were no limitations on, or failure to conduct, the data verification for this report. Additional confirmation of the drill data's suitability for use are the analyses of the project QA/QC procedures and results as described in Section 12.6.

12.1 Site Visit

Mr. Paul Tietz visited the Igor project office and field site on January 12-18, 2018. During the site visit, the project geology was reviewed, which included: a) a field tour of the deposit area; b) visual inspection of drill core; and c) discussion of the current geologic interpretations with PPX personnel. Verification of drill sites was conducted, and core drilling and sampling procedures were appraised. Mr. Tietz has also maintained communication in 2018 through telephone calls and emails with PPX project personnel in which the project status, procedures, and geologic ideas and concepts have been discussed. The result of the site visits and communications is that the author has no significant concerns with the project procedures or the geologic interpretations used in building the geologic model.

Mr. Seamons visited the Igor project site between June 17th to the 22nd, 2018 examining current surface and underground infrastructure, mining methods and general geotechnical conditions to validate the mining methods used for the statement of reserves as well as current costs.

12.2 Metallurgical Data Verification

The sampling of mineralized material in the Callanquitas mine and the metallurgical testwork carried out by METTS, as well as other laboratories, were not supervised or observed by Dr. Uceda. As such, the results were not independently verified, but Dr. Uceda has reviewed the reports and data of METTS and PPX, and believes the information is sufficiently reliable to be used in this report and support the conclusions summarized in Section 13.0.

12.3 Plant Design, Cost and Economic Assumptions

Dr. Uceda has evaluated the information received from MTTS and PPX concerning the laboratory test work related to process requirements and plant design. Dr. Uceda has not visited the project site. The review of the test work and design work completed by MTTS allowed Dr. Uceda to consider that the work completed satisfies the quality and engineering level for the process design for the plant.

Mr. Burga has completed an evaluation for validation of the information received from METTS and PPX related to Process and Plant facilities presented in Sections 17 and 18 of the Technical report. He considers that Capex definition and process OPEX definition are suitable for use in Section 21. He has not visited the project site. Mr. Burga considers that the engineering work done to complete the technical information is in accordance with engineering level required for the NI43-101 Pre-Feasibility Technical Report.

Mr. Dyer reviewed costing structures and cost assumptions from site to be used for costs. This was benchmarked with costs from other similar operations.

Metal prices used for the economic evaluations were based on public information gathered by Mr. Dyer including Kitco metal prices and consensus prices from other studies. See Section 19 for additional details.

12.4 Independent Verification of Mineralization

PPX has been disclosing gold and silver production data from underground test mining and bulk sampling since 2017, and selling these metals (see Section 9.4). Therefore, Mr. Tietz has not collected and analyzed samples for independent verification of mineralization.

12.5 Database Verification

Mr. Tietz has verified the Igor project database and compiled and analyzed available QA/QC data collected by PPX.

The Igor project database includes information derived from 136 drill holes, of which 93 drill holes are within the Callanquitas deposit and are used in the current resource estimate. The database also includes location, rock type, and assay data for 1,126 underground chip and channel samples collected by PPX from within development drifts and stopes of the Callanquitas mine.

For this resource estimate, Mr. Tietz completed a full audit of the Callanquitas drill collar, down-hole survey, and drilling assay data as summarized below. The database also includes geology, geotechnical and bulk density drill data which were reviewed to check for material inconsistencies.

The underground sample location data was checked against current surveyed underground mine working shapes. The underground chip and channel sample assays were audited by comparison to the original laboratory certificates.

12.5.1 Drill-Collar Audit

The drill hole collar locations in the database were checked against records of the original survey readings and no errors were found in the easting, northing or elevation values. Mr. Tietz also compared the drill hole locations to the current topographic surface and found there are no drill holes lying a significant distance above or below the topography.

During the site visit, Mr. Tietz verified the location of seven core holes drilled in 2017 and six pre-2017 drill holes using a hand-held GPS receiver. All of the field readings were within a few meters of the database collar coordinates.

The drill collars are marked with a concrete monument. The 2017 drill hole locations are evident and undisturbed, while many of the pre-2017 drill hole locations are no longer visible due to post-drilling surface disturbance, primarily new road construction to access further drill sites and truck traffic to access these sites. PPX has provided photos of all pre-2017 drill sites which show the concrete monuments marking these now mostly covered drill locations.

12.5.2 Down-Hole Survey Audit

There are downhole Reflex survey readings for 85 of the 93 Callanquitas drill holes. Azimuth and dip measurements, along with magnetic and gravity values, were collected on about 25 to 50 meter-spaced depth intervals. Mr. Tietz checked the database downhole survey readings against the original data and no significant errors were found. Minor changes to some readings were completed, especially where the original readings were associated with anomalous magnetic values. But these were minor and the author believes the downhole survey data is adequate for use in the current resource estimate.

12.5.3 Assay Audit

The drill assay database, which included the drill data and all QA/QC analyses, was compared to the original SGS and ALS laboratory assay certificates. Twenty-eight errors were noted for an error rate of about 0.13% of the total assay database. Two of the errors involved gold assays and 26 involved the silver data. Corrections were made to the data before use in the resource estimate.

Mr. Tietz noted that the SGS internal laboratory duplicate assays were used by PPX in calculating the database entries which are average gold and silver values for the 2011 and 2012 drilling. This was not done in either the earlier or later drill campaigns where just the original assays were used as the “final” value. Mr. Tietz removed the SGS lab duplicate gold assays from the 2011 and 2012 drill samples so the data is consistent across all drill campaigns and only the original value was used in the current resource estimate.

12.5.4 Geology Review

Drill logs and core photographs are available for almost all Callanquitas drill holes. A detailed audit of the complete geology database was not conducted but the author did review the logged mineralized breccia/vein intervals for all prominent holes in the database and compared these to the drill logs and core photographs. Lithology, breccia/vein types, structural orientations, and mineralization type (oxide, mixed, or sulfide) were evaluated with no material differences noted between the database, drill log, and core photos.

12.5.5 Underground Sample Audit

Underground sample locations were checked against the current surveyed underground as-built shapes; no material location errors were noted. Approximately one half of the underground assay data was digitally compared to the original SGS laboratory certificates. Only two errors in the gold data were found and these were corrected.

12.5.6 Surface Sample Audit

Mr. Tietz reviewed the surface sample locations and geochemical results, as presented by PPX on company geology maps, for reasonableness and believes that the surface work completed by PPX appears typical of projects at this intermediate stage of development and can be used to help guide further exploration. Mr. Tietz has not analyzed the sampling methods or reviewed quality control on surface sampling at the Igor property and these data were not used in the determination of the current mineral resource estimate described in Section 14.0.

12.6 Quality Assurance/Quality Control

Mr. Tietz reviewed the QA/QC procedures and results for the 2007 through 2018 drilling, and the 2018 underground sampling within the Callanquitas deposit. As discussed in Section 11.4, the QA/QC data included assays of preparation coarse blanks, analytical fine blanks, 11 different CRMs, quarter-core field duplicates, and second-lab pulp checks sent to two different laboratories.

12.6.1 Blanks

Preparation "Coarse" Blanks: Preparation blanks were inserted into the sample stream to monitor possible laboratory contamination during sample preparation. In order for analyses of blanks to be meaningful, they must be sufficiently coarse to require the same crushing and pulverizing stages as the drill samples. PPX has used a coarse quartzite gravel collected from off-site as the blank material. PPX has stated that SGS conducted round-robin testing of the source material to verify that the quartzite is unmineralized, though Mr. Tietz has not seen these data.

A total of 834 preparation blank analyses are within the project database. These blanks have been inserted into the sample stream at a consistent rate of one every 20th drill sample for the 2007 through 2008 drilling and then one every 30th drill sample for the 2010 through 2018 drilling. A preparation blank is also usually inserted at the beginning of each sample shipment sent to the lab, which is often the start of a new drill hole. Instead of inserting the blanks on a regular, consistent rate, it would have been preferable if the blanks were placed in the sample stream within, or immediately following, a set of mineralized samples, which would be the source of most contamination issues.

Blank results that are greater than five times the lower detection limit of the relevant analyses are typically considered failures that require further investigation and possible re-assaying of associated drill samples. The detection limits of the ALS and SGS analyses was 0.005g Au/t and 0.2g Ag/t, so blank samples assaying in excess of 0.025g Au/t and 1.0g Ag/t are considered to be failures.

Table 12.1 is a summary of the preparation blanks results for all of the PPX drill campaigns. In total, there were nine gold analyses which exceeded the failure threshold, for an average failure rate of 1.1%. The highest gold failure rate of 3.3% was in the 2007 – 2008 ALS analyses; though with only one failure in this data set, the high rate results from the low total number of analyses. For all other drill campaigns, the failure rate in the gold analyses for the blanks is under 1.5%. The single highest blank value is 0.096g Au/t in the 2011 – 2012 SGS laboratory data.

Table 12.1 Preparation Blanks Results

	Lab	Count	Gold		Silver	
			Failures	Fail_%	Failures	Fail_%
2007-2008	ALS	30	1	3.3%	2	6.7%
2010-2011	ALS	108	1	0.9%	2	1.9%
2011-2012	SGS	480	6	1.3%	7	1.5%
2017-2018	SGS	216	1	0.5%	2	0.9%
Total		834	9	1.1%	13	1.6%

For the silver blank analyses, there were thirteen silver analyses which exceeded the failure threshold, for an average failure rate of 1.6%. The highest silver failure rate of 6.7% (from two failures) was also in the 2007 – 2008 ALS analyses. For all other drill campaigns, the failure rate in the silver analyses for the blanks is under 2.0%. The single highest blank silver value is 7.5g Ag/t in the 2011 – 2012 SGS laboratory data.

Figure 12.1 and Figure 12.2 are graphical representations of the 2011-2012 SGS preparation blank gold analyses. Similar graphs were completed for the gold and silver in all other drill campaigns. Figure 12.1 plots the gold blank values in the y-axis while the numbered temporal sequence of analyses is plotted in the x-axis of the graph. The over-limit failure value is indicated by the red line. The six gold failures are scattered throughout the sequence so there are not any indications of any one specific time period with increased blank failures.

Figure 12.1 Gold in Preparation Blanks: 2011 - 2012 SGS Analyses

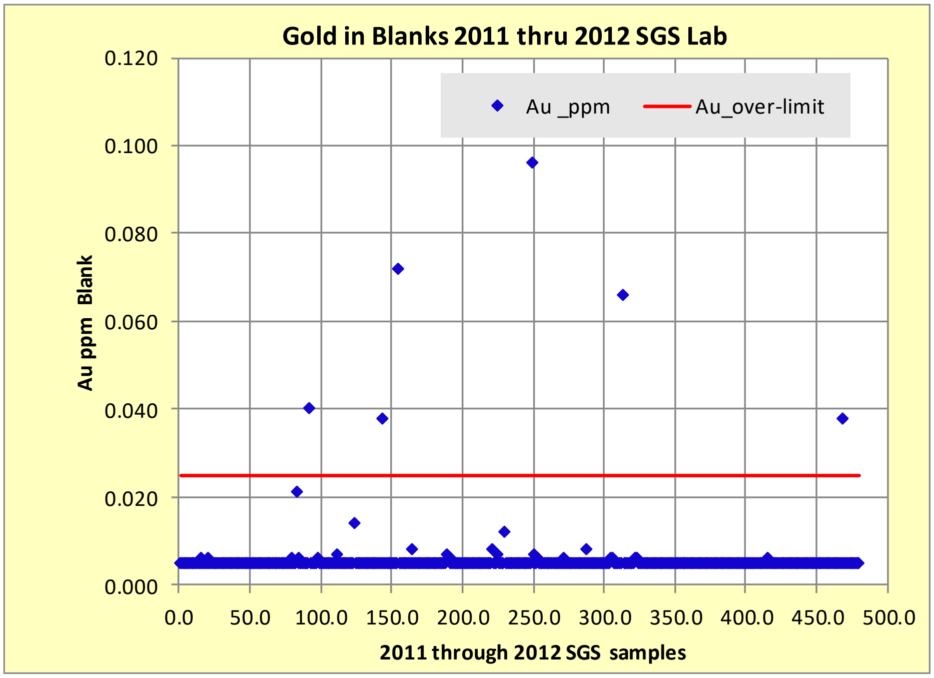
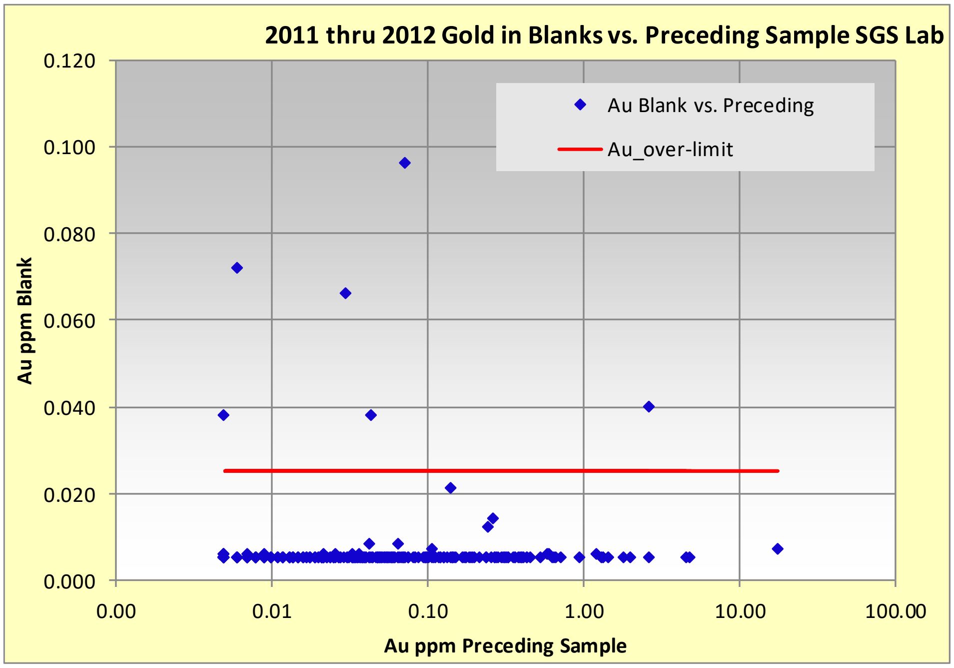


Figure 12.2 presents the gold blank analyses, again in the y-axis, versus the preceding sample's gold value in the x-axis. There is no relation between the blank failures and higher-grades within the

preceding samples so the failures are not likely due to contamination. In fact, three of the blank failures have higher gold values than the very low-grade preceding sample. There is the same lack of correlation between blank failures and the preceding sample values in the gold and silver analyses in all other drill campaigns.

Figure 12.2 Gold in Preparation Blanks vs Preceding Sample: 2011 - 2012 SGS Analyses



While the blank data shows the occasional failure in the gold and silver analyses, there is no evidence of significant cross contamination during sample preparation. The failures are not considered to have a material negative impact on the use of the assay data in resource estimation and classification.

<u>Analytical “Fine” Blanks</u>: Analytical blanks were used to monitor possible contamination or calibration problems during the determination of gold concentrations. PPX used a blank commercial pulp supplied by COMACSA (Agregados Calcareos S.A.) for the 2017-2018 QA/QC program. There are 85 SGS analyses of the analytical blank, and none of the analyses exceeded the 0.025 g Au/t threshold.

12.6.2 Reference Standards (CRMs)

CRMs were used to monitor the analytical accuracy and precision. PPX has used 11 different CRMs during the various drill campaigns. The standards used are pulp material purchased from certified commercial laboratories Smee and Associates (“Smee”) (Vancouver, Canada) and ROCKLABS (Auckland, New Zealand). Table 12.2 is a listing of the various standards and includes the laboratory source, and expected value and standard deviation (“SD”), for each reference standard. As indicated in Table 12.2, the ROCKLABS CRMs were only certified for gold.

Table 12.2 Igor Project Reference Standards

Standard	Source	Gold		Silver	
		Mean (g/t)	SD (g/t)	Mean (g/t)	SD (g/t)
OxF100	Rocklabs	0.804	0.019		
OxF125	Rocklabs	0.806	0.02		
OxJ80	Rocklabs	2.331	0.042		
SE29	Rocklabs	0.597	0.008		
SE58	Rocklabs	0.607	0.019		
SL51	Rocklabs	5.909	0.136		
AuOx28	Smee	2.391	0.049	488	8
AuOx29	Smee	0.606	0.028	174	3
AuOx34	Smee	8.81	0.026	1077	15
PAL-11	Smee	2.05	0.11	536	13
TR 11209	Smee	0.663	0.024	1.77*	0.15

* Expected silver value is considered "Provisionary" by Smee who advises caution when assessing the accuracy of individual analyses

A total of 729 analyses of CRM samples are within the project database. These have been inserted into the sample stream at a consistent rate of one every 20th drill sample for the 2007 through 2012 drilling and then one every 30th drill sample for the 2017 and 2018 drilling. The CRMs are pulps so the standards are not blind to the laboratory, which could reduce their effectiveness. All 11 CRMs were not used in all drill campaigns. Usually just one or two different CRMs were inserted within the same sample shipment. This resulted in just a handful of analyses for some of the standards.

The gold and silver analyses for each CRM were evaluated independently and a summary of the results listed by date is shown in Table 12.3 and Table 12.4. A graphical example of the individual analysis is presented in Figure 12.3. Each table shows the number of reference samples ("Count"), the expected mean value from either Smee or ROCKLAB, the actual mean of the reference analyses, and the difference in percent between the expected and analytical mean value ("diff%"). The difference values for gold range from -6% to 7%, and average less than 1%. While some of the standards show a small bias, there is not a material bias with analytical accuracy for the reference gold samples. For silver, CRM analyses are only available for a portion of the 2012 drilling and then 2017 through 2018. Except for standards TR 11209, which is both very low-grade and the expected value is judged to be "Provisional" by Smee, there is a low (2% to 7%) positive bias in the analyses versus the expected values.

Table 12.3 Reference Standards – Gold Analyses

Standard	Date	Lab	Count	Expected mean (g Au/t)	Analyses mean (g Au/t)	Mean Diff%	Analyses SD	Analyses Warnings (>±2SD)	Warnings %	Analyses Failures (>±3SD)	Failures %
SE29	2008-2011	ALS	112	0.597	0.563	-6.0%	0.019	4	3.6%	3	2.7%
SE29	2012	SGS	37	0.597	0.594	0.0%	0.016	1	2.7%	0	0.0%
SE58	2012	SGS	193	0.607	0.605	0.0%	0.017	8	4.1%	2	1.0%
TR 11209	2012	SGS	52	0.663	0.635	-4.0%	0.023	3	5.8%	0	0.0%
OxF100	2012	SGS	70	0.804	0.792	-2.0%	0.026	1	1.4%	1	1.4%
OxJ80	2012	SGS	46	2.331	2.341	0.0%	0.054	1	2.2%	0	0.0%
PAL-11	2012, 2017	SGS	27	2.05	2.123	4.0%	0.043	1	3.7%	0	0.0%
SL51	2017	SGS	8	5.909	5.704	-3.0%	0.105	0	0.0%	0	0.0%
OxF125	2017-2018	SGS	115	0.806	0.769	-5.0%	0.016	9	7.8%	0	0.0%
AuOx28	2018	SGS	26	2.391	2.393	0.0%	0.076	1	3.8%	0	0.0%
AuOx29	2018	SGS	29	0.606	0.646	7.0%	0.646	0	0.0%	0	0.0%
AuOx34	2018	SGS	14	8.81	8.923	1.0%	0.152	2	14.3%	0	0.0%
all			729					31	4.3%	6	0.8%

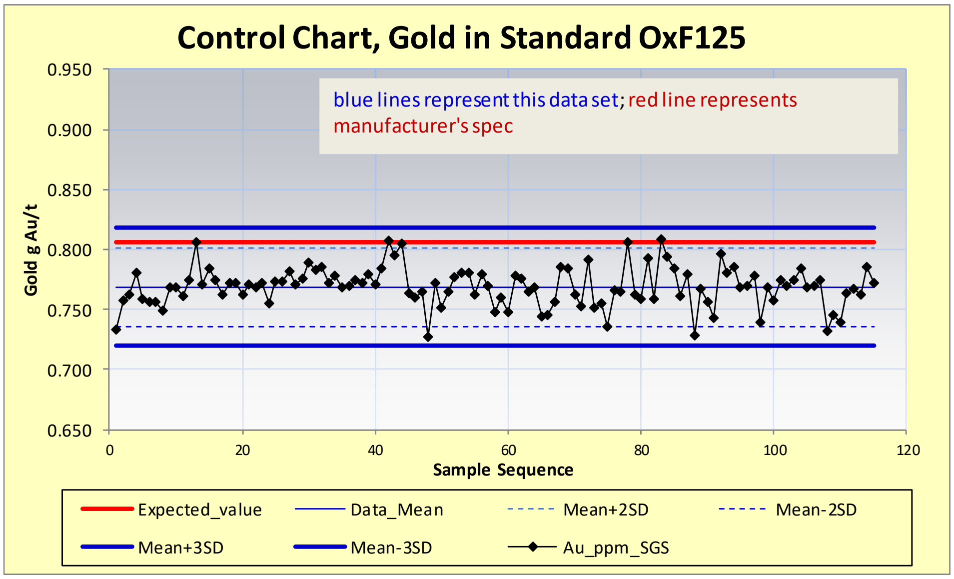
Table 12.4 Reference Standards – Silver Analyses

Standard	Date	Lab	Count	Expected mean (g Ag/t)	Analyses mean (g Ag/t)	Mean Diff%	Analyses SD	Analyses Warnings (>±2SD)	Warnings %	Analyses Failures (>±3SD)	Failures %
TR 11209*	2012	SGS	52	1.77	1.16	-34.0%	0.18	2	3.8%	0	0.0%
PAL-11	2012, 2017	SGS	27	536	563	5.0%	14	3	11.1%	0	0.0%
AuOx28	2018	SGS	26	488	498	2.0%	6	0	0.0%	0	0.0%
AuOx29	2018	SGS	29	174	181	4.0%	2	2	6.9%	0	0.0%
AuOx34	2018	SGS	14	1077	1151	7.0%	10	1	7.1%	0	0.0%
all			148					8	5.4%	0	0.0%

Analytical precision, which is based on the actual ALS or SGS laboratory results, is noted in the columns on the right side of Table 12.3 and Table 12.4. The analytical standard deviation is noted in the "Analyses SD" column while number of samples and percentage of samples greater than ±2SD (warning level) and ±3SD (failure level) are shown in the respective columns. Overall just 4% of the gold analyses (Table 12.3) fall in the warning category and just 1% are considered failures. All of the failures occurred in the early 2008 through 2012 drilling. For the silver analyses (Table 12.4), 5% of the reference samples are in the warning category but there were no failures. Mr. Tietz is not aware of any re-assaying of the original samples due to the occasional CRM failure.

A control chart for CRM OxF125 is shown in Figure 12.3. Similar charts were prepared for both gold and silver for all CRMs. In the Figure 12.3 chart, the expected value is shown as the red line while the analytical mean ("data_mean") and ±2SD and ±3SD limits are shown as blue solid or dashed lines. The -5% bias in the data, as noted in Table 12.3, is evident as all but four of the data points are below the expected value line. But the data is relatively consistent in grade and though there are nine samples outside the ±2SD limit, there were no failures.

Figure 12.3 CRM OxF125 – Gold Analyses



The results of the CRM analyses indicate no material concerns with the standard data, the results of which provide confidence in the use of the drill and underground sample data.

12.6.3 Quarter-Core Field Duplicates

Field duplicates are secondary quarter-core samples collected from the same drill interval as the original sample. They are mainly used to assess the natural grade variability of the deposit, as well as to evaluate the total subsampling variances attributable to splitting both in the field and in all subsequent subsampling steps in the laboratory.

In PPX's 2010 through 2012 drill program, the field duplicate sampling procedure was to take both the original and duplicate as quarter-core samples from the same half-cut core piece. The sampling procedure changed in the 2017 through 2018 drill program where the original sample was a full half-core and the duplicate was a quarter-core split of the remaining half core.

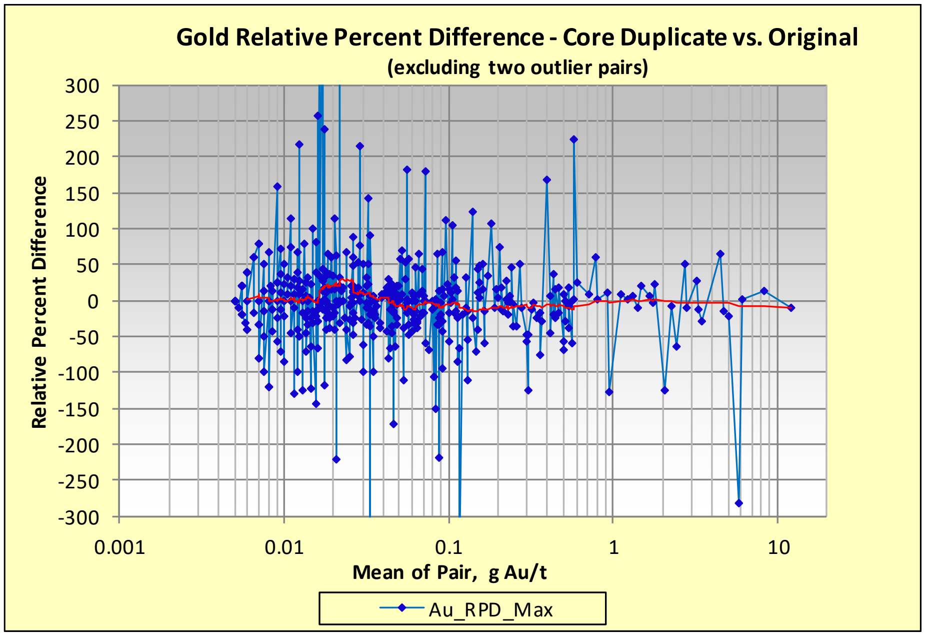
A total of 556 field duplicate samples were analyzed by ALS (2010 and 2011) and then SGS (2011 through 2018). The individual data sets were evaluated and there are no material differences between the ALS and SGS data. The data presented herein are the results of the combined data sets.

The relative-difference graph in Figure 12.4 shows the percentage difference (plotted on the y-axis) of each field-duplicate assay relative to its paired primary-sample analysis. This relative difference ("RD") is calculated as follows:

$$100\, x \frac{(duplicate - original)}{lesser\ of\ (duplicate, original)}$$

The x-axis of the graph plots the means of the gold values of the paired data (the mean of the pairs, or "MOP") in a logarithmic fashion so all grade ranges are easily observed. The red line shows the moving average of the RDs of the pairs, thereby providing a visual guide to trends in the data that aids in the identification of potential bias. Positive RD values indicate that the duplicate-sample analysis is greater than the primary-sample assay. Two sample pairs characterized by unrepresentatively high RDs (>500%) are excluded from Figure 12.4.

Figure 12.4 Relative Percent Difference – Quarter-Core Field Duplicates – Gold



The graph indicates generally a low bias in the duplicate gold results relative to the original assays over the mid- and high-grade portions of the data. There can be significant differences between individual sample pairs, as shown by the number of spikes in the data, but there is not a strong bias in the over-all data set. There are only 23 sample pairs with mean grades greater than 1.0g Au/t, and the data set could be improved by the inclusion of more strongly mineralized duplicate samples. Figure 12.5 plots the absolute value of the relative difference ("AVRD") data which provides guidance on the total grade variability between the original and duplicate samples. This variability measure includes variability inherent in the deposit plus any sub-sampling and analytical variability. The two sample pairs characterized by unrepresentatively high RDs (>500%) excluded from Figure 12.4 are also excluded from Figure 12.5. The trendline in Figure 12.5 shows that there is a 25% to 60% difference in gold grades between the original and quarter-core samples. At higher grades, near 1.0g Au/t and above, the variability starts to level out at about 35%. This amount of variability is not unusual in high-grade gold deposits.

Figure 12.5 Absolute Relative Percent Difference – Quarter-Core Field Duplicates - Gold

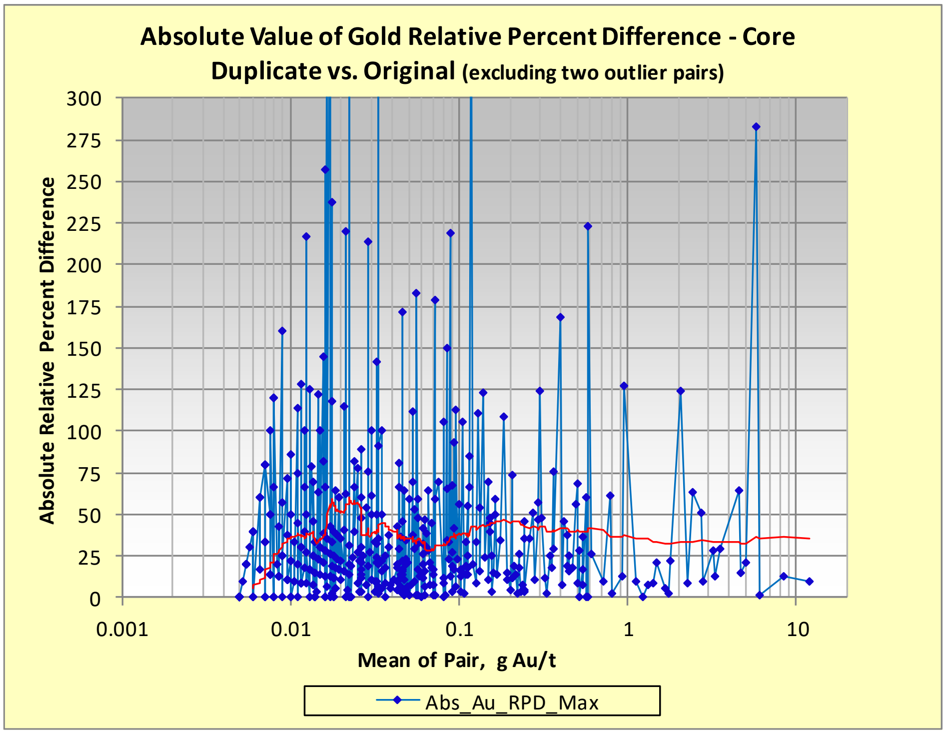


Table 12.5 and Table 12.6 are statistical summaries of the quarter-core analyses for gold and silver, respectively, listed by metal grade categories. The tables include the number of sample pairs in each grade category, along with original and duplicate mean grades, mean grade differences (diff%), RD mean grades (mean rel_diff%), and AVRD mean grades (mean abs_rel_diff%). The statistics for gold in Table 12.5 are reflective of the graphical representations in Figure 12.4 and Figure 12.5. The duplicate values are on average lower grade than the original, though the increased low bias (-15.8% mean RD) in the higher-grade category must be weighed against the low number of sample pairs and the effect of a single, very low, RD value. The latter is indicated by the large downward spike at 6g Au/t in Figure 12.4. If that single low RD value is removed, the mean RD goes from -15.8% to -4%.

Table 12.5 SGS Quarter-Core Field Duplicates - Gold (2011 - 2018)

Au (g/t)	count	mean orig.	mean dupl.	diff%	mean rel_diff%	mean abs_rel_diff%
>0.005	498	0.249	0.231	-7.5	1.1	40.1
>0.1	118	0.960	0.883	-8.4	-7.7	39.0
>1.0	23	3.685	3.336	-9.9	-15.8	35.4

The summary statistics for silver are in Table 12.6. At economic grade ranges (>10g Ag/t), there is a small positive bias (6.2% mean RD) in the duplicate analyses though the mean variability is high (65% mean AVRD). The difference in mean grades (diff%) shows the duplicate values to be 34% to 38%

higher in mean grade than the originals. These values are very different than the RD values. A review of the sample data indicates that the mean grade differences are strongly influenced by the very high-grade samples in the data set. Table 12.7 is the same data set, though with the four sample pairs that are over 200g Ag/t removed from the data. The maximum sample value removed is 2,587g Au/t. The difference in mean grades (diff%) is now down below 10% with just minor changes in the RD and AVRD values. Five sample pairs characterized by unrepresentatively high RDs (>500%) are excluded from the Table 12.6 and Table 12.7 statistics.

Table 12.6 SGS Quarter-Core Field Duplicates - Silver (2011 - 2018)

Ag (g/t)	count	mean orig.	mean dupl.	diff%	mean rel_diff%	mean abs_rel_diff%
>0.2	485	13.1	18.4	33.7	-10.1	72.0
>1.0	265	23.6	33.3	34.1	-8.5	67.0
>10.0	62	90.4	132.5	37.8	6.2	65.0

Table 12.7 SGS Quarter-Core Field Duplicates – Silver* (2011 - 2018)
(* Sample Pairs >200g Au/t Removed)

Ag (g/t)	count	mean orig.	mean dupl.	diff%	mean rel_diff%	mean abs_rel_diff%
>0.2; <200	481	5.7	5.9	3.4	-10.9	71.5
>1.0; <200	261	10.1	10.5	3.9	-9.9	66.7
>10.0; <200	58	34.2	36.7	7.1	1.0	64.4

12.6.4 Second-Lab Pulp Check Assays

Same-pulp duplicates, consisting of original assay pulps from the primary lab that are analyzed by a second laboratory, were used to test for assaying accuracy. PPX sent 200 original pulps to Inspectorate in 2012 and then sent an additional 347 original pulps to Certimin in 2018. SGS was the primary lab for all but four samples submitted to Inspectorate in 2012.

Mr. Tietz evaluated the same-pulp duplicates data in a similar way as the field duplicate analyses shown above in Section 12.6.3. Total original and duplicate mean grades were evaluated along with RD and AVRD data to assist in determining bias and sample variability, respectively.

Table 12.8 and Table 12.9 show the same-pulp duplicate analyses for gold and silver, respectively, while Figure 12.6 and Figure 12.7 are graphical examples of the RD and AVRD data, respectively, for the gold same-pulp duplicates. Within the tables, the 2012 Inspectorate and 2018 Certimin results are shown separately though as is readily apparent the results are similar for both data sets. There is no material bias between SGS and the second laboratories, and the sample variability is below 10% for both gold and silver analyses. These results provide confidence in the use of the SGS data.

Table 12.8 Second-Lab Same Pulp Checks of SGS Drill Hole Gold Assays

Year	Second Lab	Au g/t	count	mean orig.	mean dupl.	diff%	mean rel_diff%	mean abs_rel_diff%
2012	Inspectorate	>0.1	199	4.186	4.171	-0.4	-1.6	8.7
2012	Inspectorate	>1.0	124	6.489	6.466	-0.4	-3.2	7.6
2018	Certimin	>0.1	301	3.215	3.195	-0.6	2.1	5.5
2018	Certimin	>1.0	166	5.497	5.451	-0.8	1.2	5.0

Figure 12.6 Relative Percent Difference: Certimin Gold vs. SGS Original Gold

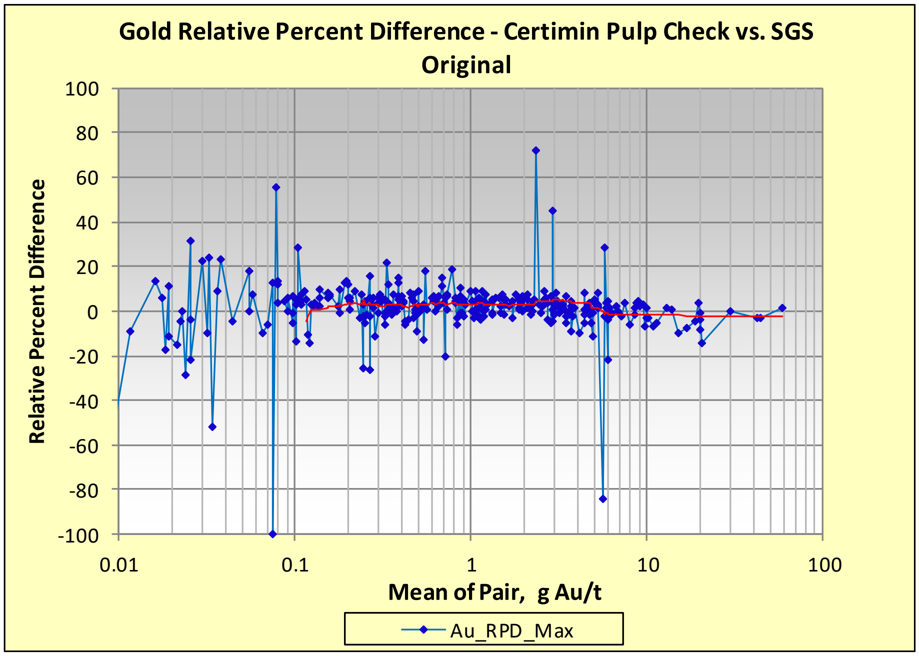


Figure 12.7 Absolute Relative Percent Difference: Certimin Gold vs. SGS Original Gold

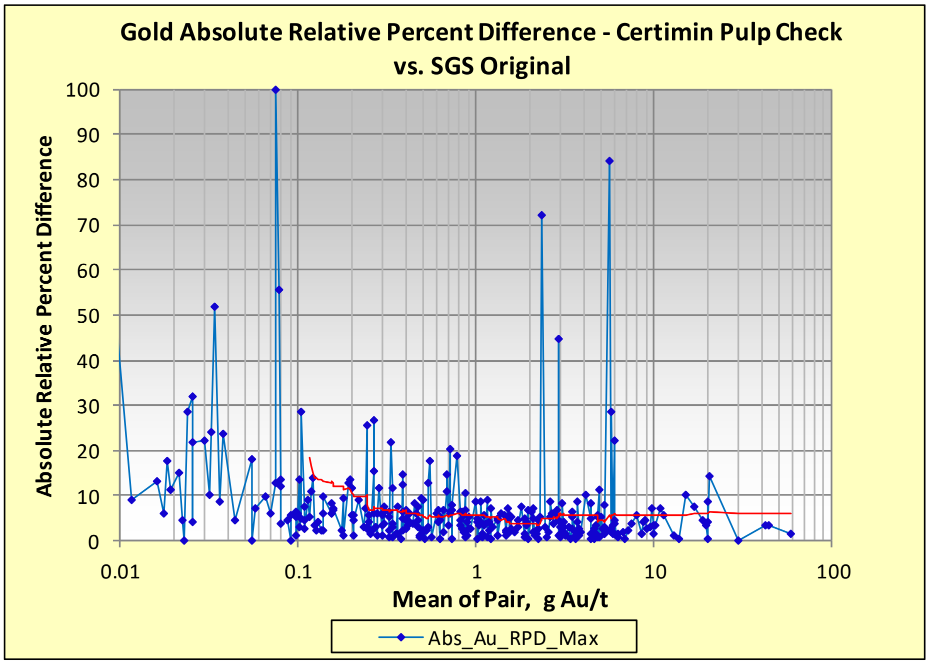


Table 12.9 Second-Lab Same Pulp Checks of SGS Drill Hole Silver Assays

Year	Second Lab	Ag g/t	count	mean orig.	mean dupl.	diff%	mean rel_diff	mean abs_rel_diff
2012	Inspectorate	>10.0	132	439.7	444.4	1.1	-1.3	5.7
2012	Inspectorate	>100	51	1077.4	1090.0	1.2	-2.8	5.2
2018	Certimin	>10.0	168	123.6	122.3	-1	0.2	6.6
2018	Certimin	>100	42	404.9	397.6	-1.8	-2.1	4.5

12.6.5 Discussion of QA/QC Results

QA/QC samples have been inserted into the sample stream on a very regular basis with multiple blanks, CRMs, or field duplicates within each laboratory batch. There is no evidence of significant cross contamination during sample preparation, although there were occasional blank failures. The failures, which are mostly very low-grade, are not considered to have a material negative impact on the use of the assay data in resource estimation and classification.

The failure rate for the CRMs is less than 1% with all failures (a total of six) occurring in the early 2008 to 2012 drilling. There have been zero failures in the silver CRM analyses.

Quarter-core field duplicates show a low mean bias (-8% to -15%) in the duplicate gold results relative to the original assays over the mid- and high-grade portions of the data. Total variability is at 25% to 60% between the original and quarter-core samples. At higher grades near 1.0g Au/t and above, the variability starts to level out at about 35%. This amount of variability is not unusual in high-grade gold deposits.

Second-lab same-pulp duplicate assays show close agreement with the original lab analyses with no material bias (less than 5%) and the total variability averaging less than 10% for all laboratory comparisons.

12.7 Summary Statement on Data Verification

Mr. Tietz believes the drill sample data is of good quality and representative of the deposit geology and mineralization. There are no material concerns arising from the data audit or QA/QC analyses. Accordingly, the data is of sufficient quality for use in the resource estimation.

The data and reports concerning the mine and process designs, capital and operating costs, and economic analyses have been reviewed for reasonableness and are considered of sufficient quality mine design and in accordance with an engineering level required for the NI43-101 Pre-Feasibility Technical Report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING (ITEM 13)

This section was prepared under the supervision of Dr. Diogenes Uceda, P.Eng. and Consulting Metallurgist for PPX. Dr. Uceda has reviewed the available results from tests performed on the Igor project samples by METTS Asociados (“METTS”) of Lima, Perú. The original testing reports were provided in Spanish. The information summarized in this section of the report represents the current understanding of the metallurgical characteristics of the Callanquitas gold – silver deposit. The term “ore” used in this report refers to the representative mineralized material samples used for the laboratory metallurgical tests, and for the potential feed to the mineral processing facility considered in this PFS. In cases where the original test parameters or results were reported in Imperial units of measure, those units are retained to avoid unintended changes in precision and rounding of the results.

A cyanide heap-leaching operation for the entire life-of-mine (“LOM”) is considered to be a favorable option from the metallurgical point of view. The analysis of the Igor project is based on average grades of 4.05g Au/t and 90.5g Ag/t over the LOM. Typically, medium- to high-grade gold deposits can be processed by conventional cyanide tank leaching. However, the metallurgical characteristics of the Callanquitas ore have led to the selection of heap leaching as an economically viable process for this deposit. PPX’s strategy is to implement a crushing circuit, a cyanide heap-leach operation, and a Merrill-Crowe recovery circuit which will be operated until the end of the mine life. The final product will be gold-silver doré that will be shipped off-site.

13.1 2014 - 2015 Testwork

PPX commissioned preliminary metallurgical tests in four laboratories during 2014 and 2015 to investigate ore mineralogy and the response of Callanquitas gold and silver mineralization to cyanide leaching. These tests included bottle-roll, agitated-tank and column-leach tests which provided basic information on parameters such as grind size, leach time, and reagent consumption. A gravity-separation test was conducted using a Falcon concentrator and a measurement of Bond work index was performed. The tests were carried out at:

- OMI Mining (“OMI”) in Lima, Perú - bottle-roll cyanide leach, gravimetric separation, and Bond work index;
- Certimin S.A. (“Certimin”) in Lima, Perú - bottle-roll cyanide leach;
- Alex Stewart International (“Alex Stewart”) in Perú - bottle-roll cyanide leach; and
- BMI Ingenieros S.A.C (“BMI”) in Lima, Perú - bottle-roll cyanide leach, agitated-tank and column cyanide leach, and gravimetric by Falcon concentrator.

Also during 2015, a detailed mineralogic study was conducted to characterize the deportment of gold, silver and other minerals in various size fractions from representative samples.

13.1.1 2014 – 2015 Column-Leach, Agitated Tank and Bottle-Roll Tests

The samples used for the 2014 – 2015 tests consisted of “oxide samples obtained by drilling”. A total of four column-leach tests, three agitated tank-leach tests, and three bottle-roll tests were conducted in

2014-2015 with oxidized material from the Callanquitas Este zone. These results are summarized in Table 13.1.

Table 13.1 Summary of 2014-2015 Bottle-Roll and Column-Leach Tests

Laboratory (Head Grade)	Type	Extractions (%)		Leach Time	Date
		Au	Ag		
OMI (Au:12.38 g/t, Ag: 35.98 g/t)	Bottle Roll	96.10	89.20	96 h	May 2014
Certimin (Au: 10.80 g/t, Ag: 39.60 g/t)	Bottle Roll	87.48	66.11	24 h	Jun 2014
Alex Stewart (Au: 19.26 g/t, Ag: 138 g/t)	Bottle Roll	81.37	7.98	120 h	Ago 2014
BMI (Au: 15.68 g/t, Ag: 47.10 g/t)	Column Leach	83.70	28.57	24 h	May 2015
BMI (Au: 10.34 g/t, Ag: 61.78 g/t)	Column Leach	88.30	77.73	48 h	May 2015
BMI (Au: 15.68 g/t, Ag: 47.10 g/t)	Column Leach	76.05	19.68	10 d	Mar 2015
BMI (Au: 15.68 g/t, Ag: 47.10 g/t)	Column Leach	85.40	14.72	10 d	Jul 2015

From the tests carried out at BMI in March 2015, gold extraction of about 76%, and silver extraction of about 19.7% was achieved after 10 days of leaching. By increasing the time to 20 and 30 days, extraction for both gold and silver remained almost constant. For the BMI column-leach tests, crushed ore at 100% -1/2 inch with agglomeration of material less than ¼ inch was utilized. Another test carried out in July 2015 with the same type of material resulted in extractions of 85.4% gold and 14.7% silver after 10 days of leaching. The better gold extraction could be due to the fact that the fines were not agglomerated as in the previous test.

The bottle-roll tests performed at Certimin obtained recoveries of about 87.5% gold and 66.1% silver after 24 hours leach time. Increasing the leach time to 48 hours and 120 hours showed no significant increases in gold recovery. In contrast, after 48 hours of leaching, there was an increase in the silver recovery to 77.7%. The granulometry of the sample was in the order of 70% at -200 mesh (Tyler). At higher percentages of -200 mesh fines, greater percentages of other elements and higher cyanide consumption were observed.

The results of testing by OMI (Quintanilla, 2014) are interesting since they obtained high recoveries of gold and silver, (96% and 89%, respectively), in comparison with the results from the other laboratories where gold recovery was approximately 88% and silver was 66%. OMI used a sample with a coarser granulometry of 85% at -200 mesh and the addition of silver nitrate in the ranges of 262 g/t. In addition, the granulometric analysis confirmed that there was an important concentration of gold and silver in the fine (-400 mesh) portions of the sample. The OMI results need to be examined further in future testwork to determine optimal leach characteristics of the Callanquitas ore.

Based on the testing, we can infer that is possible to obtain high gold extractions, around 80%, through cyanide heap leaching.

13.1.2 2014 OMI Bond Work Test

OMI utilized the same samples for Bond work index ("W_i") measurements. A measurement of 10.9 kwh/t was reported by Quintanilla (2014). Dr. Uceda has no information on the nature or location of the sample, or the representativeness of the sample.

13.1.3 Gravity Separation Test

During 2014 or 2015, a gravity separation test was conducted at BMI (Benites Malpica, 2015). The author has no information on what type of sample was tested, the location or size of the sample, and no information on the sample representativeness. Gravity separation test results were interesting, as a fairly high-grade gold-silver concentrate was produced. The gravity concentrate contained 117g Au/t and 583g Ag/t with 0.7% recovery of mass. Extraction of gold and silver in the gravity concentration test were 8% and 6%, respectively. The results suggest that there is a potential for the presence of free gold or electrum, which can create the so-called “nugget” effect or sampling and analytical inconsistencies. This can explain the scatter in the head grade analyses for the same material for different tests.

13.1.4 2015 Microscope Mineralogical Study

Reports of microscope mineralogical studies performed for PPX and BMI were reviewed. There were 21 samples examined by optical and or scanning-electron methods. The samples examined were identified as “K_ Number” and “I_ Number’ samples. There was no information or correlation between the samples examined in the mineralogical study and the 2017-2018 metallurgical program. In general, the samples contained hematite, goethite, and limonite. Rutile and clay minerals were also identified. Sulfide minerals included pyrite and arsenopyrite. There was no gold observed but a few samples contained silver as native silver and silver sulfide minerals.

Another optical microscopic study of thin sections and polished sections made from a mineralized sample referred to as “Sample 1” was performed by Cesar Cánepa, an independent consultant of Lima, Péru (Cánepa, 2015). The sample was collected from the surface by PPX geologists. The report refers to the qualitative and quantitative study, by optical microscopy, of four granulometric fractions obtained from the oxidized Sample 1 with silver and gold values. The objectives of the study were to characterize the mineralogy and determine the degree of liberation of the present mineral so as to suggest the method of adequate recovery. The report is illustrated with photographs that highlight the most outstanding mineralogical characteristics; the graphic scale of each photo is expressed in microns.

Granulometry and Chemical Composition of the Sample

The results of the chemical and granulometric analysis of Sample 1, according to information provided by PPX, are those shown in Table 13.2, in which the selected microscopic fractions are also indicated, both for the study in polished section and in thin section.

Table 13.2 Petrographic Sample Size Fractions Studied 2015

Size Fraction	Study Type	Weight (%)	Grades (g/t)	
			Au	Ag
+60 mesh	Q & Q	47.43	16.10	47.00
+100 mesh	Unk	14.62	13.16	40.00
+200 mesh	Q & Q	13.51	11.34	34.80
+400 mesh	Q & Q	12.67	13.22	38.10
-400 mesh	Q	11.77	18.15	82.40
Calculated		100.00	14.90	47.37
Assayed			15.68	47.10

Note: Q & Q = quantitative and qualitative; Q = qualitative; Unk = unknown

Regarding the granulometry of the sample, the results indicate a material with a large fraction of coarse grains (slightly more than 75% of the total weight of the sample does not pass the 200 mesh), as a consequence of the size reduction method used (crushing in open circuit). The sample has high values of gold and silver, the only two elements analyzed. The variations of gold and silver grades, as a function of the granulometry, describe inverted parabolas, with minimum values in the fraction +200 mesh and maximum values in the -400 mesh fraction that present a clear increase in comparison with the +60 mesh fraction (12.5 % in the case of gold and 75% in the case of silver).

Methodology

A representative portion was taken from each of the four selected size fractions to prepare the respective polished or thin section. The description of the results of the microscopic study of the various fractions of the single sample includes the qualitative and quantitative aspects, except in the -400 mesh fraction, in which technically only a qualitative study is possible. Quantitative results, summarized in the respective tables, include the percentages of abundance (volume%) of the mineral species present, both in the form of "free particles" (in which a mineral species constitutes more than 90% of the area of the particle), and in the form of "mixed particles" (constituted by the association of two or more mineral species and where each of them occupies not less than 10% of the total area).

Additionally, the quantitative microscopic study allows the morphological characterization of each mixed particle by means of the simultaneous determination of the percentages of area and periphery occupied by each participating mineral species. With such data, the degree of partial release corresponding to each mineral species in each type of mixed particle that would be the average obtained for the set of mixed particles of a given type, and that appears as a value in parentheses has been calculated. In this way, the degree of partial release turns out to be an indicator of the reactivity, compared to a certain type of concentration process, of each of the associated mineral species. The degrees of partial release of the mixed and free particles (whose degree of partial release is logically considered as 100) weighted by the respective percentages of abundance, allows for the calculation of the degree of total liberation (“G.L.”) for each mineral species. The calculation is made mesh size by mesh size. The degree of total release is thus an objective expression of how the release of a mineral species varies from one size fraction to another, depending on the size of the grains. On the other hand, using the respective specific weights, the percentages in volume can easily be converted into percentages by weight. Details of the calculation of the degree of partial and total release for each mineral species can be found in Cánepa and Manzaneda (2005).

13.1.4.1 Qualitative Mineralogical Characteristics

Metallic minerals were identified within the gangue, which is predominantly composed of quartz, as follows:

- **Sphalerite** is scarce and usually occurs as inclusions within quartz, with sizes <50 microns; sometimes they appear as free particles or forming ties with the gangue;
- **Pyrite** is quite scarce and generally occurs as disseminations within quartz, with sizes generally <30 microns; very occasionally, with the gangue;

- **Chalcopyrite** is extremely scarce. Like pyrite, it occurs mostly as fine inclusions within quartz, with general sizes <20 microns, and very occasionally as free and associated with gangue;
- Within the quartz grains occur some inclusions of such fine size (<10 microns) that their exact identification is not possible. Due to the morphology and brightness of the grains, the presence of **galena**, **arsenopyrite**, and probable **lead sulfo-salts** is suspected;
- **Limonites** is a generic name that is used here to include all occurrences of oxidized minerals. These oxidized minerals appear as nests of varied size, sometimes closely associated with gangue and other times individually. Some limonite appears as compact material, filling veins or forming banded aggregates; the remainder occurs as powdery material that dyes the gangue to which it is impregnated. In compact limonite, areas with very varied brightness are observed, which includes carbonaceous material. Additional oxidized mineral species (probable sulfates, arsenates, etc.) were noted, in many of them it is possible to observe relicts of sulfides from which they were formed;
- Occasionally, the occurrence of small particles of metallic material, probably from the crushing equipment, is observed; some of these particles have an oxidized edge; and
- In the -400 mesh fraction, the liberation of the limonite is nearly complete and there is a clear predominance of the limonite over the gangue. Occasionally, some free pyrite and sphalerite grains were observed.

13.1.4.2 Quantitative Mineralogical Characteristics

The quantitative study results from polished sections are those shown in Table 13.3; these results correspond to three granulometric fractions that together represent more than 72% of the total weight of the sample. Within the free particles, only the presence of gangue and limonite was detailed, which means that the other minerals identified were in amounts <0.05%. Limonite percentages increase steadily from coarse to finer fractions, while gangue minerals (which are more abundant) show a positive parabolic variation curve, with its highest percentage in the +200 mesh fraction.

The mixed particles consist only of limonite/gangue tie-ups and are very abundant in the +60 mesh fraction where they represent 46.3% of the total volume of particles. However, they decrease sharply towards the finer fractions and in the fraction +400 mesh they only constitute 5.5% of the total volume.

Table 13.3 Summary of Optical Microscopy, Sample 1

Sample: Head oxides									
	Mesh +60 (47.43%)			**Mesh +200 (13.51%)**			**Mesh +400(12.67%)**		
Free particles	**∑**	**LMs**	**GGs**	**∑**	**LMs**	**GGs**	**∑**	**LMs**	**GGs**
LMs	17.10	17.10		26.65	26.65		41.30	41.30	
		(100)			(100)			(100)	
GGs	36.60		36.60	56.25		56.25	53.25		53.25
			(100)			(100)			(100)
∑	53.70	17.10	36.60	82.90	26.65	56.25	94.55	41.30	53.25
Mixed particles									
LMs/GGs	46.30	23.25	23.05	17.10	8.10	9.00	5.45	2.35	3.10
		(34.70)	(24.10)		(31.65)	(30.30)		(27.30)	(33.40)
Total (vol %)	100.00	40.35	59.65	100.00	34.75	65.25	100.00	43.65	56.35
G.L. (%)		62.37	70.67		84.07	90.39		96.09	96.34

LMs = Limonite; GGs = Gangue; G.L. = Degree of liberation

13.1.4.3 Comments on the Microscopic Mineralogy Results

The results of the Cánepa (2015) study imply that the mineral carrying the gold and silver is limonite. The remaining metallic minerals identified (pyrite, sphalerite, and chalcopyrite) or possibly present (arsenopyrite, galena, lead sulfosalts) do not have a relevant importance as carriers of gold and silver. In view of these results, it is advisable to include routine analyses for copper and zinc, given the presence of sphalerite and chalcopyrite, and for lead, arsenic, antimony and bismuth given the probable presence of arsenopyrite, galena and sulfosalt.

The granulometric data indicates that in the +60 mesh fraction there is a very important percentage of "-grain" material (47.4% of the total weight of the sample). On the other hand, the results indicate that in this fraction there is a strong proportion of mixed particles (46.3% of the total volume of particles). Alternatively, limonite constitutes 17.1% of the volume of free particles and 23.3% of the volume of mixed particles. Gangue constitutes 36.6% of the volume of free particles and 23.1% of the volume of mixed particles. Limonite totals 40.4%, both in the form of free and mixed particles. In the case of the gangue, the sum of both percentages represents a total of 59.7%. Finally, in this fraction the degree of total liberation of the limonite reaches a moderate value of 62.4%.

The +200 mesh fraction represented 13.5% of the total weight of the sample. Conversely, in this fraction there was a strong decrease in the proportion of mixed particles (17.1% of the total volume of particles). On the other hand, limonite constitutes 26.7% of the volume of free particles and 8.1% of the volume of mixed particles. Instead, the gangues constitute 56.3% of the volume of free particles and 9.0% of the volume of mixed particles. Adding the percentages of limonite both in the form of free and mixed particles, there is a total of 34.8%. In the case of the gangue, the sum of both percentages represents a total of 65.3%. Finally, in this fraction the degree of total liberation of the limonite increases up to 84.1%.

The granulometric data of the sample indicates that the +400 mesh fraction represents 12.7% of the total weight of the sample. In this fraction, the proportion of mixed particles decreases (5.5% of the total volume of particles). Limonite constitutes 41.3% of the volume of free particles and 2.4% of the volume of mixed particles. Conversely, gangue constitutes 53.3% of the volume of free particles and 3.1% of the volume of mixed particles. Adding the percentage of limonite, both in the form of free and mixed particles, there is a total of 43.7%. In the case of gangue, the sum of both percentages represents a total of 56.4%. In conclusion, in this fraction the degree of total liberation of the limonite increases up to 94.6%.

The granulometric data indicates that the -400 mesh fraction represents 11.8% of the total weight of the sample. The qualitative results indicate that in this fraction a practically total release of limonite is achieved and that the abundance of limonite exceeds that of gangue.

If the chemical-granulometric data of Table 13.2 is compared with the quantitative mineralogical results of Table 13.3, a close correlation is clearly observed between the abundance of limonite and the grades of gold and silver. In effect, the parabolic variation of the grades is reflected exactly in the total volume of limonite that decreases from the +60 mesh fraction to the +200 mesh, and then increases in the +400 mesh fraction. This is evidence that oxidized minerals (generically referred to as limonite) are the fundamental carriers of gold and silver. Additionally, a very important result is that the very coarse or very fine fractions are those that contain the highest grades, independently of the larger or minor release of the limonite.

In conclusion the most advisable metallurgical treatment for the recovery of gold and silver is cyanide leaching. According to the results above, two types of metallurgical processing could be carried out: heap leaching, with material that is not fragmented; and leaching in tanks with 80% -200 mesh ground material. It is important to complete the chemical characterization of the material by carrying out complementary analyzes by Cu, Zn, As, Sb, Pb and Bi, with a view to controlling eventual problems derived from the presence of Cu and / or arsenate-antimonite compounds.

13.2 2017 Testwork

The main objective of metallurgical studies in 2017 was to confirm the metallurgical response to cyanide heap leaching of two different types of Callanquitas mineralized materials. Composite samples identified as M1 and M2 were prepared from drill core (METTS, 2017). Using these results, a process can be designed based on the parameters such as: granulometry of the ore, use of agglomeration with cement, curing of the crushed and agglomerated mineralization, consumption of reagents in kg/t (lime, cyanide and cement), leaching rate (L/h/m^2), concentration of cyanide in the leaching solutions, leaching period (days), percentage of gold and silver extraction, and distribution of gold and silver in the pregnant leach solution (“PLS”).

The 2017 results indicate that it is feasible to reach gold extractions of 80% for the M2 sample at granulometries lower than 12mm (½ inch) using agglomeration and cyanide leaching at concentrations of sodium cyanide (“NaCN”) of 2,000 ppm and a leaching rate of 6.0 L/h/m^2. The gold extractions for the M1 composite are close to 65 to 67% under the same conditions evaluated to the M2 composite. It is suggested that the mineralized material should be crushed to a smaller granulometry to achieve important increases in the extraction of gold and silver.

The granulometric evaluation carried out for both composites (M1 and M2) concluded that gold is distributed in the larger grain size ranges in both the M1 and M2 samples, with important values of about 60% in the +6mm (+ ¼ inch) size fraction. The differences between both ore types are given by the silver contents. The M1 composite has much lower silver (48g Ag/t), while M2 averages 141g Ag/t.

Due to the generation of large a percentage of fines (percentage -12mm), the mineralized material may require a previous treatment with cement for agglomeration, as well as curing of the agglomerated material. The curing is necessary to improve the kinetics of the cyanidation leaching. The irrigation rate was 6.0 L/h/m^2 for materials that have a high content of fines as is the case for the M2 composite. There is no risk of flooding and that the NaCN solutions would not percolate through the heap because of impervious areas or "blind" zones formed.

The low cyanide consumption of less than 1.5 kg/t, for both composites M1 and M2, is an indication that there is no presence of elements that consume NaCN (cyanicides) such as arsenic, and base-metal elements (Cu, Pb, Zn). Material with finer granulometries (100% less than 8 to 10 mm) should be irrigated at a low leaching rate and NaCN concentrations of 2,000 ppm.

Because the best results were obtained for granulometries of 100% at -12.5 mm, it was proposed to reduce the granulometry to 100% at -9 mm to obtain better recoveries of gold. Future testwork should be completed for 100% at 9 mm granulometries to see the results in terms of gold and silver recoveries.

13.2.1 M1 and M2 Composites Preparation

Two composites were prepared at the METTS laboratory in Lima, Perú using assay coarse rejects of ½-core samples from core holes drilled in the Callanquitas area in 2011 and 2012. Details on the intervals selected, their lengths, gold and silver grades, sample numbers, oxidation states, and shipping were documented by Gutierrez and Chavez (2017). The composites were identified as “M1 (Au)” and “M2 (Au/Ag)”, which were composed of groups of core samples identified as “Lot 1” and “Lot 2”, respectively, as shown in Table 13.4.

METTS received the samples for composites M1 and M2 and provided a sampling report that was shared with PPX. Within the scope of the METTS testwork, the sampling protocol was designed to generate representative samples for each composite. According to METTS (2017), the samples were homogenized and cone and quartered to form the composites with the flow sheet shown in Figure 13.1.

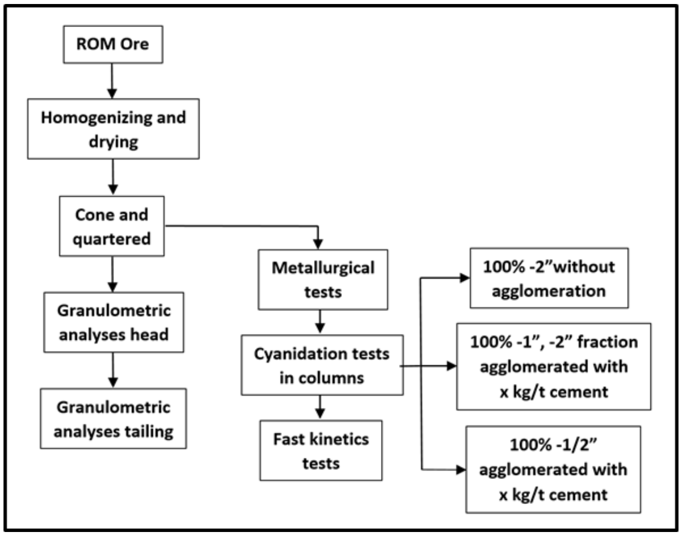
Chemical analyses for the samples used to make the composites were provided by MinLab in Lima, Perú, and by AGQ Labs in Lima, Perú. The individual assays for each sample are listed in Table 13.4 The overall calculated head assays, obtained from the individual sample assays, were 4.13g Au/t and 25g Ag/t for composite M1 and 3.09g Au/t and 239g Ag/t for composite M2.

Table 13.4 2017 – 2018 Composites at METTS

Comp M1 - Au				
Number	Lot 1	Weigh (Kg)	Au, g/t	Ag, g/t
1	CA-12-30	26.8	13.01	58.9
2	CA-11-07	31.2	4.12	45.0
3	CA-12-42	19.0	2.27	15.1
4	CA-12-43	14.0	6.39	10.5
5	CA-11-10	42.3	2.82	6.4
6	CA-12-38	8.6	3.25	7.1
7	CA-12-20	19.8	3.43	52.1
8	CA-12-54	3.3	3.73	4.0
9	CA-12-52	73.7	2.06	15.4
10	CA-12-57	1.4	2.38	49.9
	CALC HEAD	**240.1**	**4.13**	**25.0**

Comp M2- Au/Ag				
Number	Lot 2	Weight (Kg)	Au	Ag
1	CA -12 - 53	69.5	3.93	73
2	CA - 12 - 31	47.2	2.77	129
3	CA - 11 -08	17.8	4.33	578
4	CA - 11 - 13	8.2	2.98	1446
5	CA - 12 - 47	46.0	0.76	300
6	CA -12-48	13.8	4.04	143
7	CA - 12 - 27	42.8	3.34	263
8	CA -12- 44	9.8	4.87	64
	CALC HEAD	**255.1**	**3.09**	**238.7**

Figure 13.1 Flow Sheet for 2017 – 2018 METTS Composites



13.2.2 M1 and M2 Composite Head Analyses

METTS received additional samples from PPX and blended them into composites M1 and M2. The composites' calculated and direct head assays (METTS, 2017) are shown in Table 13.5. In addition, size fraction analysis ("SFA") was conducted on the composite samples as they were received from the mine, as estimated to be the run-of-mine ("ROM") size. The calculated overall head grades obtained from the SFA are shown in Table 13.5 for comparison with the direct head grades. The calculated head grades from the metallurgical balances from the column tests are also shown. The variance in the silver grade has been noted. This is probably due to different analytical methods applied, but this variance will need to be verified and investigated.

The SFA results from composite M1 indicated that the coarse fractions (from 9 mm to 75 mm) contained relatively low gold (0.3-2g Au/t) and silver (2-11g Ag/t) concentrations. However, approximately 32% of the gold and 14% of the silver were contained in these fractions. These results suggest that the coarse fractions would need to be crushed down in a heap-leach scenario in order to expose the contained gold and silver.

Table 13.5 METTS Composite Head Assays

Comp	Results	Comp Weight kg	Assays, g/t	
			Au	Ag
M1 (Au)	Calc Comp Head Assays	324	4.13	25
	Direct Head Assays	-	4.34	48*
	Calc from Comp SFA		2.50	22
	Calc Ave from Columns Met Balances	-	5.02	40
M2 (Au/Ag)	Calc Comp Head Assays	309	3.09	239
	Direct Head Assays	-	3.78	141*
	Calc from Comp SFA		4.72	172
	Calc Ave from Columns Met Balances	-	4.97	155

* ICP Scan Assays

The concentration of gold and silver in the finer fractions increased approximately to the average head grade. There were a few spikes in the middle fractions. For example, the -10 mesh fraction reported 6g Au/t and 105g Ag/t, which should have been further investigated and re-assayed. The finest fraction of -325 mesh contained 7% of the gold and 15% of the silver. It is unlikely that this portion of gold and silver will be recoverable by direct cyanidation at a conventional grind size.

The SFA results for composite M2 also showed substantial spikes in the analyses for a number of fractions. For example, the minus 25 mm fraction assayed 19g Au/t and 1,508g Ag/t. These results indicate that about 44% of the gold and 68% of the silver reported into the coarsest fractions and about 6% of the gold and 9% of the silver reported into the -325 mesh fraction, which will not be recoverable by conventional cyanidation.

The head samples were also submitted for a multi- element ICP scan; elevated levels of arsenic, iron, and lead were found (Table 13.6). Elevated levels of copper and antimony were noted in other samples, which were analyzed previously, but not included in this investigation. The presence of elements such as iron, copper, and zinc can result in high cyanide consumption during leaching. The extraction of arsenic, antimony and other metals can result in some challenges for the downstream effluent treatment.

There were no sulfur and carbon speciation assays available for review. These elements are important to evaluate, as the mineralogy examination (Section 13.1.4) identified the presence of sulfides and organic carbon in a number of samples examined. Whole rock analysis is also recommended.

Table 13.6 ICP Analyses of 2017 – 2018 Composites

ELEMENT	Units	MIN DL	MAX DL	M1 - ICP	M2 - ICP
Ag	ppm	1	350	46.3	140
Al	%	0.01	15	0.27	0.14
As	ppm	3	10000	9220	5995
Ba	ppm	5	10000	77	71
Be	ppm	0.5	10000	<0.5	<0.5
Bi	ppm	5	10000	55.8	37.6
Ca	%	0.01	15	0.06	0.02
Cd	ppm	1	10000	1	<1
Co	ppm	1	10000	1	2
Cr	ppm	1	10000	34	43
Cu	ppm	0.5	10000	129.8	121.2
Fe	%	0.01	15	13.73	7.42
Ga	ppm	10	10000	<10	<10
Hg	ppm	1	10000	3	3
K	%	0.01	15	0.17	0.07
Mg	%	0.01	15	<0.01	<0.01
Mn	ppm	2	10000	29	28
Mo	ppm	1	10000	5	3
Na	%	0.01	15	0.01	<0.01
Ni	ppm	1	10000	3	6
Pb	ppm	2	10000	2256	4369
Sb	ppm	5	10000	465	426
Sc	ppm	1	10000	1	<1
Se	ppm	10	1000	<10	<10
Sn	ppm	10	1000	27	32
Sr	ppm	0.5	5000	7.7	7.7
Ti	%	0.01	15	<0.01	<0.01
Tl	ppm	2	10000	<2	<2
V	ppm	2	10000	31	18
Zn	ppm	0.5	10000	112.3	149.4

13.2.3 Characterization of Grades by Size Fraction

13.2.3.1 Grain Size Analysis

The granulometric analysis was performed on the composite samples M1 and M2 as they were received from the mine (it is estimated to be ROM size). Samples from each composite were sieved using screens of 76.2 mm, 50.8 mm, 25.4 mm, 19.05 mm, 12.7 mm, and 6.35 mm sizes. Sieves with Tyler mesh sizes 10, 20, 40, 80, 100, 140, 200, 270, 325, and - 325 were also used. All fractions retained were analyzed for gold and silver. The results of this analysis are presented in Table 13.7.

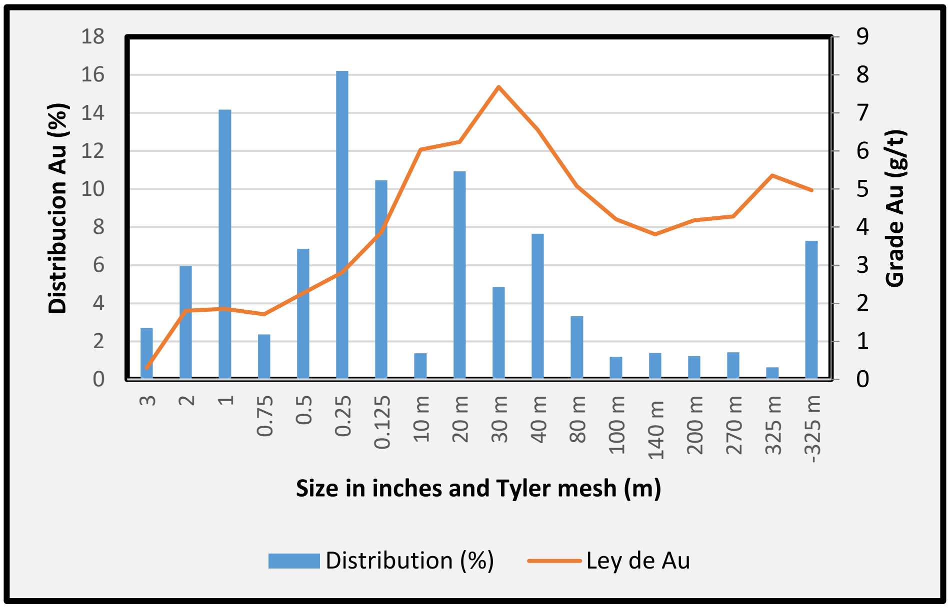
Table 13.7 Granulometric Analysis of Sample from Composite M1

Malla Tyler	Analisis de mallas					Analisis (gr/ton)		Distribucion (%)			
	Rango de tamaños	Peso de muestra, kilogramos	Distribucion Peso, %	Peso Retenido Acum, %	Peso pasante Acum., %	Au	Ag	Au	Cum	Ag	Cum
3	+75	1.125	22.19%	22.19%	77.81%	0.304	2.0	2.70	2.70	2.06	2.06
2	- 75 + 50.8mm	0.419	8.26%	30.45%	69.55%	1.805	4.0	5.97	8.66	1.54	3.60
1	- 50.8 + 25.4mm	0.970	19.13%	49.59%	50.41%	1.853	6.0	14.18	22.84	5.34	8.94
0.75	- 25.4 + 19.05m	0.175	3.45%	53.04%	46.96%	1.711	6.0	2.36	25.20	0.96	9.90
0.5	-19.05 + 12.7mm	0.384	7.57%	60.61%	39.39%	2.267	11.0	6.87	32.07	3.87	13.78
0.25	-12.7 + 6.35mm	0.730	14.40%	75.01%	24.99%	2.814	31.0	16.20	48.27	20.75	34.53
0.125	-6.35 + 3.18mm	0.343	6.77%	81.78%	18.22%	3.864	63.0	10.45	58.72	19.82	54.35
10 m	- 3.18 + 2.00mm	0.029	0.57%	82.35%	17.65%	6.036	105.0	1.38	60.10	2.79	57.14
20 m	-2.00 + 0.85mm	0.222	4.38%	86.73%	13.27%	6.239	46.0	10.92	71.03	9.37	66.51
30 m	-0.85 + 0.60mm	0.080	1.58%	88.30%	11.70%	7.681	56.0	4.85	75.88	4.11	70.62
40 m	-0.60+ 0.43mm	0.148	2.92%	91.22%	8.78%	6.559	59.0	7.66	83.53	8.01	78.63
80 m	-0.43 + 0.18mm	0.083	1.64%	92.86%	7.14%	5.076	30.0	3.32	86.86	2.28	80.91
100 m	-0.18 + 0.15mm	0.036	0.71%	93.57%	6.43%	4.207	26.0	1.19	88.05	0.86	81.77
140 m	-0.15+0.106mm	0.046	0.91%	94.48%	5.52%	3.812	22.0	1.38	89.43	0.93	82.70
200 m	-0.106 + 0.075mm	0.037	0.73%	95.21%	4.79%	4.182	25.0	1.22	90.65	0.85	83.54
270 m	-0.075 + 0.053mm	0.042	0.83%	96.04%	3.96%	4.284	25.0	1.42	92.07	0.96	84.51
325 m	-0.053 + 0.045mm	0.015	0.30%	96.33%	3.67%	5.359	35.0	0.63	92.71	0.48	84.99
-325 m	-0.045mm	0.186	3.67%	100.00%	0.00%	4.971	88.0	7.29	100.00	15.01	100.00
Cabeza Calculada		**5.07**	100.00%			**2.501**	**21.51**	100.00		100.00	

The granulometric analysis of the M1 sample showed that about 32.1% of the gold was in the coarse-grained fragments of 76.2 mm, 50.8 mm, 25.4 mm, and 19.05 mm sizes. It should be liberated in the process of crushing to expose gold to the action of cyanide. In addition, about 7.3% of the gold was in the fine and ultrafine size ranges, less than 45 microns. Due to the lack of liberation it would not be recovered even by a process of leaching with agitation.

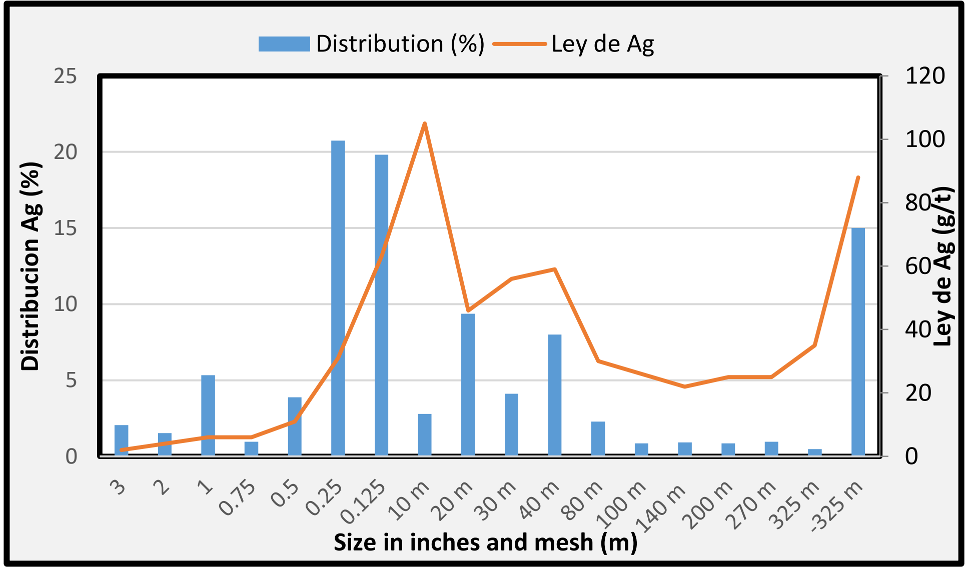
It is important to note that gold with grades less than 2.0g Au/t are found in the coarser sizes (greater than 12.7 mm). For sizes less than 12.7 mm, the gold grades vary between close to and greater than 4.0g Au/t, even in the fine grain sizes. This indicates that in order to achieve improvements in heap leaching the particle sizes of the coarse fractions must be reduced to expose the gold contained. Figure 13.2 shows a graph for the gold distribution by particle size (mesh) for the M1 sample.

Figure 13.2 Gold Grade by Size Fraction, Composite M1



From the granulometric analysis of the M1 sample, about 13.8% of the silver was associated with the coarser, + 12.7 mm granulometry and about 86.2% was distributed in the fine fractions, with 15.0% of the silver in the ultrafine sizes less than 45 microns. Due to the presence of the silver in such fine sizes, it would be impossible to recover even by leaching by agitation because the defined crushing and milling processes would provide particle sizes >45 microns. Figure 13.3 shows the silver distribution by grain size for a sample from composite M1.

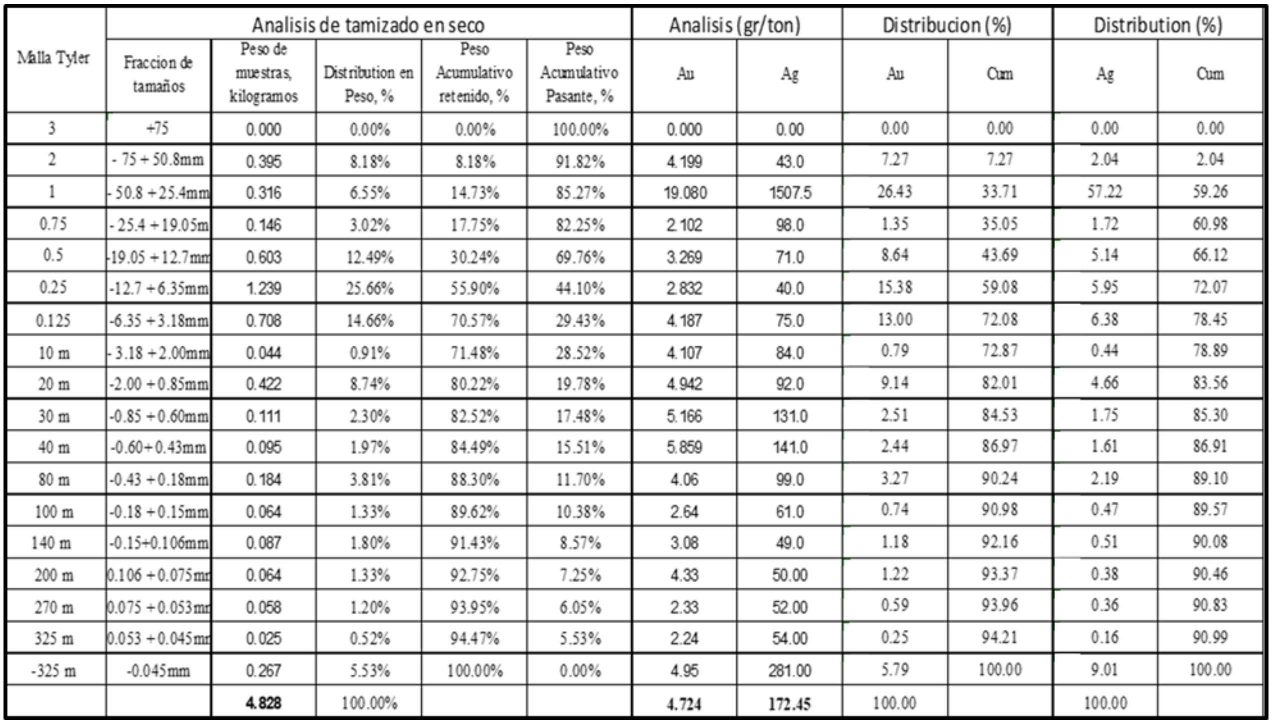
Figure 13.3 Sample M1, Silver Distribution by Grain Size



It is important to note that the silver grades, in important values greater than 20g Ag/t, were in particle sizes less than 6.3 mm. For grains greater than 6.3 mm, the silver grades were lower than 5g Ag/t (Figure 13.3).

The granulometric analysis of a sample from composite M2 is summarized in Table 13.8.

Table 13.8 Granulometric Analysis of Sample from Composite M2



Malla Tyler	Analisis de tamizado en seco					Analisis (gr/ton)		Distribucion (%)		Distribution (%)	
	Fraccion de tamaños	Peso de muestras, kilogramos	Distribution en Peso, %	Peso Acumulativo retenido, %	Peso Acumulativo Pasante, %	Au	Ag	Au	Cum	Ag	Cum
3	+75	0.000	0.00%	0.00%	100.00%	0.000	0.00	0.00	0.00	0.00	0.00
2	- 75 + 50.8mm	0.395	8.18%	8.18%	91.82%	4.199	43.0	7.27	7.27	2.04	2.04
1	- 50.8 +25.4mm	0.316	6.55%	14.73%	85.27%	19.080	1507.5	26.43	33.71	57.22	59.26
0.75	- 25.4 +19.05m	0.146	3.02%	17.75%	82.25%	2.102	98.0	1.35	35.05	1.72	60.98
0.5	-19.05 +12.7mm	0.603	12.49%	30.24%	69.76%	3.269	71.0	8.64	43.69	5.14	66.12
0.25	-12.7 +6.35mm	1.239	25.66%	55.90%	44.10%	2.832	40.0	15.38	59.08	5.95	72.07
0.125	-6.35 +3.18mm	0.708	14.66%	70.57%	29.43%	4.187	75.0	13.00	72.08	6.38	78.45
10 m	- 3.18 +2.00mm	0.044	0.91%	71.48%	28.52%	4.107	84.0	0.79	72.87	0.44	78.89
20 m	-2.00 +0.85mm	0.422	8.74%	80.22%	19.78%	4.942	92.0	9.14	82.01	4.66	83.56
30 m	-0.85 +0.60mm	0.111	2.30%	82.52%	17.48%	5.166	131.0	2.51	84.53	1.75	85.30
40 m	-0.60+0.43mm	0.095	1.97%	84.49%	15.51%	5.859	141.0	2.44	86.97	1.61	86.91
80 m	-0.43 +0.18mm	0.184	3.81%	88.30%	11.70%	4.06	99.0	3.27	90.24	2.19	89.10
100 m	-0.18 +0.15mm	0.064	1.33%	89.62%	10.38%	2.64	61.0	0.74	90.98	0.47	89.57
140 m	-0.15+0.106mm	0.087	1.80%	91.43%	8.57%	3.08	49.0	1.18	92.16	0.51	90.08
200 m	0.106 +0.075m	0.064	1.33%	92.75%	7.25%	4.33	50.00	1.22	93.37	0.38	90.46
270 m	0.075 +0.053m	0.058	1.20%	93.95%	6.05%	2.33	52.00	0.59	93.96	0.36	90.83
325 m	0.053 +0.045m	0.025	0.52%	94.47%	5.53%	2.24	54.00	0.25	94.21	0.16	90.99
-325 m	-0.045mm	0.267	5.53%	100.00%	0.00%	4.95	281.00	5.79	100.00	9.01	100.00
		4.828	100.00%			**4.724**	**172.45**	100.00		100.00	

The granulometric analysis of the M2 sample shows that about 43.7% of the gold was associated with coarse-grained fragments of 19.05 mm to 76.2 mm sizes. In addition, about 6.0% of the gold was associated with the fine and ultra-fine sizes of less than 75 microns. Gold in these particle sizes is not significant to recover economically because to achieve recoveries of 4% or 5% of gold contained in these fractions, ultrafine grinding should be used that increases processing costs.

Figure 13.4 shows a plot of gold grade by grain size in the sample from composite M2. As with the sample from composite M1, gold grades in M2 lower than 2.0g Au/t were found in the coarse grain sizes, greater than 12.7 mm. For fragments less than 12.7 mm, the gold grades varied between 4.0g Au/t to 8.0g Au/t even in the fine size ranges. This indicates that to achieve improvements in heap leaching, the particle sizes may be reduced from the coarse fractions to release the gold contained.

The granulometric analysis of the M2 sample indicates that 66% of the silver was associated with the +12.7 mm material. About 9.0% of the silver was distributed in the ultra-fine sizes of less than 45 microns. Figure 13.5 shows a plot of silver distribution by grain size in the sample from composite M2.

Figure 13.4 Composite M2 Gold by Grain Size Analysis

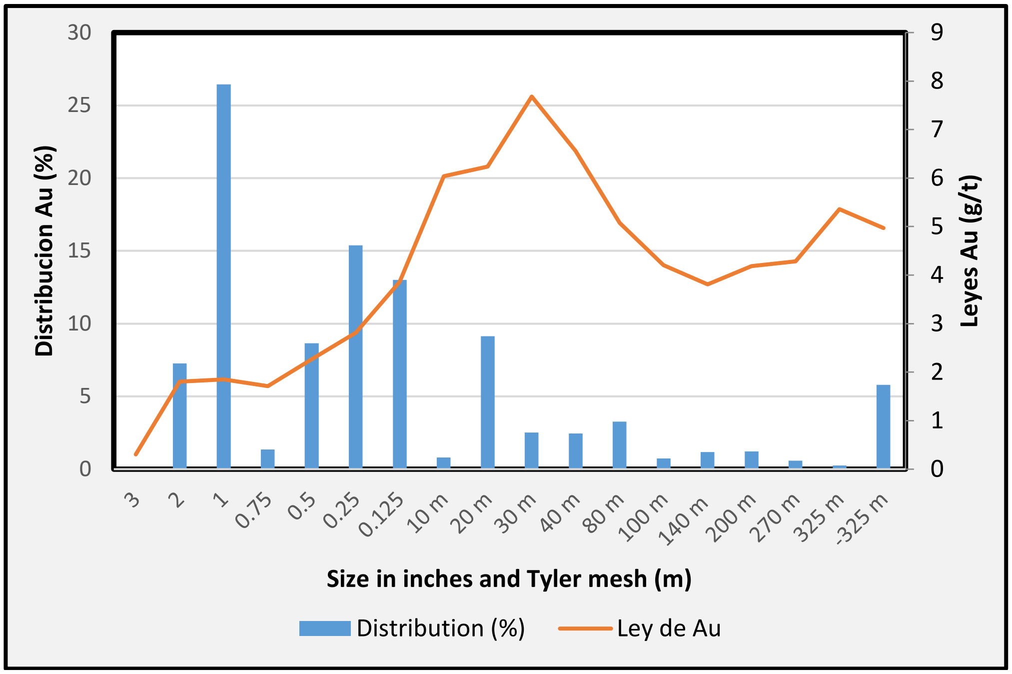
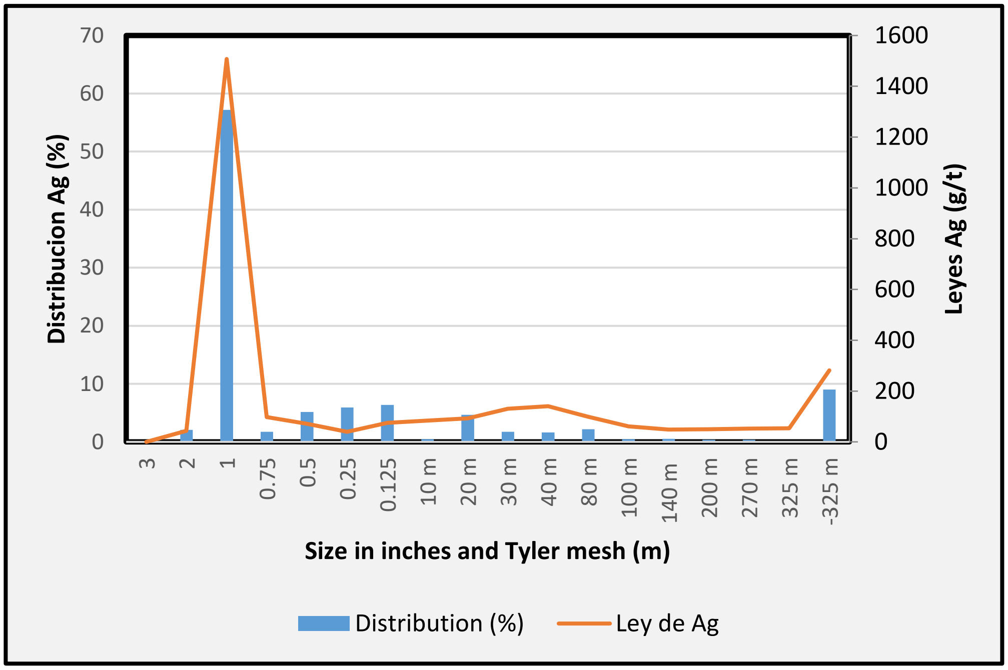


Figure 13.5 Composite M2 Silver by Grain Size Analysis



13.2.4 2017 Bottle-Roll (Quick Cyanidation) Tests

OMI conducted "quick cyanidation" tests with bottle-roll methods using samples from composites M1 and M2. The samples were reduced in particle size using jaw crushers and rollers, then sieved to 100% at -10 mesh (Tyler). For each composite, the same procedure was followed. Through successive quartering, three sub-samples were obtained, one for grindability, one for quick cyanidation testing, and a third for chemical tests.

The quick cyanidation tests were carried out in bottles on a roller table. Previously, alkalinity tests were performed in order to determine the initial lime consumption of the samples reaching the degree of protective alkalinity at a pH of 10.5. Once the pH value was reached, NaCN was added to start the test with the following parameters:

- Weight of the sample: 250 grams;
- Water volume: 500 ml;
- Granulometry: 100% -#100 Tyler;
- Test time: 2.0 hours;
- pH: 11 to 11.5;
- NaCN concentration: 0.5%, and
- Aliquot taking at the end of the test:
    - 50 ml, for chemical analysis
    - 25 ml, for titration

Table 13.9 and Table 13.10 summarize the 2017 bottle-roll results:

Table 13.9 Summary of Quick Cyanidation Gold Results

Sample	Head grade Au (g/t)		Au solution (ppm)	Dissolved Au (g/t)	Au solid residue (g/t)	Regent consumption (kg/t)		Au recovery (%)
	Tested	calculated				Cyanide	Lime	
M1	6.82	5.19	1.39	2.78	2.40	0.05	4.4	53.7%
M2	6.13	5.86	1.67	3.34	2.52	0.13	5.6	57.0%

Table 13.10 Summary of Quick Cyanidation Silver Results

Sample	Head Ag (g/t)		Ag solution (ppm)	Dissolved Ag (g/t)	Ag grade leached residue (g/t)	Reagent consumption (kg/t)		Ag recovery (%)
	Assayed	calculated				Cyanide	Lime	
M1	48	51.3	3.667	7.33	44	0.05	4.4	14%
M2	141	143.8	12.92	25.84	118	0.13	5.6	18%

The results indicated low gold and silver extractions of less than 60% for gold and 20% for silver. These were almost similar between both composites. The consumptions of lime and cyanide were low. These results indicate that the release of particles is very coarse (100% - 150 microns) and that therefore it does not reach higher gold recovery. The low consumption of cyanide indicates that there are no cyanides in the mineralized material and that greater silver liberation (100% - 75 microns) and longer leach time is required for the silver.

13.2.5 2017 Column–Leach Tests at METTS

Representative sub-samples from composites M1 and M2 underwent column-leach and agglomeration strength tests at METTS to investigate the amenability of heap-leach processing for the Callanquitas mineralized material.

Six tests, three tests on each composite, were performed in PVC columns, on approximately 85-95 kilograms of material in each column. The columns were 2.5 meters in height and 19.05 centimeters in diameter. The sample crush sizes were 100% -50.8 mm, 100% -25.4mm, and 100% --12.7 mm. The fractions finer than 12.7 mm were agglomerated with cement. The irrigation rate used was 6.0 L/hr/m^2. There was no recycle of the barren solution back to the columns. The columns were operated for 35 days and then were subjected to a wash cycle. The column-leach test conditions are summarized in Table 13.11.

Table 13.11 2017 Column-Leach Test Parameters

Sample	Columns	Conditions	Irrigation rate (L/h/m^2)	NaCN (ppm)
M1	C-1	100% -50.8 mm, w/o agglomeration	6	2000
	C-2	100% -25.4 mm”, agglomerated -12.7 mm”		
	C-3	100% - 12.7 mm”, agglomerated 100%		
M2	C-4	100% -50.8 mm”, w/o agglomeration	6	2000
	C-5	100% -25.4 mm”, agglomerated -12.7 mm”		
	C-6	100% -12.7 mm”, agglomerated 100%		

The finer fractions were agglomerated according to the standard protocol with the cement, cyanide addition (0.5 g/L), and lime. The agglomeration was conducted in large vessels placed on rolls. The agglomerated material was allowed to cure for 96 hours and then loaded into the columns.

Column leaching was carried out by pumping the cyanide solution and lime to maintain the pH at 10.5 level. The column sketch presented in METTS (2017) indicated that the cyanide solution was not recycled back to the column. Confirmatory testing should recycle the barren solution, after removal of gold and silver from the PLS, in order to ensure that the impurities build-up and corresponding cyanide consumption is properly simulated. In general, the column set-ups and operation were conducted in accordance with the standard laboratory test protocol for heap-leach amenability testing.

The 2017 column-leach test results are summarized in Table 13.12 and Table 13.13. The gold and silver extraction kinetic curves are presented in Figure 13.6 through Figure 13.8. At the completion of the leach and wash cycles, the column residues from Test 3 and Test 6 were subjected to SFAs for gold and silver. The overall SFA calculated residual grades are also shown in Table 13.12 for comparative purposes with the direct assays of the column residues.

Table 13.12 2017 Column-Leach Results for Gold

Columns	Head grade Au (g/t)		Au (g/t) in PLS	Au (%) in tailings	Au (%) Recovered	Reagents consumption (kg/t)		Leaching rate (m³/t)
	Assayed	Calculated				NaCN	Lime	
C-1	4.33	5.08	9.40	1.69	66.83	0.73	3.16	1.38
C-2		5.14	3.35	1.79	65.14	1.25	1.67	1.57
C-3		4.84	3.26	1.58	67.45	1.44	1.20	1.49
C-4	3.78	4.55	3.47	1.09	76.15	0.83	2.42	1.21
C-5		4.64	3.70	0.94	79.73	1.09	1.21	1.26
C-6		5.71	4.59	1.12	80.35	1.02	0.91	1.22

Table 13.13 2017 Column-Leach Results for Silver

Columns	Head grade Ag (g/t)		Ag (g/t) in PLS	Ag (%) in tailings	Ag (%) Recovered
	Assayed	Calculated			
C-1	48.00	41.34	3.84	37.50	9.28
C-2		39.65	3.15	36.50	7.94
C-3		38.42	2.30	35.50	7.62
C-4	141.00	160.36	32.36	128.00	20.18
C-5		142.22	32.22	110.00	22.66
C-6		162.00	53.50	108.50	33.03

Figure 13.6 METTS Composite M1 – Gold Extraction Kinetics

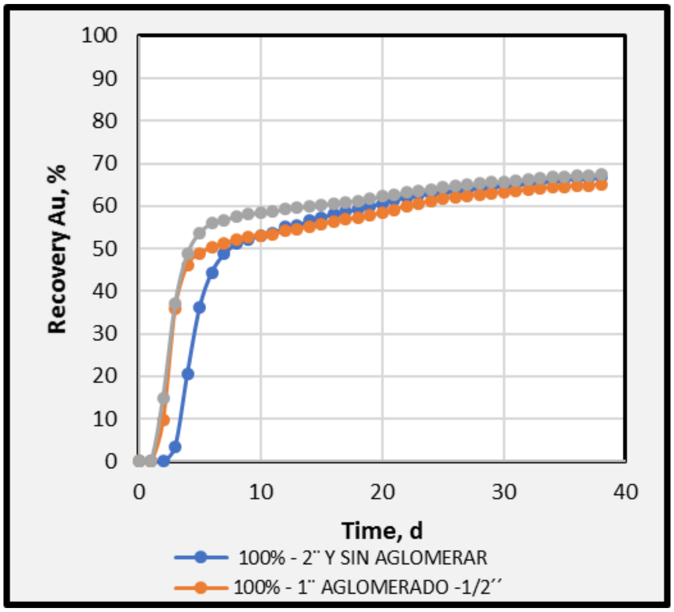


Figure 13.7 METTS Composite M2 – Gold Extraction Kinetics

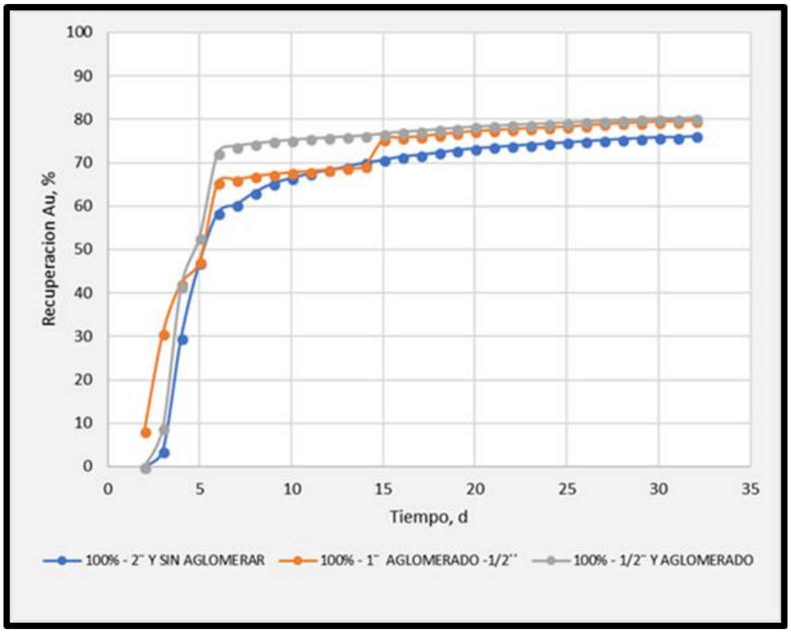
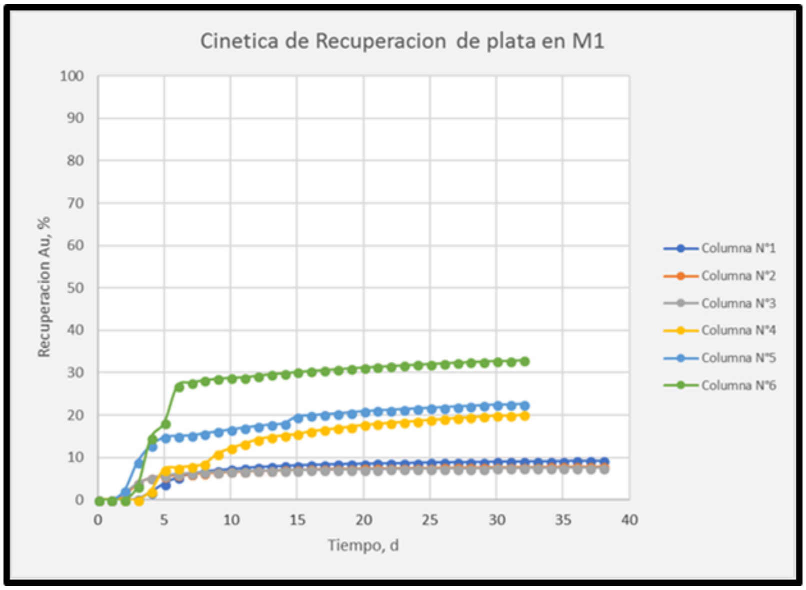


Figure 13.8 METTS Composites M1 and M2 – Silver Extraction Kinetics



Note: columna 1, columna 2 and columna 3 tested samples from composite M1; columna 4, 5 and 6 tested samples from composite M2. Columna = Spanish word for column.

The results indicate that for the sample of composite M2, about 80% of gold and 33% silver in cyanide solution is reached in a period of 32 days of irrigation. The sample from composite M1 reached lower gold extraction (around 66% in gold and 8% in silver) varying with the granulometry from 2.0 inch to ½ inch and using the agglomeration for the fines of - ½ inch. These results clearly indicate that the extraction of gold and silver could improve for particle sizes less than ½ inch, such as, for example, agglomerating 100% at - ¼ inch.

The reagent consumption was variable and increased as the particle size decreased. It can be induced that there are no elements that consume cyanide. The consumption of lime was reduced with increased amounts of cement used in the agglomerates, 1.0 to 2.0 kg/t on average.

As shown in Figure 13.6 for the sample from composite M1, gold extraction reached 65% to 67%. This behavior obeys a slow extraction and probable lack of liberation of the particle size. For the sample from M2, Figure 13.7 shows the extraction of gold reaches 76% to 80.3%. This behavior is due to a greater quantity of fines released in the sample which was agglomerated to give a better degree of percolation to the cyanide solutions.

The kinetic results indicate the rapid extraction of gold in the first five days of leaching. Extraction then increased slowly over time until reaching nonsignificant increases from day 30, where the asymptote increases slowly. The fastest gold extractions were obtained in column 3 where the material was 100% at - ½ inch and agglomerated with cement. Columns 1 and 2 had less kinetic behavior but over time were almost equal to the column with smaller granulometry.

For composite M2, there was a rapid kinetics of gold extraction in all the samples reaching higher than 70% in the first five days of irrigation for the column that had finer granulometry (100% at - ½ inch) (Figure 13.7). The columns that had coarser granulometry present lower extractions in the same period of time. The kinetic curves of the columns that utilized agglomerated material had more rapid and thorough gold extraction than did the columns of coarser material without agglomeration.

Based on the above, it is estimated that better gold recoveries could be obtained if the granulometry of the ore fed to the pads is reduced to, for example, 100% at - 6.3 mm to achieve the maximum gold and silver recovery Another variable that could be evaluated is leaching at a rate of 8.0 Lt/h/m^2, but taking special care in that this higher ratio does not produce impermeability in the column, and by extension, in the heap.

Silver extraction kinetic curves for samples from M1 and M2 are shown in Figure 13.8. The kinetic curves indicate that the silver extractions in samples from M2 increased as the particle size of the sample was reduced and the extraction of silver was fast until days 6 to 7 of irrigation (see orange curve). The other tests at 100% - 1.0 inch and 2.0 inch granulometries show similar behaviors of silver extractions. With respect to the silver extractions for samples from M1, it is either reduced to less than 10% under all agglomeration conditions and without agglomeration.

It is concluded from the results of the 2017 column-leach tests that the best results were obtained with the samples from composite M2, whose condition of leaching was with the material at 100% less than ½ inch and agglomerated at 100%. This sample had a gold head grade of 3.78 g/t, and it is possible to have a gold recovery of 80.4%, having a cyanide consumption of 1.02 kg/t and lime at 0.91 kg/t.

13.2.6 2017 Agglomeration Strength Tests

According to granulometric distribution, the material in composites M1 and M2 contained 39.3% and 69.8% at minus -½ inch size, respectively. To quantify the agglomerate strength and therefore heap height, a series of tests were carried out, the procedure being the following:

- M1 and M2 sub-samples were prepared by agglomerating with different amounts of cement (1, 3, 5, and 10 kg/t) and adding 0.5% NaCN solution to form an agglomerate with 13% humidity;
- It was cured for between 72 and 96 hours to produce the agglomerated pellets;
- After the curing time had elapsed, a sample of 1 kilogram of dry agglomerated pellets was taken (previously the pellet moisture was calculated) and placed on top of a 10 mesh screen. The mesh with the agglomerated pellets was placed in a container with water and intensively sieved for 60 seconds while submerged.
- At the end of the immersion screening, the weight retained in the mesh was removed and dried in an oven, and then weighed.

Table 13.14 shows the amounts retained by the screen, which represents a quantification of the strength of the agglomerated material. Greater strengths are represented by greater weights retained on the screen.

Table 13.14 2017 Agglomerate Strength Tests, Composites M1 and M2

Sample	Cement (kg/t)	Agglomerated Weight (kg)	Retained weight (+10 mesh)	% Retained
M1	1	0.999	0.623	62.3
	3	0.995	0.514	51.6
	5	0.998	0.492	49.3
	10	0.998	0.373	37.3
M2	1	0.898	0.444	49.4
	3	1.012	0.506	50.0
	5	1.002	0.494	49.3
	10	1.004	0.360	35.8

According to the results of the weights retained in test 10, and the conformation of the pellets of both samples, it can be stated that for the M1 sample the strength decreased as more cement was added. So, the addition of cement must be maintained between 1 and 3 kg/t. From the column tests, agglomeration of M1-type material for the heaps will be require an estimated 3.0 kg of cement per tonne.

For the M2 sample, the strength also decreased as more cement was added. The addition of cement must be maintained between 1 and 5 kg/t. From the column tests, agglomeration of M2-type material for the heaps will be require an estimated 5.0 kg of cement per tonne.

13.3 2018 Testwork at METTS

During 2018, METTS performed metallurgical tests using mixtures between higher-grade (5g Au/t) and lower-grade mineralization 2g Au/t), but with a higher proportion of coarse content, to provide information that can help to make decisions about whether or not to use agglomeration prior to heap leaching. The results were documented in the reports of METTS (2018a; 2018b). The main objective of the study was to determine the recovery of gold and silver, and consumption of reagents, using different mixtures of the higher and lower grade materials to achieve a higher degree of percolation and thus avoid the process of agglomeration. The conditions or parameters to be used were based the earlier metallurgical tests. For the 2018 tests, the granulometric characterization was made to better understand the gold and silver distribution in the different size ranges, both in the head and in the tails. The 2018 column-leach tests used mixtures of higher-grade/lower-grade at 80/20, 70/30, and 60/40. To achieve the objective of the tests, the following activities were carried out:

- Preparation of the sample: homogenization, quartering and formation of mixtures;
- Chemical analysis of the head grades of the mixtures for gold plus 32 elements by ICP;
- Granulometric analysis to evaluate the head to obtain the distribution of gold and silver in the respective size fractions;
- Cyanidation in columns of the different mixtures while maintaining constant variables such as irrigation rate (L/h/m^2) and cyanide concentration (ppm); and
- Granulometric analysis of the leached residue of the best column test, in terms of gold recovery, to obtain the distribution of gold and silver in the respective size fractions.

The following results were reported in METTS (2018a; 2018b):

- Distribution of gold and silver from the best column test in terms of gold recovery for both the head and tail materials (residue after the column test);
- Consumption of process reagents lime and NaCN;
- Rate of irrigation (L/h/m^2);
- Concentration of NaCN in leaching solution;
- Leaching cycle or irrigation period;
- Percentage of slump (settlement of the material due to irrigation);
- Percentage extraction of gold and silver obtained during the irrigation cycle; and
- Kinetic curves of extraction of gold and silver.

13.3.1 2018 Sample Preparation for Column Tests

During 2018, METTS received samples of two types of material: one of low gold grade (“LG”) and another of higher gold grade (“HG”) as listed in Table 13.15. The samples were collected by PPX from five locations in the underground workings of the Callanquitas mine in April of 2018. The gold grades of these materials were supplied to METTS by PPX; the methods of assay and their validity are not known to the author.

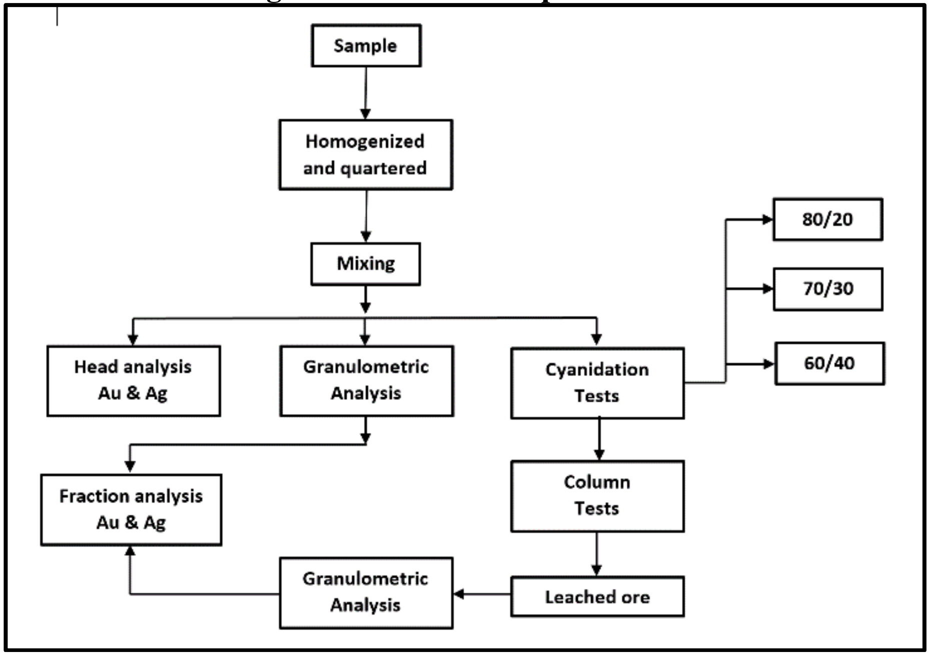
Table 13.15 2018 Low- and Higher-Grade Materials

	Units	Low Grade LG	Higher Grade HG
Weight	kg	180.00	350.00
Au	g/t*	2.69	6.80

** indicates gold grades sent from PPX*

Following the sampling protocol, the samples were distributed according to the type of metallurgical evaluation to be carried out. Figure 13.9 shows the preparation scheme for obtaining the different sub-samples used in the column-leach tests.

Figure 13.9 2018 Sample Scheme



13.3.2 Chemical and Other Characterization of the 2018 Samples

Assays were performed at METTS on samples of the 2018 composite mixtures. The results for gold and silver are shown in Table 13.16.

Table 13.16 Gold and Silver Grades of the 2018 Composites

Composite	HG:LG Ratio	Units	Au	Ag
M80/20	Mixture 80/20	g/t	5.43	34
M70/30	Mixture 70/30	g/t	3.88	72
M60/40	Mixture 60/40	g/t	3.81	24

Natural pH, the specific gravity and humidity were measured in samples of each 2018 composite as shown in Table 13.17.

Table 13.17 2018 Composite pH, Specific Gravity and Humidity

	Units	M80/20	M70/30	M60/40
Natural pH		6.8	6.6	6.7
Specific gravity*	g/cm^3	2.73	2.78	2.86
Moisture	%	0.83	1.21	0.76

** Measured by the pycnometer method*

13.3.3 2018 Column Tests

Three cyanidation tests were performed in PVC columns whose dimensions were 190.5 mm in diameter by 2.5 meters in height. The behavior of the mineralized material with cyanidation in the columns was evaluated using the three mixtures but maintaining the granulometry at 100% -3 inch, concentration of NaCN at 1,000 ppm, and irrigation rates of 6.0 L/h/m^2, 8.0 L/h/m^2 and 10 L/h/m^2 as shown in Table 13.18.

Table 13.18 2018 Column Test Parameters

Columns	HG/LG	Dry Weight (kg)	Granulometry (inches)	Irrigation Rate (L/h/m^2)	NaCN Solution (ppm)
Column 1	80/20	105.81	100% - 3”	6 – 8 – 10	1,000
Column 2	70/30	96.72	100% - 3”	6 – 8 – 10	1,000
Column 3	60/40	98.64	100% - 3”	6 – 8 – 10	1,000

The leaching solutions were transported by means of dosing pumps at rates of 6.0 L/h/m^2 during days 1 through 5, then at 8.0 L/h/m^2 during day 6 through 15, and finally at 10 L/h/m^2 from day 16 until the end of the test. The percolated PLS solutions were monitored by calculating the volume, cyanide concentration, pH, gold and silver grades.

The PLS solutions and leached column residues were sent to the MinLab chemical laboratory where they were assayed. Results are summarized in Table 13.19 and Table 13.20 and the plots in Figure 13.10 and Figure 13.11.

Table 13.19 2018 Column Test Gold Results

Column	Au Head Grade (g/t)		PLS Au (g/t)	Solid residue Au (g/t)	Recovery Au (%)	Reagents Consumed (kg/t)		Leach Rate (m^3/t)
	Assayed	Calculated				Cyanide	Lime	
Column 1	5.43	3.99	3.21	0.78	80.44	0.545	4.45	2.00
Column 2	3.88	3.93	3.23	0.70	82.25	0.633	4.86	2.20
Column 3	3.81	3.55	2.88	0.68	80.99	0.705	4.77	2.22

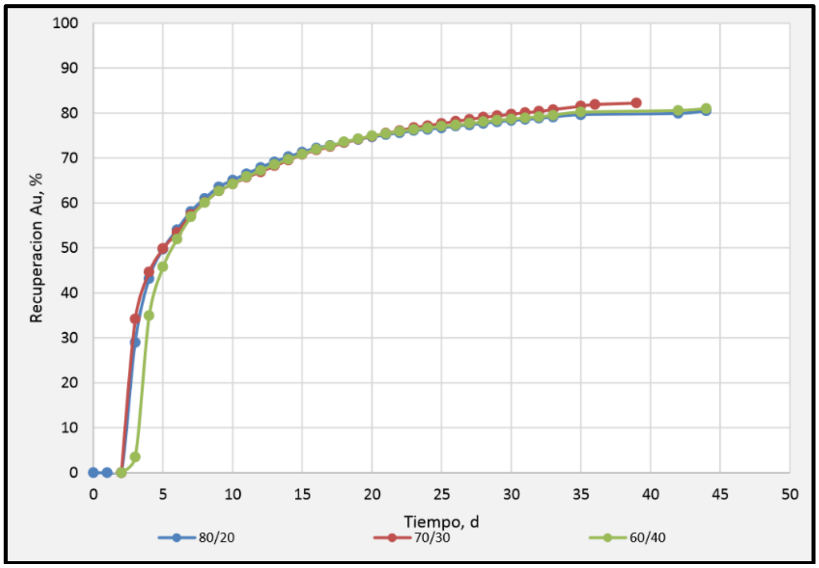
Table 13.20 2018 Column Test Silver Results

Column	Ag Head Grade (g/t)		PLS Ag (g/t)	Solid residue Ag (g/t)	Recovery Ag (%)
	Assayed	Calculated			
Column 1	34.00	55.38	11.38	44.00	20.55
Column 2	72.00	33.21	9.21	24.00	28.44
Column 3	24.00	26.38	7.38	19.00	29.02

The results indicate that there are similar kinetics of gold extraction for the different mixtures. We had a slightly higher response in the 70/30 mixture (column 2) with extraction of gold and silver of 82.3% and 28.4% during a period of 35 days of irrigation. These values were achieved using granulometry smaller than 3 inch without the use of agglomeration.

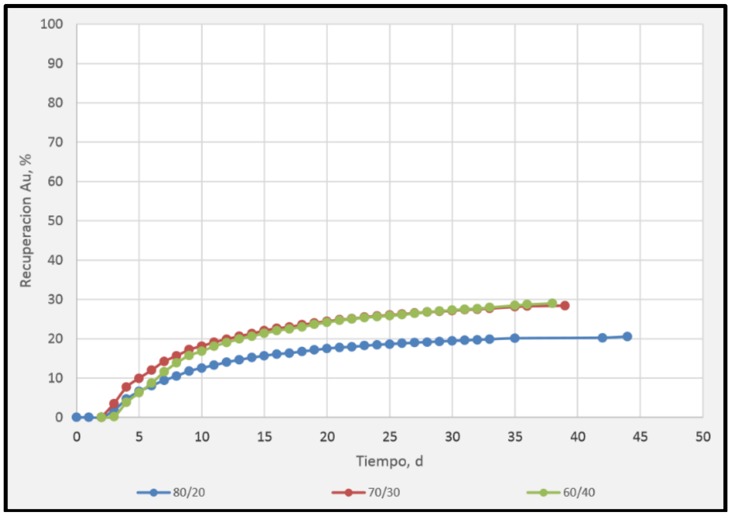
Cyanide consumption and lime consumption increased within the lower grade material (columns 2 and 3) with final values of 545g/t, 633g/t, and 705g/t for cyanide consumption, and 4. 5kg/t, 4.97kg/t and 4.8kg/t for lime consumption. The increased cyanide consumption is possibly related to the presence of cyanicide elements (Cu, As) while the higher consumption of lime could indicate that there were sulfates in the composites that produced hydroxyl ions, which reduced the pH of the column material.

Figure 13.10 2018 Column-Leach Gold Extraction Kinetics



The results indicate that there is a rapid extraction of close to 70% of the gold before the 15 days, and then a slow increase over time to about 80% at day 35. The fastest gold extractions were obtained in column 2 where the ore was 100% at -3 inch, without agglomeration, and a HG:LG mixture of 70/30.

Figure 13.11 2018 Column-Leach Silver Extraction Kinetics



The results show better kinetics of silver extraction in mixtures 70/30 and 60/40 compared to the 80/20 mixture. The achieved silver extractions were on the order of 20% to 30% at close to 40 days and granulometries smaller than 3.0 inch.

13.3.4 Granulometric Analysis of the Head and Leached Spent Material

Granulometric analysis was performed at representative size fractions on the head and tail samples of the 70/30 column 2. All fractions were retained and assayed for gold and silver. The results of this analysis are shown in Figure 13.12, Figure 13.13 and Figure 13.14.

Figure 13.12 Granulometric Analysis of M70/30

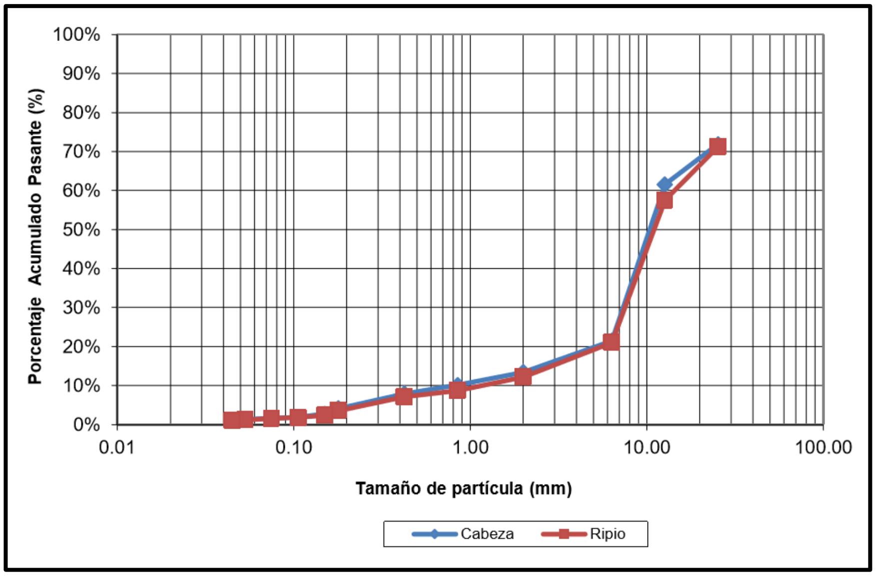


Figure 13.13 Gold Distribution by Particle Size M70/30

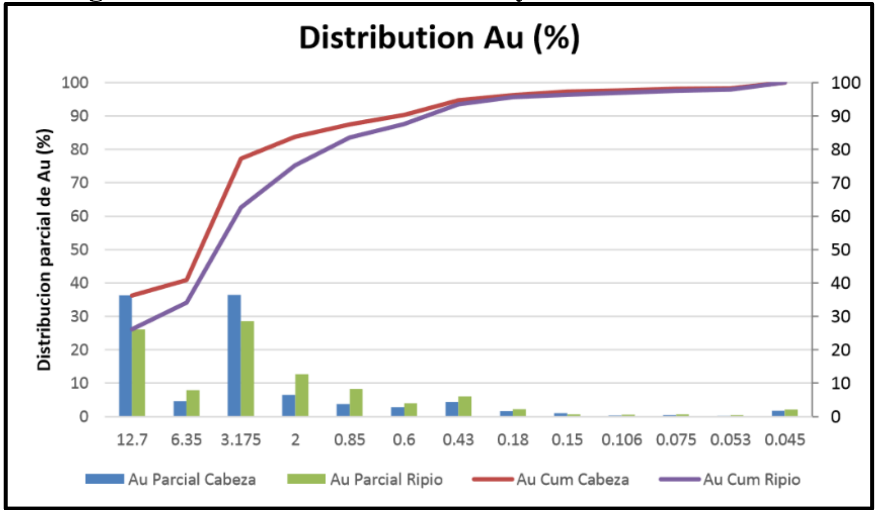
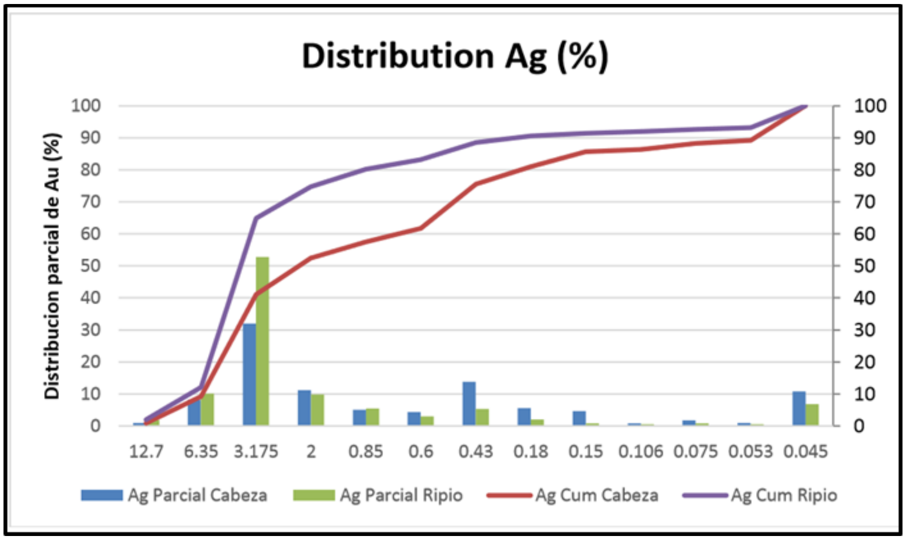
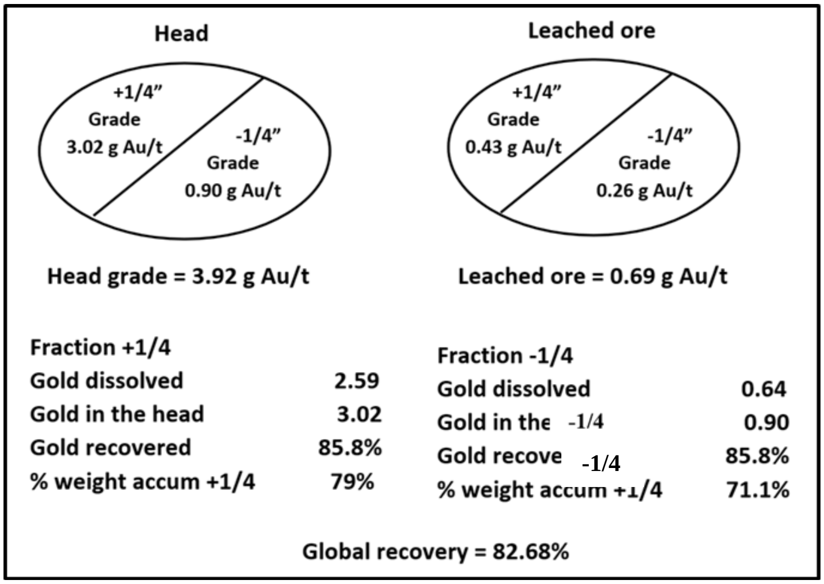


Figure 13.14 Silver Distribution by Particle Sizes M70/30



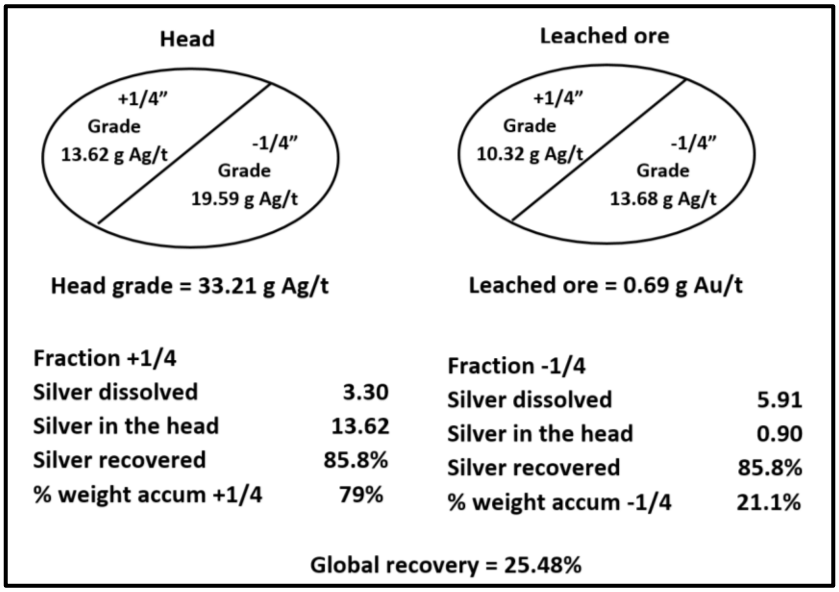
From the granulometric analysis of the composite M70/30, it can be confirmed that 77% of the gold in the head has a granulometry of plus ¼ inch (6.35 mm), which was associated with coarse-grained fragments of up to 1.0 inch. Quantifying the gold grade in this granulometry should be 3.02g Au/t (3.92 x 0.77). The gold contained in the tail (0.70g Au/t) was 62.6% distributed in particle sizes +1/4 inch that would have a gold grade of 0.43g Au/t (0.69 x 0.63). Thus, it can be confirmed there was a recovery of 85.5% of gold in the coarse-grained ranges (fractions greater than ¼ inch) while there was a recovery of 72.2% in the fine ranges (less than ¼ inch). The average gold recovery for the entire range of particle sizes is calculated to be 82.7% as represented in Figure 13.15.

Figure 13.15 Gold Distribution in + ¼ inch and - ¼ inch Particles



It can be stated that 41.1% of the silver in the head has a granulometry of plus ¼ inch (6.35 mm), which was associated with coarse-grained fragments of up to 1 inch. Quantifying the silver grade in that granulometry, there should be 13.6g Ag/t (33.21 x 0.41). The silver contained in the tail (24g Ag/t) was 64. 9% distributed in + ¼ inch particles that quantified would have a silver grade of 10.32 g Ag/t. (24 x 0.6489). Thus, it can be confirmed that there was a silver recovery of 19.9% in the coarse-grained fractions (greater than ¼ inch) while there was a silver recovery of 30.2% from the fine-grained fractions (less than ¼ inch). The average silver recovery was calculated at 22% for both size fractions as represented in Figure 13.16.

Figure 13.16 Silver Distribution in + ¼ inch and - ¼ inch Particles



13.3.5 Discussion of 2018 Blended Column-Leach Tests

The 2018 column tests reported by METTS (2018b) had the objective to evaluate the possibility of avoiding the process of agglomeration. The results were encouraging since it was possible to dissolve gold between 80% to 82%, and silver at 20% to 28%. Based on these results, it will be possible to have the option to load ore on the heap using material of low grade with coarse particle size. The results indicate that it is feasible to reach gold extractions of 82.3% for the 70/30 type of HG:LG mixture with a high proportion of coarse grains in the low-grade material for adequate percolation.

The 2018 column-leach tests indicated consumption of lime between 4.4kg/t to 4.8kg/t while cyanide consumptions were 0.5kg/t to 0.7 kg/t. The % of slump (settlement) was in the range of 0.12% to 0.46%. Therefore, the irrigation rate should be between 8.0 and 10 L/h/m^2.

The granulometric evaluation that gave the best result in terms of gold extractions was for the 70/30 mixture. Gold recovery in the coarse range (+1/4") was 85.8% while gold recovery in the fine range (-1/4") was 71.1% This implies that the fragments of material in the coarse size ranges are altered and easy to penetrate with the leach solutions. To obtain better recoveries in coarse range is required to have a longer period of irrigation. The recovery of silver is almost similar in the fine range as in the coarse range with similar values of 24.2% to 30.2% reported. It can be said that the recovery of silver is uniform throughout the range of sizes.

Based on the 2018 column tests, the mixture of low-grade mineralization with greater than 80% at +¼ inch is a good option for adequate percolation. It can be seen that the coarse range contributes more gold recovery than the fine range and it was demonstrated that the coarse material could reach important gold recovery values if a longer period of leaching was applied. The same could not be said for the fine range with gold extractions of 71.1%, confirming that the greater distribution are in ultrafine range that do not allow the passage of leaching solutions.

13.4 Toll Processing of Bulk Samples 2016 - 2018

Useful metallurgical data was produced by the off-site Malin toll mill from processing 31,437 tonnes of PPX's bulk sampling and test mining material. A total of 7,028oz of gold was produced with an overall recovery of 81.27 %. The Malin plant is located 60 km from the Callanquitas mine. No grinding data is available, but Malin used a standard ball mill stage after a jaw crusher. No report is available as the focus was gold production, not metallurgical testing. The summary result is the following:

Table 13.21 Malin CIL Plant Test Gold Production

Dates of Production	Tonnes Processed	Grade Au g/t	Recovery %	Au Oz Produced
Jan-Sep 2018	21,404	8.64	81.4	4,837
Oct-Nov-Dec Q4 2017	5,230	9.28	82.3	1,287
Jul-Aug-Sep Q3 2017	2,792	7.76	79.0	551
Subtotal	29,425	8.67	81.4	6,672
Oct 16 – Feb 17	2,011	6.99	78.7	356
Grand Total	31,437	8.56	81.2	7,028

13.5 Conclusions

The sampling of mineralized material in the Callanquitas mine and the metallurgical testwork carried out by METTS, as well as other laboratories, were not supervised or observed by Dr. Uceda who is responsible for this section of the report. As such, the results were not independently verified, but Dr. Uceda has reviewed the reports and data of METTS and PPX, and believes the information is sufficiently reliable to be used in this report and support the conclusions summarized below.

The 2017 results demonstrated cyanide gold extractions of 80% for the M2 composite at granulometries lower than ½ inch using agglomeration using a NaCN concentration of 2,000 ppm and a leaching rate of 6.0 L/h/m^2. For composites M1 and M2, gold was distributed in the larger grain size ranges with about 60% of the gold in the + ¼ inch size fraction. Tests on these composites showed that the generation of a large percentage of fines may require agglomeration with cement and curing of the agglomerated material to improve the kinetics of the cyanidation leaching. The best results were obtained for granulometries of 100% at - ½ inch (-12.5 mm). This suggests that reducing the granulometry to 100% at - 3/8 inch (-9 mm) may obtain better recoveries of gold. For Section 17, the process design considers recoveries of gold and silver of 80.4%, and 33.0% respectively, as was reported for composite M2. The NaCN consumption of 0.18 kg/t used in Section 17 should be verified.

Based on the 2018 metallurgical test work performed on Callanquitas gold and silver mineralized samples, the following conclusions can be made:

- Crushing to granulometry of less than 100% at - 3 inch (-75mm) for HG:LG material mixed in 70/30 is recommended using primary crushing only for the heap-leach operation, while the HG material is reduced in primary, secondary and tertiary crushing processes;
- Use an irrigation rate of 8.0 to 10 L/h/m^2;
- The heap-leaching solution should have NaCN concentrations that fluctuate between 500 to 1,000 ppm, or less if the operation so indicates;
- Cyanide consumption will fluctuate between 0.5 to 0.7 kg/t;
- Lime consumption should be between 4.4 to 4.8 kg/t;
- Better gold extractions were achieved in the coarser particle size ranges than in the finer ranges, which indicates longer leach times improve extractions; and therefore
- The heap should be irrigated longer than the estimated 90 to 100 day estimated leach cycle;
- Due to the percentage of fines, agglomeration with cement will be required;
- It is important to perform Merrill-Crowe testing for the recovery of gold and silver as well as testwork program for cyanide destruction; and
- Some additional column-leach tests should be completed to confirm the recovery of gold and silver.

The only deleterious elements affecting gold and silver recovery are relatively high concentrations of arsenic and lead. However, they do not cause a major increase in the consumption of cyanide.

14.0 MINERAL RESOURCE ESTIMATES (ITEM 14)

14.1 Introduction

Mineral resource estimation described in this section follows CIM standards and the disclosure and reporting requirements set forth in NI 43-101. The modeling and estimation of the Igor project mineral resources were done under the supervision of Mr. Paul G. Tietz. There is no affiliation between Mr. Tietz and PPX or their subsidiaries, except that of an independent consultant/client relationship. Although Mr. Tietz is not an expert with respect to any of the following aspects, Mr. Tietz is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Igor mineral resources as of the date of this report. The effective date of the mineral resource estimate is July 27, 2018.

The Igor project mineral resources discussed in this section are entirely within the Callanquitas deposit which is the current focus of mine development. The Igor project contains a number of other exploration and development target areas (Domo, Tesoros, Portachuelos), with limited drilling, that are viable drill targets with the potential to add to the project-wide resources.

Mr. Tietz classifies resources in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories to be in accordance with the CIM Standards. CIM mineral resource definitions are given below, with CIM's explanatory material shown in italics:

> **Mineral Resource**
>
> *Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.*
>
> A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.
>
> The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
>
> *Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.*
>
> *The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in*

respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

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

Inferred Mineral Resource

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

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

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

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

Indicated Mineral Resource

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

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

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

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

Measured Mineral Resource

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

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

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

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

Modifying Factors

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

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

14.2 Resource Data

The Igor project drilling database contains 136 holes for a total of 30,752.5 meters drilled. Of this total, 93 drill holes for 23,264.3 meters are within the Callanquitas deposit and were used in creating the resource block model and estimate.

The Igor drilling database contains 18,740 sample intervals with gold and silver assays. All less-than-detection values were converted to "0" for use in the resource estimate.

Additional sample data used in the resource estimate are 1,126 underground channel samples collected in 2016 through 2018 from within the Callanquitas mine development areas. The channel samples are treated as point data within the database though the sample lengths, which range from 0.2 to 3 meters with a median length of 0.9 meters, are used to determine mineral widths within the mineral domain model. Very often a single ~1-meter channel sample covers the full width of the mineralized structures. Where the vein/breccia structures widen, two to three contiguous channel samples characterize the mineral zone. The center sample usually covers the high-grade vein/hydrothermal breccia and the two surrounding samples comprise lower-grade wallrock or weakly mineralized tectonic breccia.

The project drill-hole database contains down-hole survey information for 85 of the 93 Callanquitas drill holes. For most holes outside the resource area, the down-hole orientation is based on the collar set-up. Geologic data includes logged lithology, oxidation class (oxide, mixed, and sulfide), and structural features (type and orientations relative to core axis).

Project digital topography was provided by PPX. The current project coordinate system is WGS84 Zone 17. The pre-2017 collar surveys were originally surveyed in PSAD56. All modeling of the Callanquitas resource was performed using GEOVIA Surpac^{TM} mining software.

14.3 Deposit Geology Pertinent to Resource Modeling

The Callanquitas mineralization is considered an intermediate-sulfidation type, consisting of mineralized structural breccias that originally contained variable quantities of pyrite, arsenopyrite, and minor chalcopyrite. These sulfide minerals were subsequently strongly oxidized to significant depths. The mineralized structures, primarily the Callanquitas Este and Oeste structures, are nearly vertical and cut through a sequence of sandstone, siltstone and quartzite. The silicified sandstone and quartzite units were more amenable to brecciation and therefore more favorable for migrating hydrothermal fluids and mineralization.

Detailed underground mapping by PPX, along with a re-logging of early core holes, indicates multiple stages of veining, brecciation, alteration, and mineralization. The initial event appears to be emplacement of quartz veins within faults, followed by multiple stages of hydrothermal brecciation ("HBx") with associated pervasive silicification, which was then cut by sulfide veinlets. The last stage is a fault gouge or tectonic breccia ("TBx") usually occurring adjacent to or cross-cutting the HBx. The highest gold grades are associated with the early quartz and HBx while increased silver occurs with the late-stage sulfide veinlets. The TBx is argillized and appears to be mostly post-mineral in the paragenesis, though it often contains low-grade gold in the range of 0.5 to 1.0g Au/t. Gold within the TBx is believed to be due to the presence of mineralized clasts within the argillized fault gouge, though there are exposures underground of TBx with oxidized veinlets that can assay up to multiple grams of gold. It is possible that there are multiple TBx events with early-stage faulting prior to the formation of silica-sulfide veinlets.

Drilling indicates that mineralization in the Callanquitas Este and Oeste veins occurs over a 1,400 meters north-south strike length and extends to depths of up to 500 meters below the topographic surface. Both veins are near-vertical and widths are variable from 0.5 meter to over 5 meters. Oxidation within the mineralized structures occurs to depths of 200 to 300 meters below surface. Ag/Au ratios are generally low (about 10:1) in the oxide portion of the deposit, but can increase to over 100:1 at the oxide-sulfide interface. The change in Ag/Au ratios is not just related to supergene enrichment, but also represents the occurrence of late-stage, silver-rich, silica-sulfide veinlets which cut the more gold-rich hydrothermal breccias.

14.4 Geology Modeling

PPX provided Mr. Tietz wireframe models of the Callanquitas Este and Oeste vein/breccia structures, the dacite dike, and the dominant cross-cutting fault surfaces. The Callanquitas wireframe models included separate wireframe solids of predominantly HBx and predominantly TBx volumes within the structural zones. These were rectified to drill and underground sample data and were the primary controlling features for the mineral domain model.

The dacite dike is included in the geology model due to its unique density values, as compared to the sedimentary country rock, and as a control on mineralization. Mineralization appears to favor the dike contacts, whereas breccia development and associated mineralization is not as strong within the dacite.

Faulting appears to be both pre- and post-mineral. The fault model consists of fault surfaces numbered 1 through 9 that dip predominantly to the west and cross-cut the Callanquitas Este and Callanquitas Oeste veins. Mineralization appears to be localized at structural intersections and extend out along these structures into the wallrock of the Callanquitas Este and Callanquitas Oeste veins.

14.5 Oxidation

The project database includes oxidation codes (oxide, mixed, and sulfide) for the Callanquitas drill holes. Using these data and the core photos, an oxide/sulfide surface was created for use in coding the block model for metallurgical purposes and for assigning density. The mixed material generally occurs at the oxide/sulfide interface and in most areas was considered “sulfide” for coding purposes.

14.6 Mineral Domain Modeling

The mineral resources at Igor were modeled and estimated by evaluating the drill data statistically, utilizing the geologic interpretations provided by PPX to interpret gold mineral domains on east-west-oriented vertical cross-sections, rectifying the gold mineral domains on 2.0 meter-spaced level plans which were then used to create wireframe solids, and analyzing the modeled mineralization geostatistically to establish estimation parameters. This was followed by estimating gold and silver grades into a three-dimensional block model. All modeling of the Igor resources was performed using Geovia® mining software.

A separate silver model was not constructed due to 1) the generally close association with gold, and 2) the relatively low silver values and metallurgical recoveries within the oxide portions of the deposit which results in silver having a minor impact on current projected economics. The silver assay coding and grade estimate were controlled using the same mineral domains and estimation parameters as the gold model.

A total of 27 vertical, north-looking cross sections spaced at irregular 25 to 75-meter intervals across the deposit were used for the initial modeling of the Callanquitas gold mineral domains. A mineral domain is a natural population of metal grades that occurs within a specific geologic setting. In order to define the gold mineral domains, the natural populations were first identified on quantile graphs that plot the gold-grade distributions of the drill-hole assays. This analysis led to the identification of low- (~0.3 to ~3.0g Au/t), and high-grade (>~3.0g Au/t) gold populations, assigned to domains 100 and 200, respectively. Ideally, each of these populations can be correlated with specific geologic characteristics that are captured in the project database and/or geologic model to aid in the definition of the mineral domains. At Callanquitas, the high-grade gold mineralization is closely associated with the hydrothermal breccia “HBx” units while the low-grade gold mineralization occurs within the weakly altered sedimentary wallrock or the tectonic breccias “TBx” that cut or are immediately adjacent to the HBx units.

The drill-hole traces, topographic profile, and PPX’s geologic interpretations were plotted with gold assays, colored by the grade-domain population ranges, along the drill-hole and channel sample traces, and these data were used to guide the mineral domain interpretations. Mineral-domain envelopes were interpreted on the sections to more-or-less capture assays corresponding approximately to each of the defined grade populations. The domains were modeled through all available drill and underground data, including volumes that had been mined. Representative cross sections showing the gold mineral-domains in the Callanquitas Este and Callanquitas Oeste veins are presented in Figure 14.1 and Figure 14.2, respectively. The cross-section locations are shown in Figure 10.1.

The cross-sectional mineral-domain polygons were digitized and then three-dimensionally rectified to the drill data. Level plan slices of the polygons were then created at 2-meter intervals and level plan mineral domain polygons were interpreted. These level plan polygons were then used as the wireframe skeleton for the creation of mineral domain solids. The solids were then used to code the block model on a volume percentage basis.

Figure 14.1 Cross Section 9153910 Showing Gold Mineral Domains

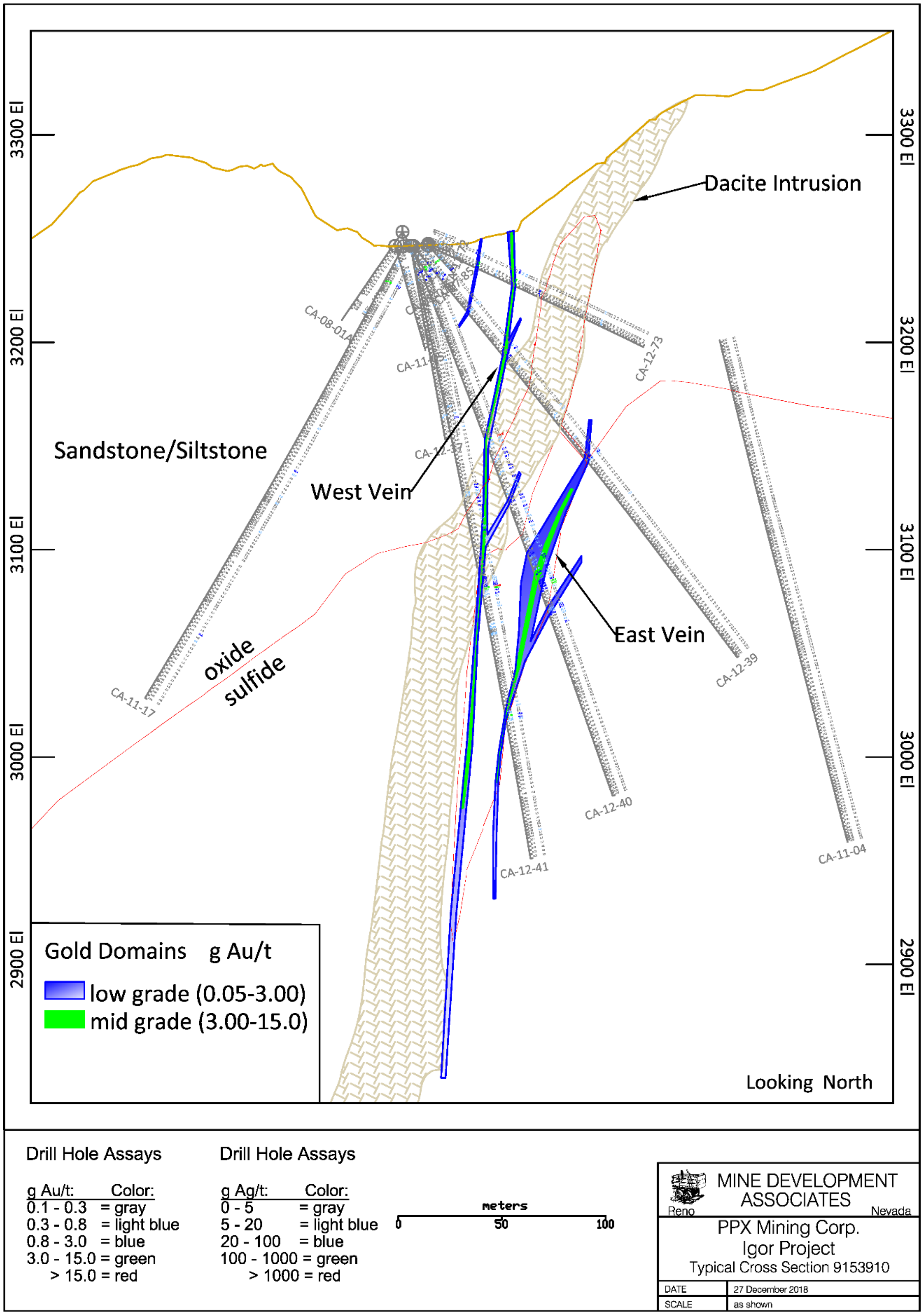
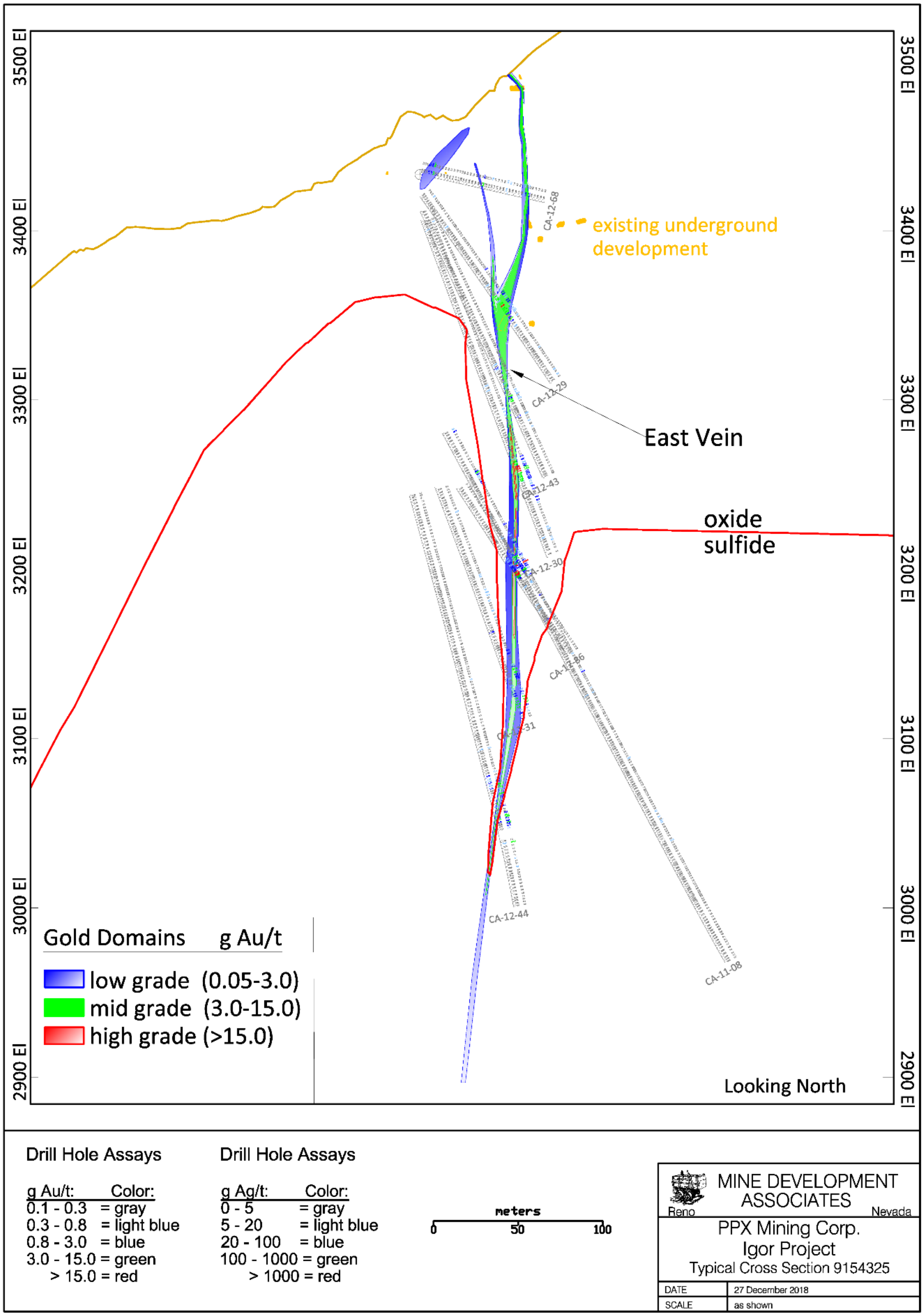


Figure 14.2 Cross Section 9154325 Showing Gold Mineral Domains



14.7 Density

PPX collected 828 samples for density determinations from their 2012, 2017 and 2018 core drilling programs. The samples were from all significant rock types and gold grade ranges. PPX conducted the density testing using both the water-immersion and volumetric methods on each sample. There were no significant differences in results from the different procedures, but the relatively low precision of the weighing scale used in the water-immersion method resulted in some uncertainties in the water-immersion results. Accordingly, the volumetric results were the primary source for the density data used in the resource block model.

Mr. Tietz evaluated the statistics of the density results and assigned density values to the block model based on general rock type, gold mineral domain, and oxidation. Due to the very similar densities for the various sedimentary wallrock units within Callanquitas deposit, the sandstone, quartzite, silty sandstone, and siltstone were assigned as one rock type in the block model and coded with a single density value. The dacite porphyry was assigned a separate density value in the block model. Densities within the mineralized breccias were found to be lower than those in the surrounding wallrock. There was no material difference in density between the low-grade (domain 100) and high-grade (domain 200) gold mineralization. Due to the fractured nature of the deposit, and the fact that voids resulting from many of the open fractures cannot be accurately reflected in density determinations, the measured density values were reduced by 1.0 percent to account for the unavoidable sample-selection bias. The factored density values shown in the "Model SG" columns in Table 14.1 reflect the density values assigned to the Callanquitas block model.

Table 14.1 Callanquitas Deposit Density Values

Model Rock Type	**Model SG (g/cm^3)**	
	Oxide	**Sulfide**
sedimentary wallrock	2.55	2.6
dacite porphyry	2.35	2.45
mineralized structures	2.52	2.62

14.8 Assay Coding, Capping, and Compositing

14.8.1 Drill Hole Sample Data

Drill sample intervals are based on modeled geology with sample breaks at geologic contacts. The average widths for samples used in the estimate is 1.3 meters with a similar value for the median sample width.

Drill-hole gold and silver assays were coded to the gold mineral domains using the cross-section mineral-domain envelopes. Descriptive statistics of the coded gold and silver assays are provided in Table 14.2 and Table 14.3.

Table 14.2 Descriptive Statistics of Callanquitas Deposit Coded Gold Assays – Drill Data

Domain	Assays	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
100	Au	1498	0.653	0.387	0.866	1.320	0.008	15.900
	Au Cap	1498	0.646	0.387	0.792	1.230	0.008	6.000
200	Au	333	5.790	4.351	6.001	1.040	0.143	63.130
	Au Cap	333	5.707	4.351	5.373	0.940	0.143	40.000
All	*Au*	*1831*	*1.493*	*0.505*	*3.181*	*2.130*	*0.008*	*63.130*
	Au Cap	*1831*	*1.473*	*0.505*	*2.957*	*2.010*	*0.008*	*40.000*

Table 14.3 Descriptive Statistics of Callanquitas Deposit Coded Silver Assays – Drill Data

Domain	Assays	Count	Mean (g Ag/t)	Median (g Ag/t)	Std. Dev.	CV	Min. (g Ag/t)	Max. (g Ag/t)
100	Ag	1498	33.100	4.700	148.800	4.490	0.000	5026.000
	Ag Cap	1498	27.700	4.700	77.700	2.810	0.000	600.000
200	Ag	333	297.300	37.400	786.600	2.650	0.600	6475.000
	Ag Cap	333	184.300	37.400	319.800	1.740	0.600	1100.000
All	*Ag*	*1831*	*76.300*	*6.800*	*359.400*	*4.710*	*0.000*	*6475.000*
	Ag Cap	*1831*	*53.300*	*6.800*	*158.500*	*2.970*	*0.000*	*1100.000*

The process of determining assay caps began with inspection of quantile plots of the coded assays by domain to assess the mineral-domain populations and identify possible high-grade outliers that might be appropriate for capping. Descriptive statistics of the coded assays by domain, and visual reviews of the spatial relationships of the possible outliers and their potential impacts during grade interpolation, were also considered in the process of determining appropriate assay caps. After this review, 11 gold samples, seven in the 100 domain and four in the 200 domain, were capped at 6.0g Au/t and 40g Au/t, respectively. A total of 36 silver assays, 13 in the 100 domain and 23 in the 200 domain, were capped at 600g Ag/t and 1,100g Ag/t, respectively. The effects of the final assay caps can be qualitatively evaluated by examination of the descriptive statistics of the capped and uncapped mineral-domain assays (Table 14.2 and Table 14.3).

The capped assays were composited at 2.0 meter down-hole intervals respecting the gold mineral domains, and length-weighted composites were used in the block-model grade estimation. The grade inside each mineral domain was estimated using only composites from inside that domain. Descriptive statistics of the gold and silver composites are shown in Table 14.4 and Table 14.5.

Table 14.4 Descriptive Statistics of Callanquitas Deposit Gold Composites– Drill Data

Domain	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
100	1094	0.646	0.437	0.663	1.030	0.009	6.000
200	241	5.707	4.643	4.621	0.810	0.733	40.000
All	*1335*	*1.473*	*0.563*	*2.713*	*1.840*	*0.009*	*40.000*

Table 14.5 Descriptive Statistics of Callanquitas Deposit Silver Composites – Drill Data

Domain	Count	Mean (g Ag/t)	Median (g Ag/t)	Std. Dev.	CV	Min. (g Ag/t)	Max. (g Ag/t)
100	1094	27.700	6.000	70.400	2.540	0.000	600.000
200	241	184.300	43.200	299.200	1.620	1.400	1100.000
All	*1335*	*53.300*	*8.200*	*148.800*	*2.790*	*0.000*	*1100.000*

14.8.2 Underground Sample Data

The underground channel sample gold and silver assays were coded to the gold mineral domains using the 2.0 meter-spaced level plan polygons. The closely-spaced level plan polygons provided much greater spatial precision than the 25- to 50-meter spaced cross-sections. Each underground sample was treated as a unique drill hole so each underground sample serves as both the initial assay and then also the composite for use in grade estimation. Descriptive statistics of the coded gold and silver channel sample assays/composites are provided in Table 14.6 and Table 14.7.

Table 14.6 Descriptive Statistics of Coded Underground Channel Gold Assays/Composites

Domain	Assays	Count	Mean (g Au/t)	Median (g Au/t)	Std. Dev.	CV	Min. (g Au/t)	Max. (g Au/t)
100	Au	245	1.558	1.039	1.954	1.250	0.080	18.300
	Au Cap	245	1.418	1.039	1.233	0.870	0.080	6.000
200	Au	856	11.472	8.120	12.756	1.110	0.220	200.980
	Au Cap	856	10.930	8.120	9.032	0.830	0.220	40.000
All	*Au*	*1101*	*9.404*	*5.870*	*12.074*	*1.280*	*0.080*	*200.980*
	Au Cap	*1101*	*8.945*	*5.870*	*8.934*	*1.000*	*0.080*	*40.000*

Table 14.7 Descriptive Statistics of Coded Underground Channel Silver Assays

Domain	Assays	Count	Mean (g Ag/t)	Median (g Ag/t)	Std. Dev.	CV	Min. (g Ag/t)	Max. (g Ag/t)
100	Ag	245	30.050	8.000	108.070	3.600	0.000	1441.160
	Ag Cap	245	25.830	8.000	57.450	2.220	0.000	600.000
200	Ag	856	96.490	37.180	224.960	2.330	0.000	2905.000
	Ag Cap	856	89.080	37.180	164.970	1.850	0.000	1100.000
All	*Ag*	*1101*	*82.630*	*30.240*	*207.880*	*2.520*	*0.000*	*2905.000*
	Ag Cap	*1101*	*75.880*	*30.240*	*151.280*	*1.990*	*0.000*	*1100.000*

The capping process and capping limits used for the underground channel sample assays were similar to those used for the drill data. A total of 27 underground gold samples, eight in the 100 domain and 19 in the 200 domain, were capped at 6.0g Au/t and 40g Au/t, respectively, while 11 silver assays, one in the 100 domain and 10 in the 200 domain, were capped at 600g Ag/t and 1,100g Ag/t, respectively. The smaller number of capped silver values within the underground data set, as compared to the number of capped samples within the drill data, reflects the generally low silver values within the upper portions of the deposit in the vicinity of the current underground development. The high silver values are predominantly at depth, near or just below the oxide/sulfide contact.

14.9 Block Model Coding

The mineral domain wireframe solids were used to code a north-south three-dimensional block model that is comprised of 0.5 meter (width) x 2 meter (length) x 2.5 meter (height) blocks. The block size was chosen to best model the narrow, mineralized structures and in consideration of current mining techniques. In order for the block model to better reflect the irregularly shaped limits of the various gold domains, as well as to explicitly model dilution, the percentage volume of each mineral domain within each block was stored (the “partial percentages”).

Each block was assigned a density listed on Table 14.1 based on its coded lithology, oxidation state, and mineral domain. The percentage of each block that lies below the topographic surface was also stored.

The underground as-built solids, dated to October 17, 2018, were used to code the model and any block that has any percentage of as-built solid is considered “mined out” and was removed from the current resource tabulation.

14.10 Resource Estimation

The resource estimate reflects the general northerly trend and vertical nature of the Callanquitas Este vein and the generally steeply west-dipping Callanquitas Oeste vein. To replicate the change in orientation observed within the Este vein, and also between the two veins, three search-ellipse orientations were used to control the resource estimate (see Table 14.8). The first two orientations reflect the subtle change in the Este vein azimuth with the majority of the vein below the underground workings (Area 10 for block coding) trending at azimuth 5 degrees while the upper portions of the vein developed within the current mine workings (Area 11 for block coding) is generally trending due north. The search ellipse for the Oeste vein is oriented at azimuth 350 degrees orientation with a steep westerly dip.

Table 14.8 Callanquitas Search Ellipse Orientations

Estimation Area	Major Bearing	Plunge	Tilt
Area 10; East Vein below current mine development	5°	0°	90°
Area 11; East Vein at current mine development	0°	0°	90°
Area 20; West Vein	350°	0°	85°

A variographic study was performed using the drill hole and underground gold composites from each mineral domain, collectively and separately, at various azimuths, dips, and lags. Acceptable variogram models were obtained from composites from the combined 100 and 200 gold domains for both the drill and underground data. A maximum range of about 300 meters was obtained along the strike of the veins (azimuth 0°, plunge 0° to 30°), though grade relationships decreased significantly past about 75 meters. Parameters obtained from the variography study were used in an ordinary-krige interpolation and also provided information relevant to both the estimation parameters used in an inverse-distance interpolation and in resource classification.

The estimation parameters applied at Callanquitas are summarized in Table 14.9. The estimation used three search passes within each estimation area with successive passes not overwriting previous estimation passes. The first-pass search distances take into consideration the results of both the variography and drill-hole spacing. The second and third passes were designed to estimate grade into all blocks coded to the mineral domains that were not estimated in the first pass.

Table 14.9 Summary of Callanquitas Estimation Parameters

Estimation Pass	Search Ranges (ft)			Comp Constraints		
	Major	S-Major	Minor	Min	Max	Max/hole
1 (area 10,20)	100	100	33	2	12	4
2 (area 10,20)	200	200	100	1	16	4
2 (area 10,20)	300	300	300	1	16	4
1 (area 11)	50	50	17	2	12	4
2 (area 11)	100	100	50	1	16	4
3 (area 11)	200	200	100	1	16	4

The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded by that domain. The estimated grades were coupled with the partial percentages of the mineral domains to enable the calculation of a single, weight-averaged, block-diluted grade for each block.

Gold and silver grades were interpolated using inverse distance to the third power ("ID3"), ordinary-krige, and nearest-neighbor methods. Except for those blocks coded as "mined out", gold and silver grades were estimated into all blocks coded by the gold mineral domains. The mineral resources reported herein were estimated by ID3 interpolation, as this technique was judged to provide results superior to those obtained by ordinary kriging. The nearest-neighbor estimation was also completed as a check on the other interpolations.

14.11 Callanquitas Mineral Resources

The Callanquitas resources were classified on the basis of the distance of the model blocks to the nearest composite, and the minimum number of composites and drill holes used in the grade interpolation of each block (Table 14.10). Measured resources are restricted to blocks defined by PPX's underground development while all sulfide resources are considered Inferred due to limited metallurgical studies.

Table 14.10 Callanquitas Classification Parameters

Este Vein Development (Area 11)	
Measured Resource	
Minimum no. of samples /minimum no. of holes / maximum distance (m)	3 / 2 / 15
Indicated	
Minimum no. of samples /minimum no. of holes / maximum distance (m)	3 / 2 / 50
Lower Este Vein and Oeste Vein (Areas 10, 20)	
No Measured Resource	
Indicated	
Minimum no. of samples /minimum no. of holes / maximum distance (m)	3 / 2 / 50
All material not classified above but lying within the modeled mineralized domains is Inferred	
Sulfide Resource – All Inferred	

Because of the requirement that the resource exists "*in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction*", Mr. Tietz is reporting the resources at an economic cutoff grade that is reasonable for deposits of this nature that will likely be mined primarily by underground methods. As such, some economic considerations were used to determine cutoff grades at which the resource is presented. In determining the resource cutoff grade, Mr. Tietz used a $1500 per ounce gold price, $75/t combined mining and processing operating cost, and an 80 percent extraction rate.

In general, there is good continuity of blocks making cutoff grade over the full 1,000 meter extent of the main Este and Ouest veins. Both veins are near-vertical and widths are variable from 0.5 meter to over 5 meters. Blocks making the cutoff grade which are spatially isolated and would not reasonably be considered for development were removed from the resource. These non-resource blocks are either within less intensely mineralized portions along strike of the main vein trends or in smaller veins offset and sub-parallel to the two main veins.

The Callanquitas reported gold and silver mineral resources are listed in Table 14.11. The reported resources are inclusive of the reserves stated in Section 15.0 and discussed in Section 16 and 22 of this report.

Table 14.11 Callanquitas Reported Mineral Resources

Type	Class	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Oxide	Measured	109,000	5.319	19,000	61.7	215,000
	Indicated	1,361,000	4.668	204,000	113.4	4,962,000
	Meas. + Ind.	1,470,000	4.718	223,000	109.5	5,177,000
Oxide	Inferred	344,000	4.581	51,000	124.6	1,376,000
Sulfide	Inferred	269,000	2.970	26,000	159.3	1,378,000

The stated resources are fully diluted to 0.5 meter (width) x 2 meter (length) x 2.5 meter (height) blocks and are tabulated on a gold-equivalent ("AuEq") cutoff grade of 2.0g AuEq/t. All material, regardless of which metal is present and which is absent, is tabulated. Because multiple metals exist, but do not on a local scale necessarily co-exist, the AuEq grade is used for tabulation.

Using the individual metal grades of each block, the AuEq grade is calculated using the following formula:

$$\text{g AuEq/t} = \text{g Au/t} + (0.004444 * \text{g Ag/t})$$

This formula is based on prices of US$1,350.00 per ounce gold and US$18.00 per ounce silver and a 3:1 recovery difference between gold and silver. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to the Indicated classification with continued exploration and metallurgical studies.

The Callanquitas resources, are tabulated at various cutoff grades in Table 14.12 and Table 14.13 for oxide and sulfide material, respectively. These block-diluted resources are tabulated at additional cutoffs in order to provide grade-distribution information, as well as to provide for economic conditions other than those envisioned by the reported resource cutoffs.

Table 14.12 Callanquitas Oxide Resource

Cutoff	Measured Resource - Oxide				
(g AuEq/t)	tonnes	g Au/t	oz Au	g Ag/t	oz Ag
1.000	304,000	2.744	27,000	35.78	350,000
1.500	177,000	3.875	22,000	48.77	277,000
2.000	**109,000**	**5.319**	**19,000**	**61.67**	**215,000**
2.250	91,000	5.952	17,000	66.28	195,000
2.500	81,000	6.452	17,000	69.36	180,000
2.750	75,000	6.772	16,000	71.42	172,000
3.000	70,000	7.041	16,000	72.98	165,000
3.500	63,000	7.497	15,000	76.65	155,000
4.000	56,000	7.979	14,000	81.14	146,000
5.000	44,000	9.011	13,000	91.66	129,000
6.000	36,000	9.902	11,000	99.49	114,000
7.000	29,000	10.797	10,000	105.70	97,000

Cutoff	Indicated Resource - Oxide				
(g AuEq/t)	tonnes	g Au/t	oz Au	g Ag/t	oz Ag
1.000	2,847,000	2.840	260,000	74.18	6,789,000
1.500	1,738,000	3.971	222,000	100.91	5,639,000
2.000	**1,361,000**	**4.668**	**204,000**	**113.42**	**4,962,000**
2.250	1,256,000	4.904	198,000	117.52	4,746,000
2.500	1,171,000	5.108	192,000	121.56	4,576,000
2.750	1,100,000	5.286	187,000	125.34	4,434,000
3.000	1,035,000	5.456	182,000	128.98	4,294,000
3.500	912,000	5.797	170,000	136.55	4,005,000
4.000	788,000	6.171	156,000	146.16	3,704,000
5.000	572,000	6.951	128,000	170.00	3,125,000
6.000	367,000	7.999	94,000	211.51	2,496,000
7.000	237,000	9.198	70,000	251.17	1,912,000

Cutoff	Inferred Resource - Oxide				
(g AuEq/t)	tonnes	g Au/t	oz Au	g Ag/t	oz Ag
1.000	805,000	2.631	68,000	66.33	1,717,000
1.500	418,000	4.043	54,000	106.95	1,437,000
2.000	**344,000**	**4.581**	**51,000**	**124.56**	**1,376,000**
2.250	322,000	4.757	49,000	129.84	1,345,000
2.500	305,000	4.901	48,000	134.36	1,319,000
2.750	289,000	5.038	47,000	138.83	1,292,000
3.000	274,000	5.177	46,000	143.54	1,264,000
3.500	240,000	5.485	42,000	155.07	1,197,000
4.000	206,000	5.826	39,000	168.65	1,118,000
5.000	156,000	6.394	32,000	194.82	975,000
6.000	113,000	6.932	25,000	225.72	817,000
7.000	76,000	7.481	18,000	262.45	639,000

Table 14.13 Callanquitas Sulfide Resource

Cutoff (g AuEq/t)	Inferred Resource - Sulfide				
	tonnes	**g Au/t**	**oz Au**	**g Ag/t**	**oz Ag**
1.000	1,050,000	1.533	52,000	79.14	2,673,000
1.500	412,000	2.384	32,000	136.21	1,806,000
2.000	**269,000**	**2.970**	**26,000**	**159.28**	**1,378,000**
2.250	217,000	3.288	23,000	173.65	1,210,000
2.500	196,000	3.443	22,000	179.72	1,130,000
2.750	176,000	3.595	20,000	185.73	1,052,000
3.000	157,000	3.755	19,000	192.15	970,000
3.500	120,000	4.116	16,000	204.45	790,000
4.000	89,000	4.529	13,000	213.73	611,000
5.000	45,000	5.447	8,000	226.43	328,000
6.000	24,000	6.192	5,000	264.73	201,000
7.000	13,000	6.805	3,000	298.24	123,000

Typical cross sections through the Callanquitas block model showing AuEq block grades are given in Figure 14.3 and Figure 14.4. Figure 14.3 and Figure 14.4 correspond to the mineral-domain cross sections in Figure 14.1 and Figure 14.2

Figure 14.3 Cross Section 9153910 Showing Block Model Gold Grades

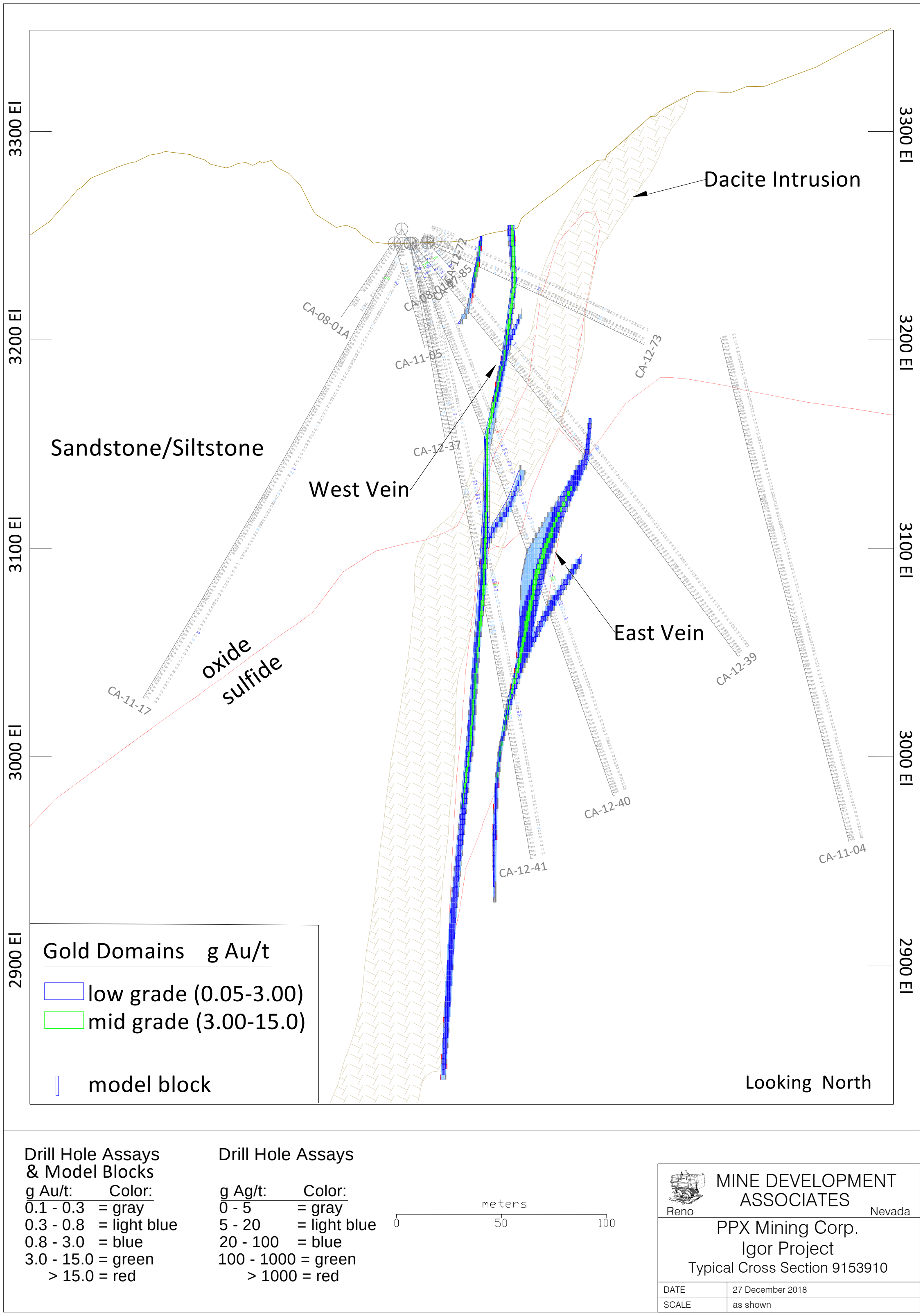
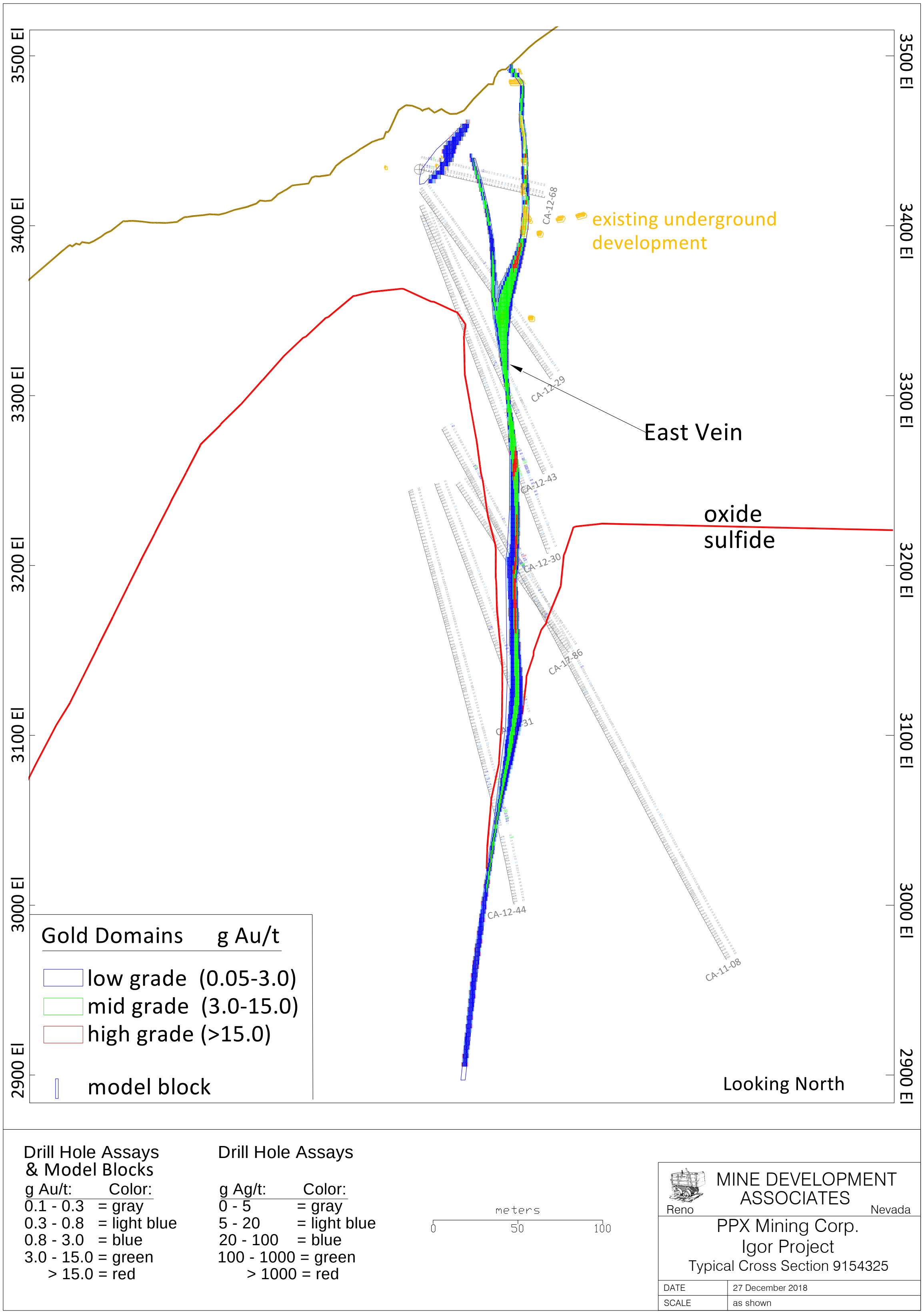


Figure 14.4 Cross Section 9154325 Showing Block Model Gold Grades



14.12 Model Checks

Volumes indicated by the sectional mineral-domain modeling were compared to the long-section volumes and those coded to the block model to assure close agreement, and all block-model coding was checked visually on the computer. Nearest-neighbor and ordinary-krige estimates of the Callanquitas resources were undertaken as a check on the inverse-distance-cubed resource model. Grade-distribution plots of assays and composites versus the nearest-neighbor, krige, and inverse-distance block grades were also evaluated as a check on the estimation. Finally, the inverse-distance-cubed grades were visually compared to the drill-hole assay data to assure that reasonable results were obtained.

14.13 Resources Exclusive of Reserves

The reported Callanquitas resources exclusive of the reserves stated in Section 15.0 are shown in Table 14.14.

Table 14.14 Callanquitas Reported Resources Exclusive of Reserves

Type	Class	Tonnes	g Au/t	oz Au	g Ag/t	oz Ag
Oxide	Measured	97,000	5.255	16,000	63.0	197,000
	Indicated	380,000	3.711	45,000	73.8	901,000
	Meas. + Ind.	477,000	3.978	61,000	71.6	1,098,000
Oxide	Inferred	329,000	4.550	48,000	122.6	1,294,000
Sulfide	Inferred	253,000	2.952	24,000	147.2	1,195,000

Note: rounding may cause apparent inconsistencies

The mine design solids discussed in Section 15.0 were used to code the model and any block that has any portion within the mine design solids is removed from the “exclusive of reserves” resource tabulation.

14.14 Comments on the Resource Modeling and Estimate

The modeling and estimation of the Callanquitas resource is a reasonable representation of the high-grade, narrow vein/structure as observed in both drill core and within the current underground development. The reported resource is based on sufficient geologic and geochemical data to provide confidence in the resource estimate. Therefore, the model and resource estimate can be used for further economic evaluation and the development of reserves.

The large majority of the resource estimate is based on core drilling data and these areas are classified as either Indicated or Inferred resources. The current underground development has provided a detailed view of the mineralized structures, thereby increasing confidence in the model and a classification of Measured for the gold and silver resources associated with the development areas.

The risk within the resource model are primarily the local continuity of the high-grade areas within the mineralized structures. Post-mineral tectonic breccias can displace the mineralized breccia/vein material, resulting in a sharp transition from high- to potentially sub-economic metal grades. This risk is considered low for the general deposit but can come into play on a local scale during mining.

Callanquitas mineralization is considered open at depth and also along strike both to the north and south. To the south, the Callanquitas structural zone extends into the Portachuelos target zone which has been the focus of much of PPX's most recent exploration. An expansion and infill drill program is recommended to increase reported resources and to convert the currently Inferred resource to a higher classification.

15.0 MINERAL RESERVE ESTIMATES (ITEM 15)

15.1 Introduction

Mr. Seamons classifies reserves in order of increasing confidence into Probable and Proven categories to be in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore Canadian National Instrument 43-101. CIM mineral reserve definitions are given below, with CIM's explanatory material shown in italics:

Mineral Reserve

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant Modifying Factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

'Reference point' refers to the mining or process point at which the Qualified Person prepares a Mineral Reserve. For example, most metal deposits disclose mineral reserves with a "mill feed" reference point. In these cases, reserves are reported as mined ore delivered to the plant and do not include reductions attributed to anticipated plant losses. In contrast, coal reserves have traditionally been reported as tonnes of "clean coal". In this coal example, reserves are reported as a "saleable product" reference point and include reductions for plant yield (recovery). The Qualified Person must clearly state the 'reference point' used in the Mineral Reserve estimate.

Probable Mineral Reserve

A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

A *The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.*

Proven Mineral Reserve

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve.

Modifying Factors

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

15.2 Economic Parameters

The Igor mineralized material will be processed through a heap leach system to produce both gold and silver metal in the form of gold doré. Therefore, an AuEq value was calculated for each block in the resource block model and used for the purposes of calculating the mineral reserve estimates. The AuEq calculation used the formulas shown in Equation 1 below. The cutoff grade calculation determined the lowest grade for economic material and was based on a grade of AuEq. The cutoff grade calculations utilized the formula shown in Equation 2.

Equation 1. Grade AuEq Calculation

$$Grade_{AuEq} = \left(\frac{Price_{Ag} * \ Recovery_{Ag}}{Price_{Au} * \ Recovery_{Au}} * Grade_{Ag}\right) + Grade_{Au}$$

Variables

Price Ag: silver price in dollars per ounce;

Price Au: gold price in dollars per ounce;

Recovery Ag: the recovery (%) of silver;

Recovery Au: the recovery (%) of gold;

Grade Ag: the grade of silver in g Ag/t for each block in the block model; and

Grade Au: the grade of gold in g Au/t for each block in the block model

Equation 2 Cutoff grade for AuEq

$$Cutoff_{AuEq} = \left(\frac{Mining\ Cost + Processing\ Cost + G\&A\ Cost}{Recovery_{Au} * (Price_{Au} - Refining\ Cost)}\right) * \left(\frac{31.1035g}{1oz}\right)$$

Variables

Cutoff AuEq: the cutoff grade for AuEq;

Mining Cost: the mining cost in $/t;

Processing Cost: the processing cost in $/t;

G&A Cost: general and administrative cost in $/t;

Recovery Au: the (%) recovery of gold;

Price Au: gold price in dollars per ounce; and

Refining Cost: the refining cost in dollars per ounce

Cutoff grade parameters were provided by PPX and were based on current actual costs, metallurgical studies, and contracts. Initial and final economic parameters are shown in Table 15.1. The initial values were used to calculate a cutoff grade of 2.33g AuEq/t. The cutoff grade value of 2.50g AuEq/t was used to define underground mineable material for reserves. The final numbers originated from the cash flow calculations and the calculated cutoff grade using these numbers is 2.45g AuEq/t which is lower than the cutoff grade value used to define underground mineable material for reserves. The lowest AuEq grade from the mineable stope shapes is 2.80g AuEq/t which is higher than the final cutoff grade calculation. Therefore, the stated reserves are economically viable and justifiable.

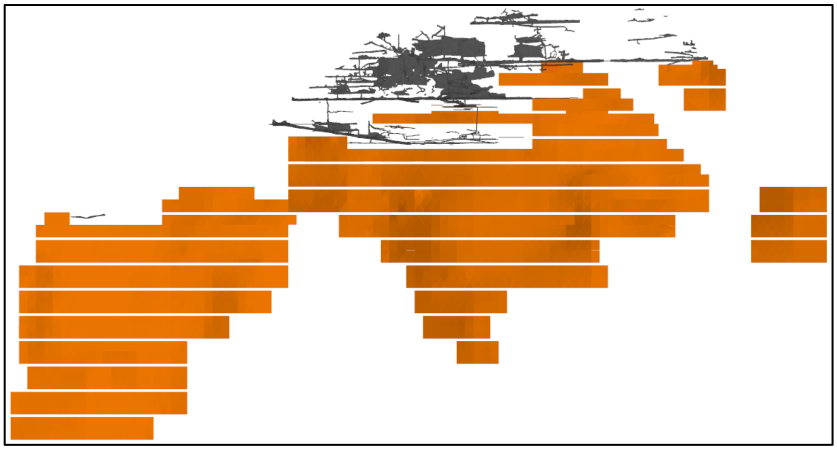
Table 15.1 Cutoff Grade Parameters and Calculation

		Economic Parameters	
Item	**Units**	**Initial**	**Final**
Au Recovery	%	80%	80%
Ag Recovery	%	33%	33%
Mining Cost	$/t	$ 50.00	$ 58.94
Processing Cost	$/t	$ 22.00	$ 17.13
Au Price	$/oz	$ 1,250.00	$ 1,250.00
Ag Price	$/oz	$ 17.00	$ 16.50
G&A Cost ($/t)	$/t	$ 3.00	$ 2.42
Au Refining cost	$/oz	$ 1.00	$ 3.00
AuEq Cutoff Grade	g AuEq/t	2.33	2.45
Grade used for Deswik.SO	g AuEq/t	2.50	
Lowest Grade Stope	g AuEq/t	2.80	

15.3 Minable Shape Optimizer

The reserves estimate utilized Deswik™ Stope Optimizer Software (Deswik.SO™ version 2018.3) which is based on the Alford Mining Systems Stope Shape Optimiser (http://alfordminingsystems.com/). The stope shapes were optimized using a cutoff grade of 2.5g AuEq/t. Only measured and indicated blocks were targeted during the optimization. The level height was 15 meters. The section length was 2.0 meters. The minimum stope width was 0.5 meters. The minimum stope pillar was 2 meters. The stope shapes were merged together to include everything within a level and zone. Any stope shapes that were intersecting survey as-built workings were removed from the reserves estimate. A sill pillar of 3.0 meters at the top of each lift was removed from the reserves estimate as shown in Figure 15.1. The sequence of the mining method requires some material to be mined before access is granted to other areas. This material is considered "must-take" material. Any gaps or inconsistencies within the must-take material were added to the stope shapes.

Figure 15.1 Stopes Shapes from Stope Optimizer with Adjustments



15.4 Dilution and Ore Loss

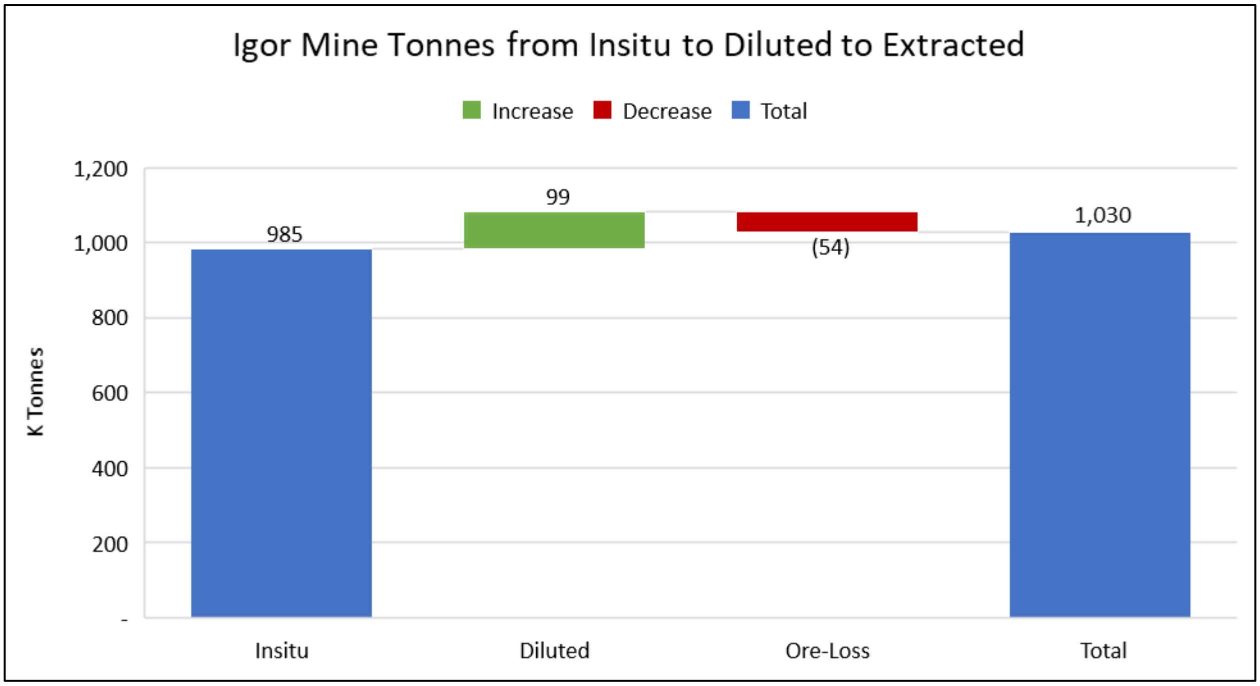
The mining method of modified cut and fill will result in some dilution. Some dilution will be added from the ribs when blasting. The dilution from the ribs will have grade and metal contained within the tonnes. Some dilution will be added from mucking on top of waste fill. The dilution from mucking on top of waste fill will not have grade nor metal contained within the tonnes. The in-situ tonnes were increased by a modifying factor of 10% to account for dilution. To determine the grade of the dilution, shells were drawn around each stope shape at a thickness of 0.5 meters. The block model was interrogated to determine the grades of each stope shell. The grades of the shells were decreased by 50% to account for the dilution from mucking on top of waste fill. The resulting dilution grades for each level can be found in Table 15.2.

Table 15.2 The Grades for Dilution shells by Zone-level.

Level	Au (g/t)	Ag (g/t)
N-3080	0.79	27.30
N-3095	0.83	28.07
N-3110	0.78	49.21
N-3125	0.71	49.70
N-3140	0.60	54.11
N-3155	0.62	58.13
N-3170	0.62	37.92
N-3185	0.61	29.43
N-3200	0.61	29.53
N-3215	0.66	32.93
N-3230	0.63	25.02
N-3245	0.66	22.80
N-3260	0.77	22.17
N-3275	0.66	19.97
N-3290	0.67	17.72
N-3305	0.69	16.16
N-3320	0.69	15.10
N-3335	0.76	10.71
N-3350	0.82	3.34
N-3365	0.81	6.64
N-3380	0.80	4.24
N-3395	0.90	6.57
N-3410	0.73	6.87
N-3425	0.80	7.00
S-2990	0.60	19.39
S-3005	0.63	18.47
S-3020	0.62	19.71
S-3035	0.60	22.93
S-3050	0.56	32.35
S-3065	0.65	39.59
S-3080	0.61	26.38
S-3095	0.65	29.45
S-3110	0.57	28.96
S-3125	0.56	28.25
S-3140	0.72	25.96
S-3155	0.64	20.40
S-3170	0.71	15.53
S-3185	0.74	16.58
S-3200	0.63	18.15
S-3215	0.61	16.72
S-3230	0.62	17.56
S-3245	0.73	20.10
S-3260	0.75	16.76
S-3275	0.72	18.37

Experience shows that the mining designs will not be executed with a 100% accuracy. Some material will be lost for various unforeseen reasons. The tonnes after dilution were decreased by 5% to account for this ore-loss. The effects of dilution and ore-loss can be seen in Figure 15.2.

Figure 15.2 Effects of Dilution and Ore-loss on Tonnes



15.5 Mineral Reserves

Proven and Probable reserves have been estimated using only the Measured and Indicated resources. The estimates use the economic and dilution factors described in the previous sections. The reserves were calculated based on the proportion, the gold grades, and the silver grades of the Measured and Indicated resource blocks contained within each designed stope. Table 15.3 shows the fully diluted Proven and Probable reserves. The effective date of the estimated reserves is December 3, 2018.

Table 15.3 Igor Mine Proven and Probable Reserves

Type	Classification	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	g AuEq/t	K Ozs AuEq
Oxide	Proven	8	4.56	1	32.3	8	4.74	1
	Probable	1,022	4.09	134	104.6	3,437	4.66	153
	Proven & Probable	**1,030**	**4.10**	**136**	**104.1**	**3,445**	**4.66**	**154**

- *CIM Definitions and Standards were followed for Mineral Resource estimates;*
- *Proven and Probable reserves are based on Measured and Indicated resources;*
- *Reserves are reported as fully diluted and reflect a 95% extraction rate (5% ore loss);*
- *Reserves are reported based on $1,250 per ounce gold and $16.50 per ounce silver prices;*
- *Equivalent gold is reported based on metal prices and 80% recovery of gold and 33% recovery of silver;*
- *Reserves are reported based on a cutoff grade of 2.50g AuEq/t, used to define underground mineable material;*
- *The point of reference is delivery to the crushing facility; and*
- *Some apparent discrepancies are due to rounding.*

15.6 Discussion of Reserves

The effects of dilution, extraction, and recovery on metal content can be found in Figure 15.3 and Figure 15.4. The reference point at which Mineral Reserves are defined is at delivery to the crushing facility. The estimated Proven and Probable reserves are entirely within the estimated Measured and Indicated mineral resources. The author is unaware of any other mining, metallurgical, infrastructure, permitting, and other relevant factors that could materially affect the estimated mineral reserves.

Figure 15.3 Effects of Dilution, Extraction, and Recovery on Gold Ounces

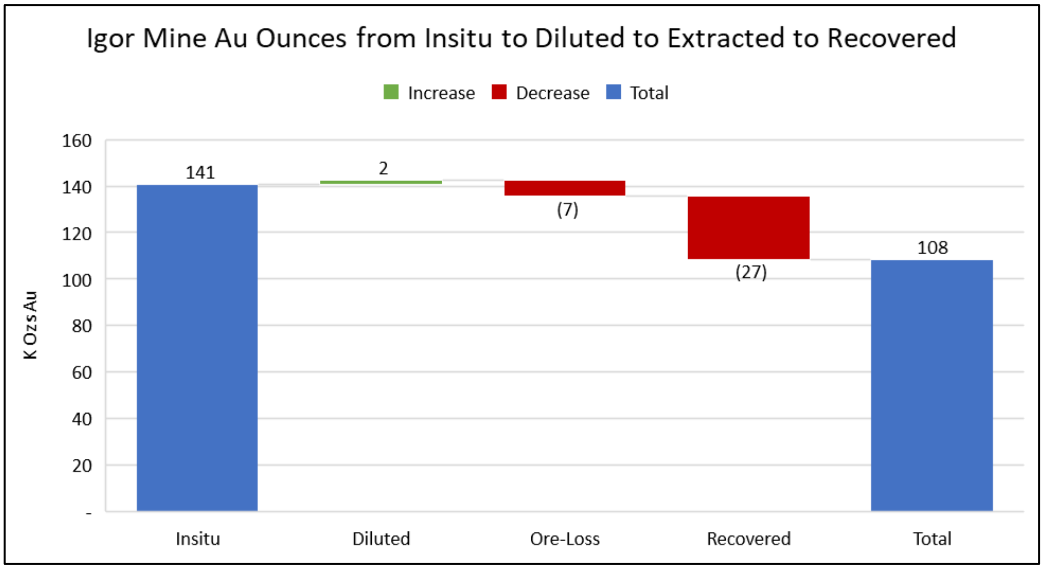
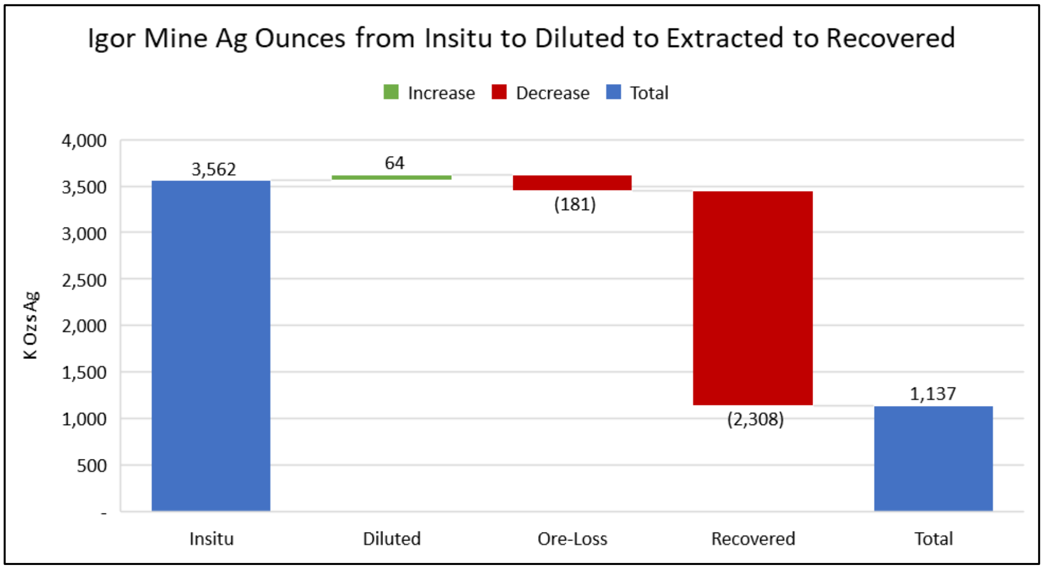


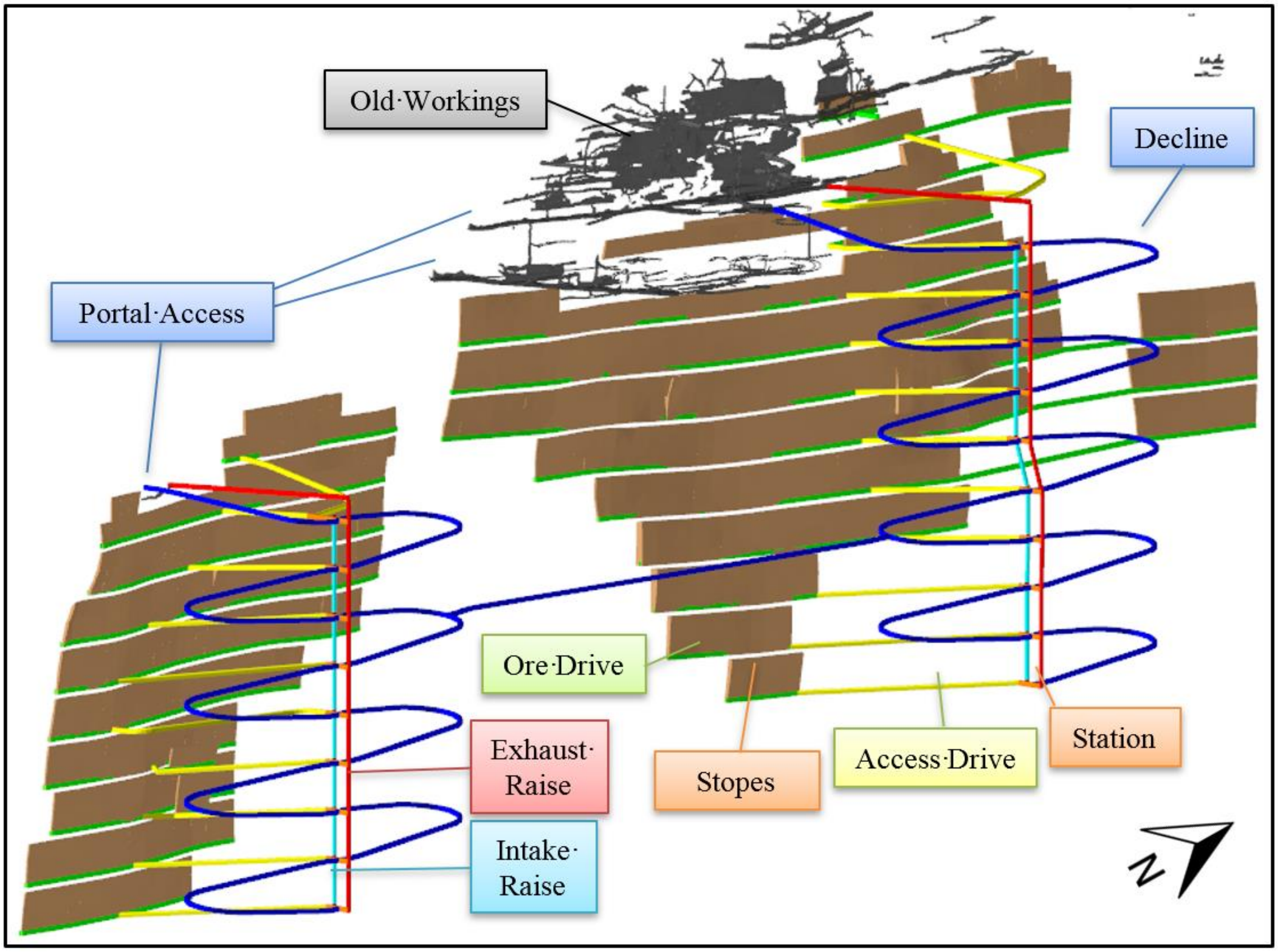
Figure 15.4 Effects of Dilution, Extraction, and Recovery on Silver Ounces



16.0 MINING METHODS (ITEM 16)

The Igor mine will be accessed via two declines and a system of internal ramps. Two raises are included in the design to be used for ventilation, secondary egress, and ore passes as shown in the isometric view in Figure 16.1. The planned mining method is a modified cut and fill.

Figure 16.1 Mine Layout in Isometric-View

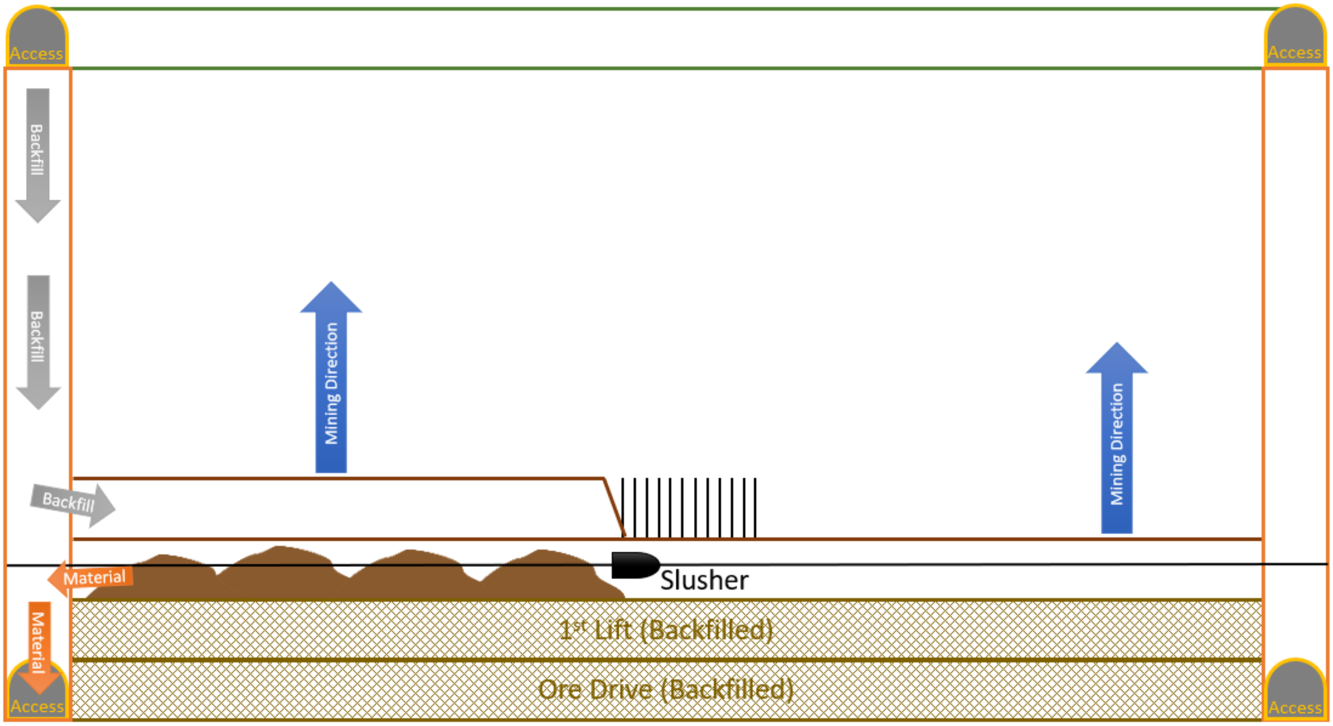


16.1 Mining Method

16.1.1 Current Mining Method

The current mining method is a modified cut and fill as shown in Figure 16.2. The development is mined first to access the mining area. Two raises are mined on both ends of the mining area. Then an ore drive is mined along the strike of the orebody. The first lift is mined by drilling and blasting up holes. The material is moved out of the drift via a slusher bucket and the raises. After the first lift is mined out the drift is filled by dropping waste fill down the raise and into the drift using the slusher bucket. This process is repeated until the mining area is mined out.

Figure 16.2 Illustration of Current Igor Mine Modified Cut and Fill

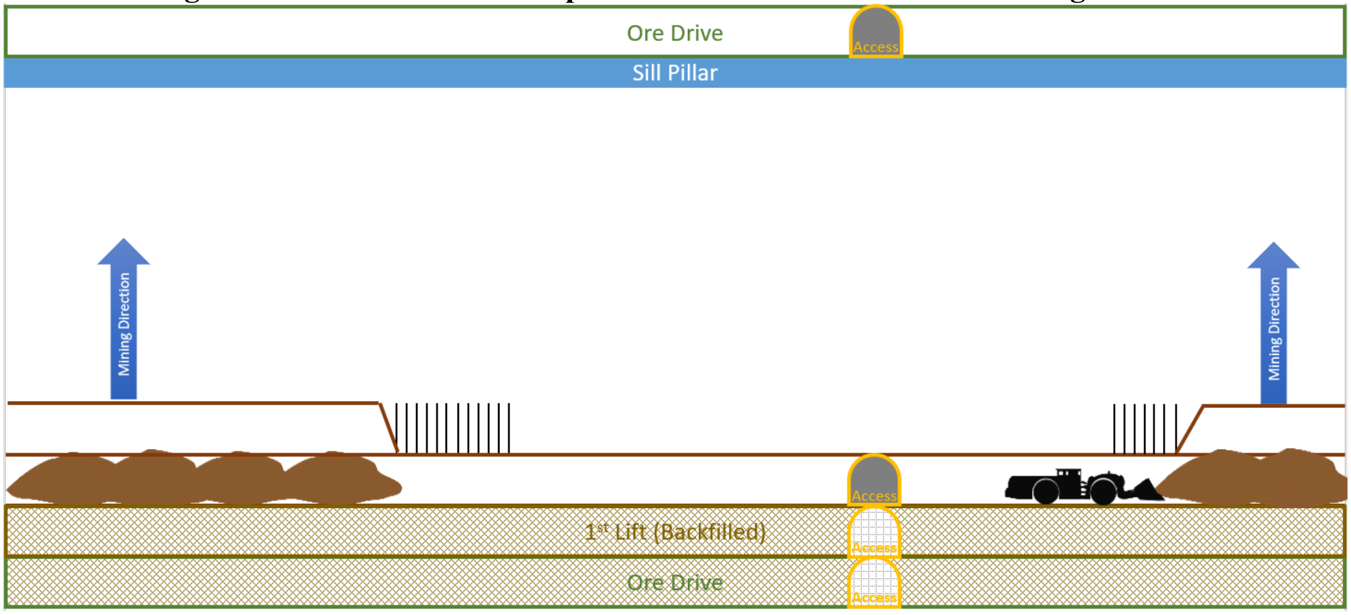


16.1.2 Proposed Mining Method

The Nicholas (1981) approach was used to assist in determine the mining method. This process suggested shrinkage stoping and cut and fill stoping as the top two results. A modified cut and fill method has been selected to extract material from the Igor mine due to the geometry for the orebody, the depth of the orebody, and the similarities to the current mining method.

The modified cut and fill mining method will start by mining all the capital development on a level including the decline, station levels, and raises. After the capital development for a level is complete, the access drive will be mined from the station level to the ore drive at a -12.5% grade. The ore drive runs along strike of the orebody. After the ore drive is mined out, the first lift will be mined by drilling up holes into the back and blasting down. The material will be moved out of the drift via an underground loader and trucks. After the first lift is mined out, the ore drive will be filled with waste rock via an underground loader and trucks. The access drive will then be adjusted to access the first lift. This process will continue until the sill pillar is reached as shown in Figure 16.3.

Figure 16.3 Illustration of Proposed Modified Cut and Fill for the Igor Mine



16.1.3 Mine Design

The level layout consists of a decline, level station, exhaust raise, intake raise, level access, ore drives, and stopes as shown in Figure 16.4. A section view and a plan map can be found in Figure 16.5 and Figure 16.6 respectively. The raises will also be used as a material pass system. The access drift is designed at -12.5% grade. The access drift will be adjusted upwards after each lift is mined and filled. The resulting access drift will be at +12.5% grade. The decline is also designed at -12.5% grade.

Figure 16.4 Development and Level Design

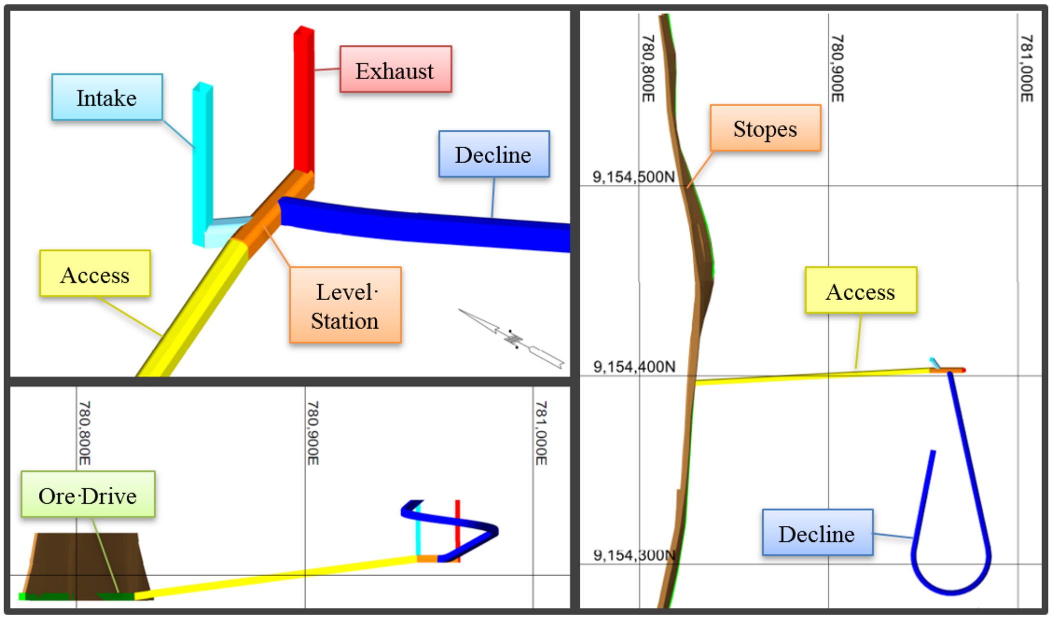


Figure 16.5 Section View of Igor Mine (Looking West)

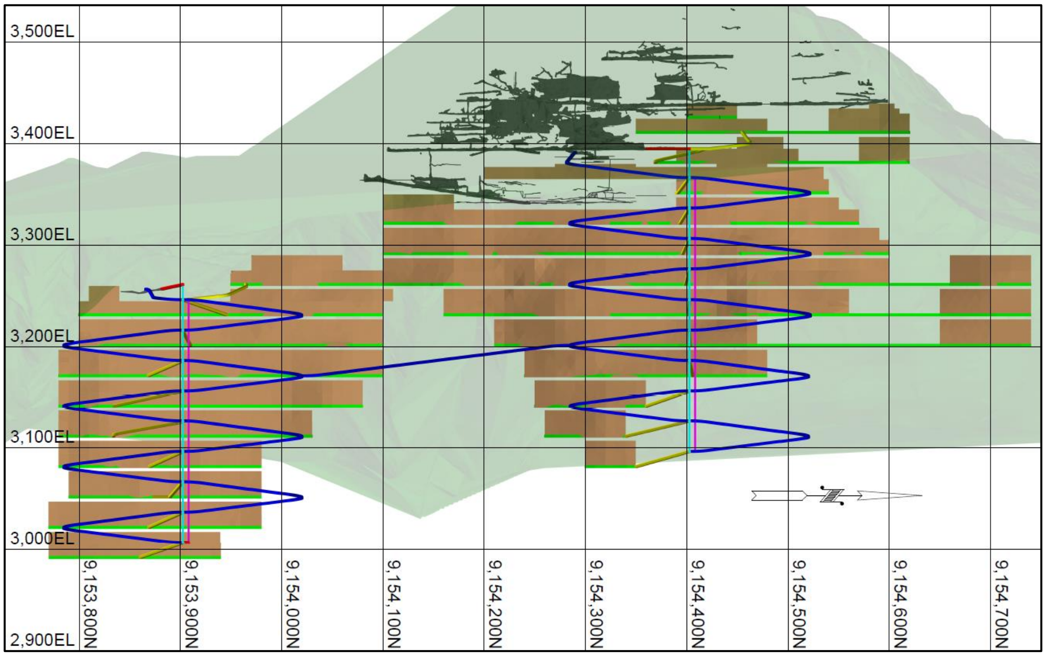
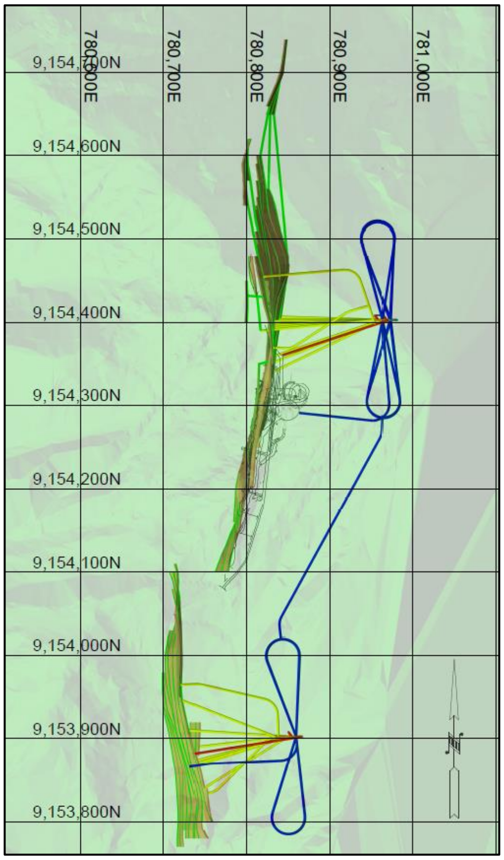


Figure 16.6 Plan View of Igor Mine



16.2 Mining Parameters

16.2.1 Geotechnical

The current ground support used in development is wire mesh with 1.5 meter and 2.1 meter split sets on a 1.0 meter by 1.0 meter spacing with spot bolting as necessary (Figure 16.7). The current ground support used in ore-drives and stopes is timber sets as shown in Figure 16.7.

Figure 16.7 Current Ground Support, Igor Mine

Development: Split Sets 1m x 1m with wire mesh

Ore-drives and stopes: Timber Sets

No geotechnical analysis has been completed. No geotechnical measurements have been taken. The geotechnical conditions are largely unknown. Mr. Seamons recommends that a proper geotechnical evaluation of the ground at the Igor mine is conducted.

16.2.2 Hydrological Conditions

Water is present at the current mine as shown in Figure 16.8. This water is believed to be surface water or excess mine water from the use of mining equipment. The current workings are above the water table, but the depth to the water table is not known, no hydrological analysis has been completed and no hydrological measurements have been taken. The hydrological conditions are largely unknown. Mr. Seamons recommends that PPX carry out a proper hydrological evaluation of the Igor mine.

Figure 16.8 Photo Showing Water in the Igor Mine

16.3 Mining Physicals

16.3.1 Production Rates

The production rates for the Igor mine are listed in Table 16.1. The production rates were applied to each task based on the mine type. A quantity constraint was applied to the tonnes going to the processing facility by year as shown in Table 16.2. The list of mine production types and the number of types used in the scheduling logic is shown in Table 16.3.

Table 16.1 Igor Mine Production Rates

Production Rates		
Mine Type	**Task Rate**	**Units**
Decline	2.4	meters/day
Station	2.4	meters/day
Vent Drift	2.4	meters/day
Access	2.0	meters/day
Ore-Drives	2.0	meters/day
Vent Raises	2.0	meters/day
Stopes	360.0	tpd
Backfill	360.0	tpd

Table 16.2 Tonnage Limits for the Scheduling Model

Period	Quantity Constraint	
Year 1	350	tpd
Year 2	425	tpd
Year 3 to LOM	500	tpd

Table 16.3 Mine Production Types in the Scheduling Model

Name	Number of Resources
Stope Production	3
Stope Backfilling	3
Lateral Development	5
Vertical Development	1

16.3.2 Drift Dimensions and Physicals

The drift dimensions for the decline, access drift, and level station are 3 meters wide by 3 meters in height with an arched back. The drift dimensions for the ore drives are 3 meters wide by 3 meters in height, without an arched back. The raises are 2 meters by 2 meters square. The drift dimensions can be found in Figure 16.9. The life of mine ("LOM") drift physicals are listed in Table 16.4.

Figure 16.9 Drift Dimensions (not to scale)

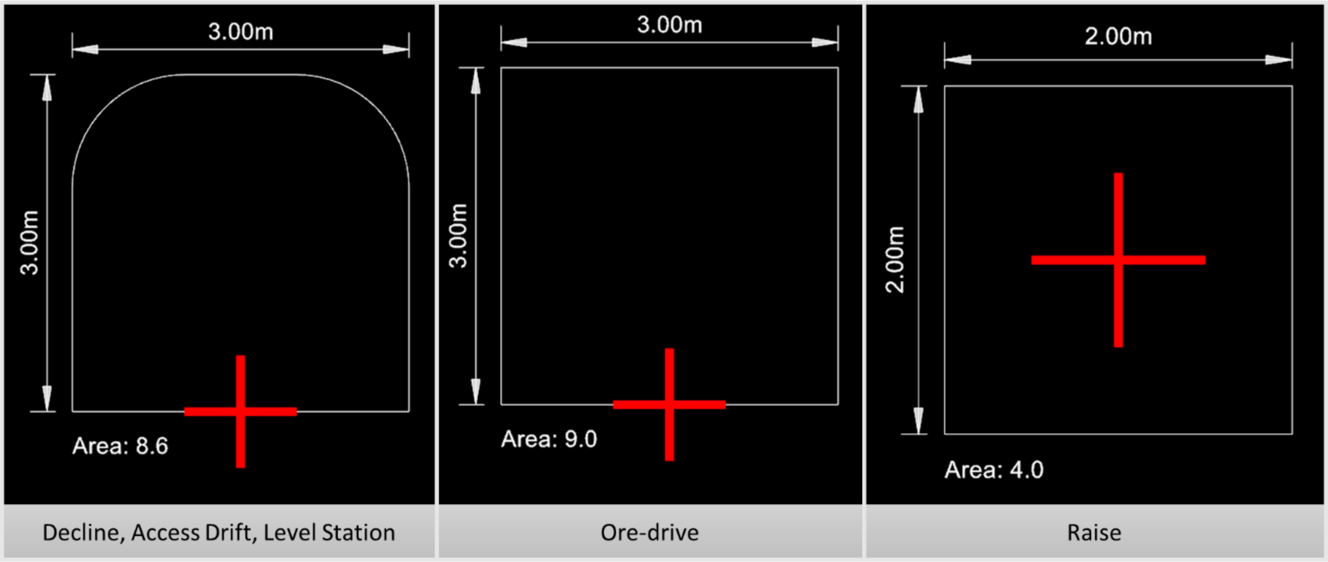


Table 16.4 Drift Physicals for Life of Mine

	Primary Development		
Mine Type	**Profile**	**Meters**	**Tonnes**
Decline	3.0m x 3.0m Arched	5,129	118,064
Station	3.0m x 3.0m Arched	355	8,178
Vent Drift	3.0m x 3.0m Arched	366	8,326
	Total Primary	5,850	134,567
	Expensed Development		
Mine Type	**Profile**	**Meters**	**Tonnes**
Access	3.0m x 3.0m Arched	2,940	67,431
Ore Drives	3.0m x 3.0m Square	6,827	161,237
	Total Expensed	9,767	228,668
	Raises		
Mine Type	**Profile**	**Meters**	**Tonnes**
Vent Raises	2.0m x 2.0m Square	1,013	10,890
	Total Raises	1,013	10,890
	Grand Total	**16,630**	**374,126**

16.3.3 Mining Schedule

For the purpose of scheduling and mine planning, the mining designs were grouped into two areas as shown in Figure 16.10. The production schedule was produced based on the availability of access to each mining task and the quantity of resources available. The development schedule is shown in Table 16.5 and the production schedule is shown in Table 16.6, for a planned mine life of seven years. The development schedule can be seen on a section map with the development colored by time period in Figure 16.11.

Figure 16.10 Mining Areas, Igor mine.

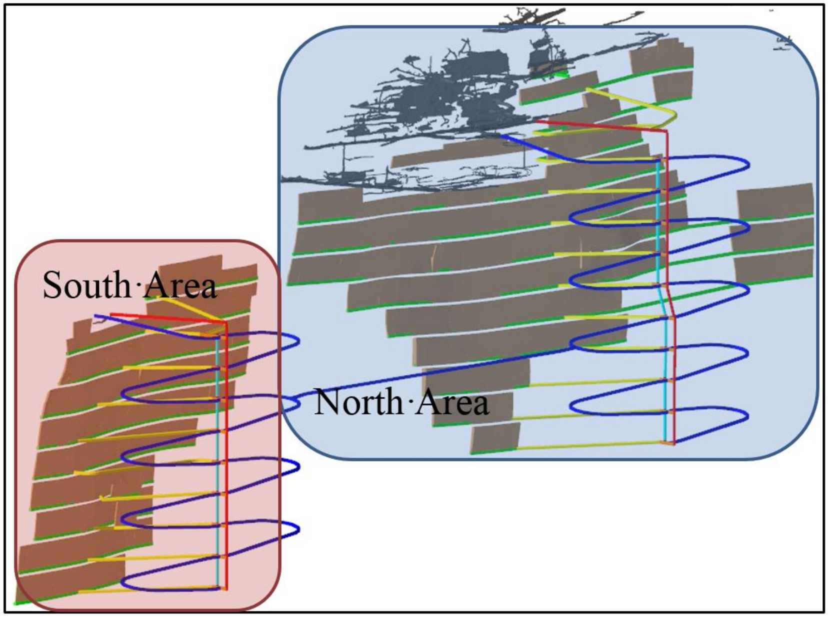


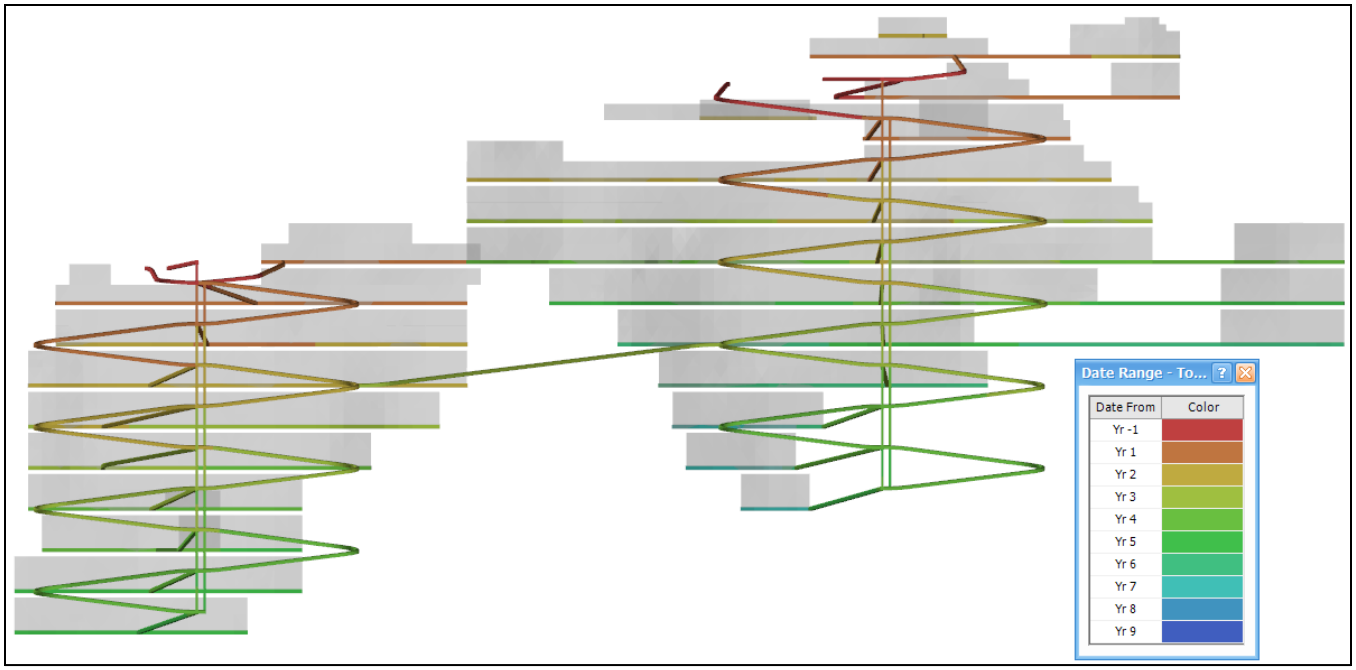
Table 16.5 Mine Development Schedule

	Profile	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Primary Development													
Decline	3.0m x 3.0m Arched	m	357	1,022	1,129	1,511	1,109	-	-	-	-	-	5,129
Station	3.0m x 3.0m Arched	m	33	70	91	72	90	-	-	-	-	-	355
Vent Drift	3.0m x 3.0m Arched	m	272	16	31	21	26	-	-	-	-	-	366
	Total Primary	**m**	**662**	**1,108**	**1,251**	**1,604**	**1,225**	**-**	**-**	**-**	**-**	**-**	**5,850**
Expensed Development													-
Access	3.0m x 3.0m Arched	m	298	670	517	627	569	259	-	-	-	-	2,940
Ore Drives	3.0m x 3.0m Square	m	20	1,240	1,453	943	1,256	1,015	677	222	-	-	6,827
	Total Expensed	**m**	**318**	**1,910**	**1,970**	**1,571**	**1,825**	**1,274**	**677**	**222**	**-**	**-**	**9,767**
Raises													-
Vent Raises	2.0m x 2.0m Square	m	14	141	245	289	237	87	-	-	-	-	1,013
	Total Raises	**m**	**14**	**141**	**245**	**289**	**237**	**87**	**-**	**-**	**-**	**-**	**1,013**

Table 16.6 Mine Production Schedule

Physical	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
K Tonnes	0	115	153	157	149	161	143	151	-	-	1,030
g Au/t	2.51	3.60	3.94	3.65	3.78	3.96	4.46	5.20	-	-	4.10
K Ozs Au	0	13	19	18	18	21	21	25	-	-	136
g Ag/t	12.28	32.07	41.55	63.41	106.31	144.80	128.47	196.24	-	-	104.08
K Ozs Ag	0	119	205	319	509	750	591	952	-	-	3,445

Figure 16.11 Cross Section with Development Schedule



16.4 Mine Services

16.4.1 Mine Equipment

Mobile equipment required for the mine development and production is summarized in Table 16.7. All mining, including production and development, will be done by mining contractors who will provide the required mobile equipment.

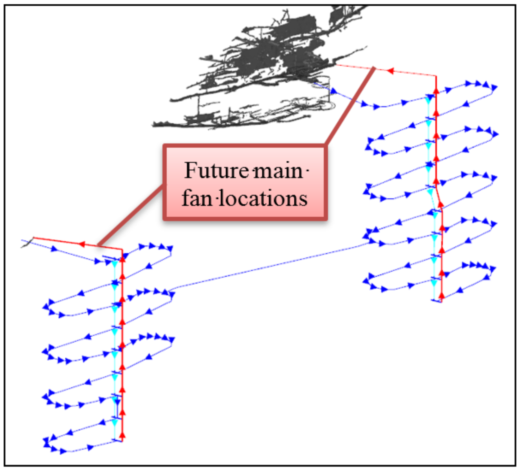
Table 16.7 Underground Mobile Equipment

Equipment	Units
Jacklegs	8
2.4cm LHD	4
15 ton truck	4
Grader	1
Personnel Truck	1
Telehandler	1
Light Vehicle	5
Concrete Truck	1

16.4.2 Ventilation

The primary ventilation circuit will be powered by two main fans, one in the north exhaust drift and one in the south exhaust drift. The fresh air will travel down the decline and down the fresh air raise, and then up the exhaust raise and out the exhaust drift as shown in Figure 16.12. Regulators will be placed on each level at the entrance of the exhaust raise to control the flow of air on each level. The secondary ventilation circuit will have auxiliary fans that take fresh air from the primary ventilation circuit and move the air through ducting to the working face and then exhaust back to the exhaust raise. Mr. Seamons recommends that an adequate ventilation study is conducted to determine fan sizes and air-flow quantities.

Figure 16.12 Primary Ventilation Circuit with Airflow Directions.



16.4.3 Utilities

Water for underground operations and dust control will be supplied through a network of 5.08 centimeter and 10.1 centimeter diameter pipes. Mine discharge water will be collected through a series of sumps and returned to the surface. A series of air compressors will be installed near the portal to supply the necessary compressed air for operations.

17.0 RECOVERY METHODS (ITEM 17)

This section was prepared under the supervision of Dr. Diogenes Uceda. Dr. Uceda has reviewed the information cited in this section and believes the information summarized herein correctly represents the current understanding of the recovery methods for the Callanquitas gold-silver deposit. In cases where design parameters or results were reported in Imperial units of measure, those units are retained to avoid unintended changes in precision and rounding.

Section 13.0 of this report evaluated the metallurgical test work that has been used to derive the preliminary flow sheet for the Igor project (Callanquitas mine). The process plant design considers a heap-leach operation for the LOM. A two-stage development is considered for the heap-leach operation: stage 1 is for mining and treating oxide ores at the rate of 350tpd and stage 2 will mine and heap-leach at the rate of 500tpd. In this evaluation, there are no provisions to implement grinding facilities for treating material from the mine. The estimated reserves are 1,030,000 tonnes, establishing a project LOM of 8.6 years. The process design considers gold and silver recoveries by cyanide heap-leaching of 80% and 33%, respectively.

This section presents the work completed to define the recovery methods and design parameters based on the test work completed for representative samples and the design update completed by METTS. Oxide ore will be crushed in three stages and then processed in a heap leach operation that will utilize a single, multiple-lift and multiple-use leach pad. The planned operation requires 8,000m^2 of leach pad space for the first two years of operation at a nominal feed rate of 350tpd. After the second year, the leached material will be reclaimed and placed as backfill in the mine. Concurrent with this reclamation, a second sector of the pad will be stacked and leached. When all of the first sector of leached material has been reclaimed, stacking of the first sector will resume and the second sector will be reclaimed. Thus, the two sectors of the leach pad will be stacked, leached and reclaimed in an alternating procedure.

Production from the mine will increase the feed to the heap leach pad to 500tpd after the second year. The initial plant design considers a design capacity of 500tpd for crushing equipment and the Merrill Crowe circuit. The gold recovery circuit includes a gold-silver bullion melting facility.

The plant design is supported by testwork completed by METTS and other laboratories for defining ore and leaching characteristics. Ore will be crushed, agglomerated, stockpiled, reclaimed, and stacked on the leach pad with a conveyor stacking system. A crushing circuit is included for sizing the material before cement agglomeration prior to heap-leach stacking. The stacked ore will be leached with a cyanide solution and the resulting pregnant solution will be processed in a Merrill-Crowe gold precipitation circuit to extract gold and silver.

Previous testwork evaluation compiled by METTS and additional tests performed for this study have indicated that material crushed 100% to -9.5 mm, followed by cement agglomeration and cyanidation in heap leach, could have the design plant throughput, head grade, recovery and expected gold production, as summarized in Table 17.1.

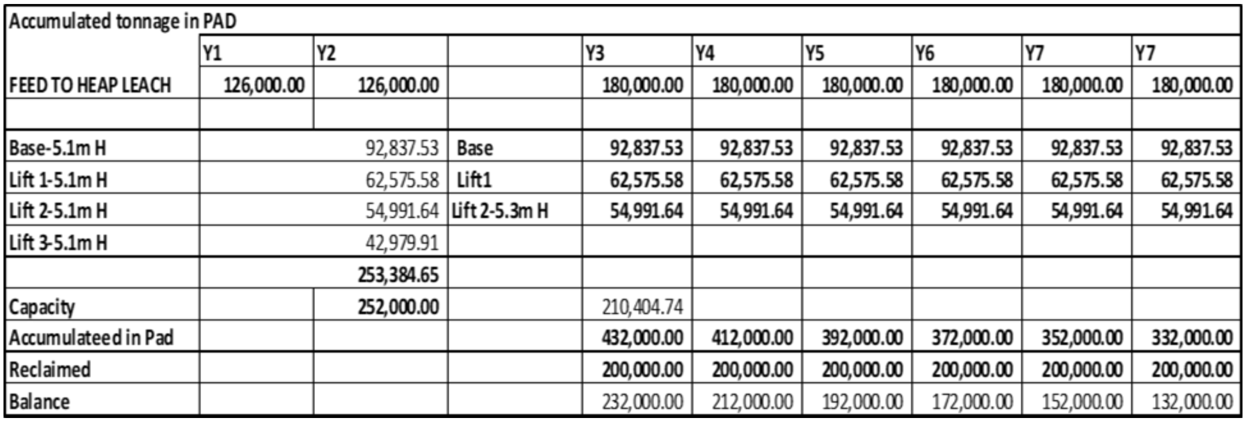
Table 17.1 Design Plant Throughput

	Year	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Throughput									
	Daily, tpd	350	350	500	500	500	500	500	500
	Year, tpy	126000	126000	180000	180000	180000	180000	180000	180000
Head Grad									
	Gold g/t	4.05	4.05	4.05	4.05	4.05	4.05	4.05	4.05
	Silver g/t	90.5	90.5	90.5	90.5	90.5	90.5	90.5	90.5
Recovery									
	Gold %	80.30	80.30	80.30	80.30	80.30	80.30	80.30	80.30
	Silver %	33.03	33.03	33.03	33.03	33.03	33.03	33.03	33.03
Production									
	Oz Gold	13,176	13,176	18,823	18,823	18,823	18,823	18,823	18,823
	Oz Silver	121,107	121,107	173,009	173,009	173,009	173,009	173,009	173,009

The heap leach design considers using an 8,800m^2 pad. By using the full base area and three lifts of 5.1 meters, it will be suitable for receiving a total of 253,384 tonnes which will be adequate for the first two years operating at the rate of 126,000 tonnes per year. This approach is required for holding material in the heap leach until the mine is ready to accept reclaimed material as mine backfill. The planned reclaiming rate from the accumulated leached material is 200,000 tonnes per year.

Table 17.2 presents the tonnage allocation for pad design evaluation. It shows the feasibility of a single pad of 8,800m^2 through the LOM. At the end of operation, 137,792 tonnes will be the balance needing final treatment during closure of the facilities.

Table 17.2 Heap Leach Accumulated Material



Accumulated tonnage in PAD									
	Y1	Y2		Y3	Y4	Y5	Y6	Y7	Y7
FEED TO HEAP LEACH	126,000.00	126,000.00		180,000.00	180,000.00	180,000.00	180,000.00	180,000.00	180,000.00
Base-5.1m H		92,837.53	Base	92,837.53	92,837.53	92,837.53	92,837.53	92,837.53	92,837.53
Lift 1-5.1m H		62,575.58	Lift1	62,575.58	62,575.58	62,575.58	62,575.58	62,575.58	62,575.58
Lift 2-5.1m H		54,991.64	Lift 2-5.3m H	54,991.64	54,991.64	54,991.64	54,991.64	54,991.64	54,991.64
Lift 3-5.1m H		42,979.91							
		253,384.65							
Capacity		252,000.00		210,404.74					
Accumulateed in Pad				432,000.00	412,000.00	392,000.00	372,000.00	352,000.00	332,000.00
Reclaimed				200,000.00	200,000.00	200,000.00	200,000.00	200,000.00	200,000.00
Balance				232,000.00	212,000.00	192,000.00	172,000.00	152,000.00	132,000.00

The process block diagram for Phase 1 is presented in Figure 17.1 and a process representation is included in Figure 17.2. The expanded flow sheet for the crushing operation is presented in Figure 17.3 and the expanded flow sheet of the heap leach and gold recovery system is presented in Figure 17.4. The process general configuration is presented in Figure 17.5.

Figure 17.1 Phase 1 Process Block-Diagram

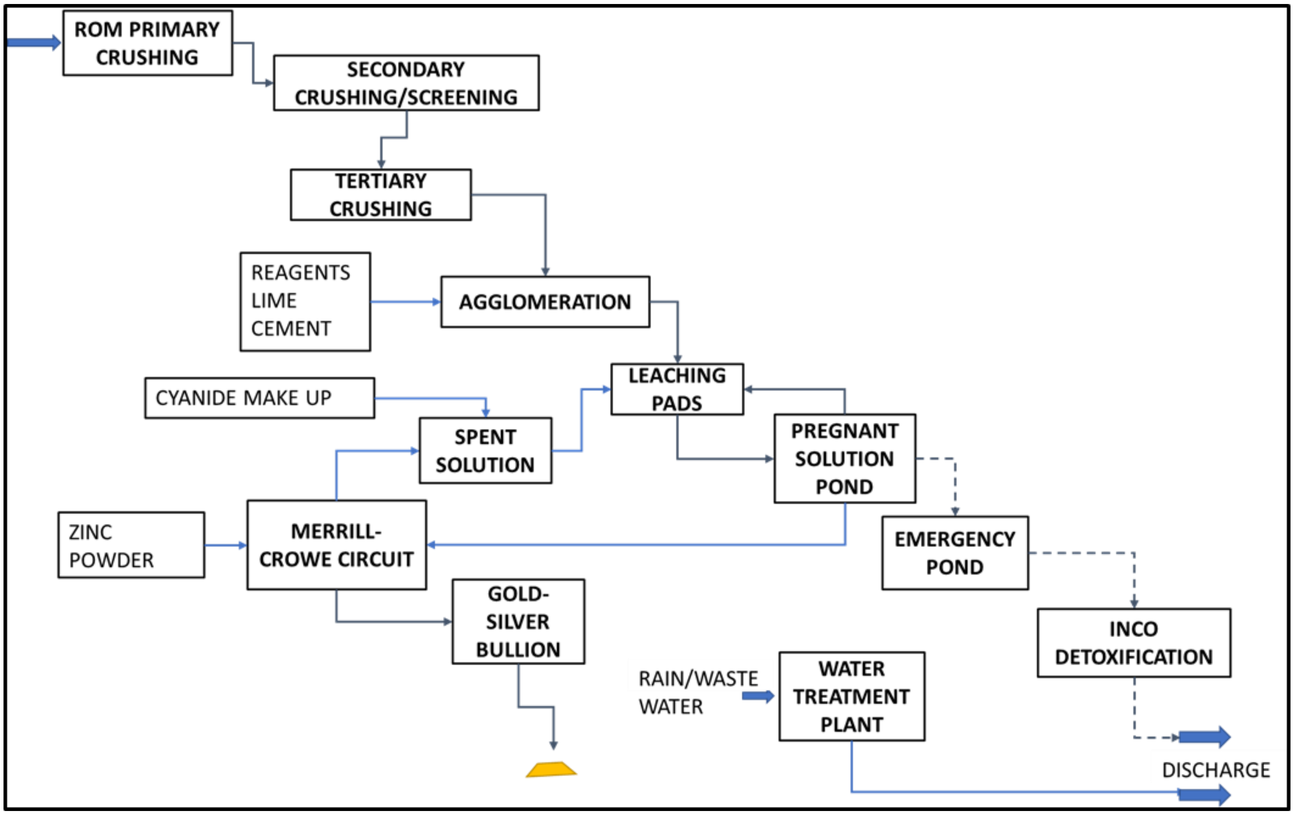


Figure 17.2 Phase 1 Process Flow Sheet

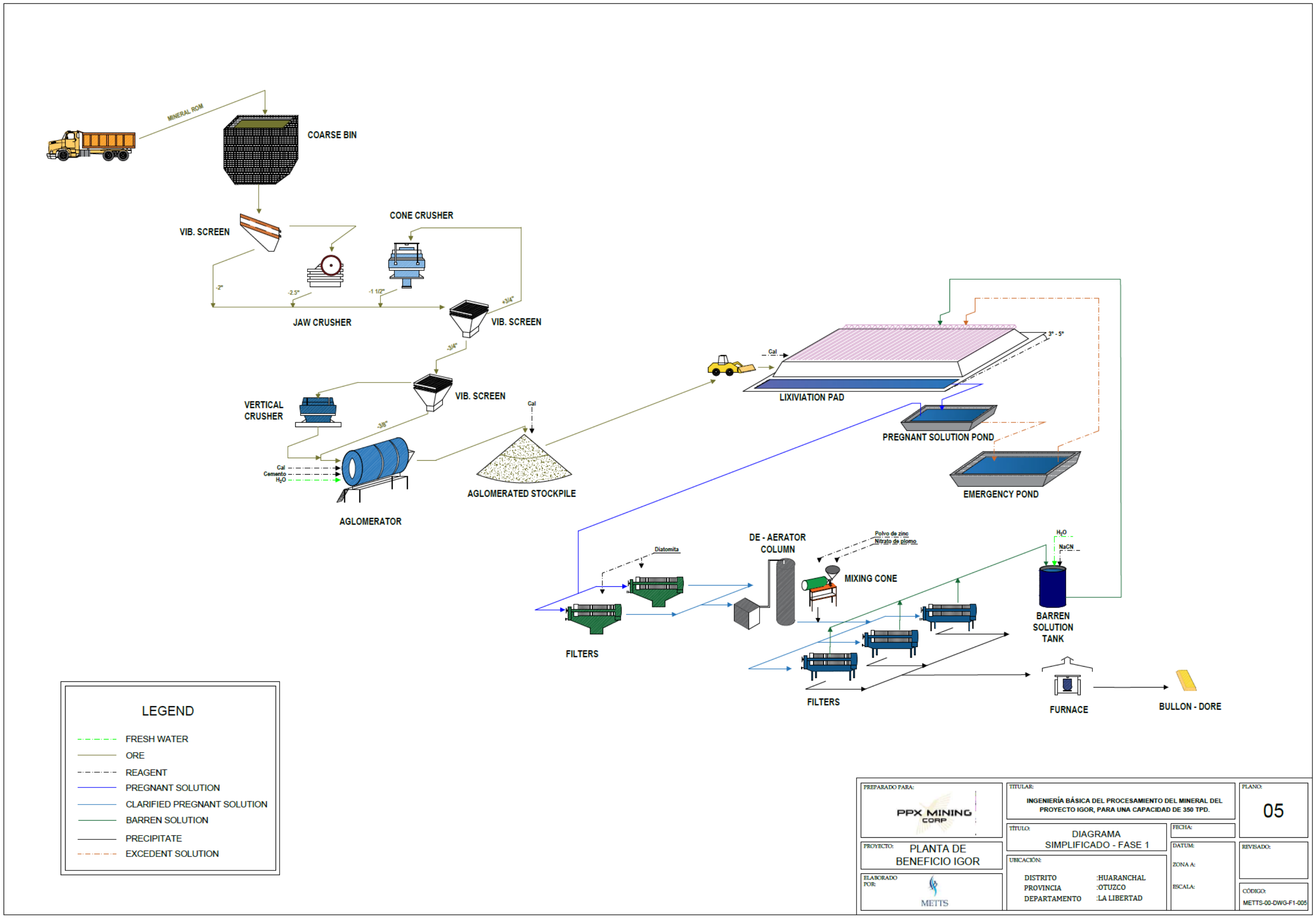


Figure 17.3 Crushing and Agglomeration Stages

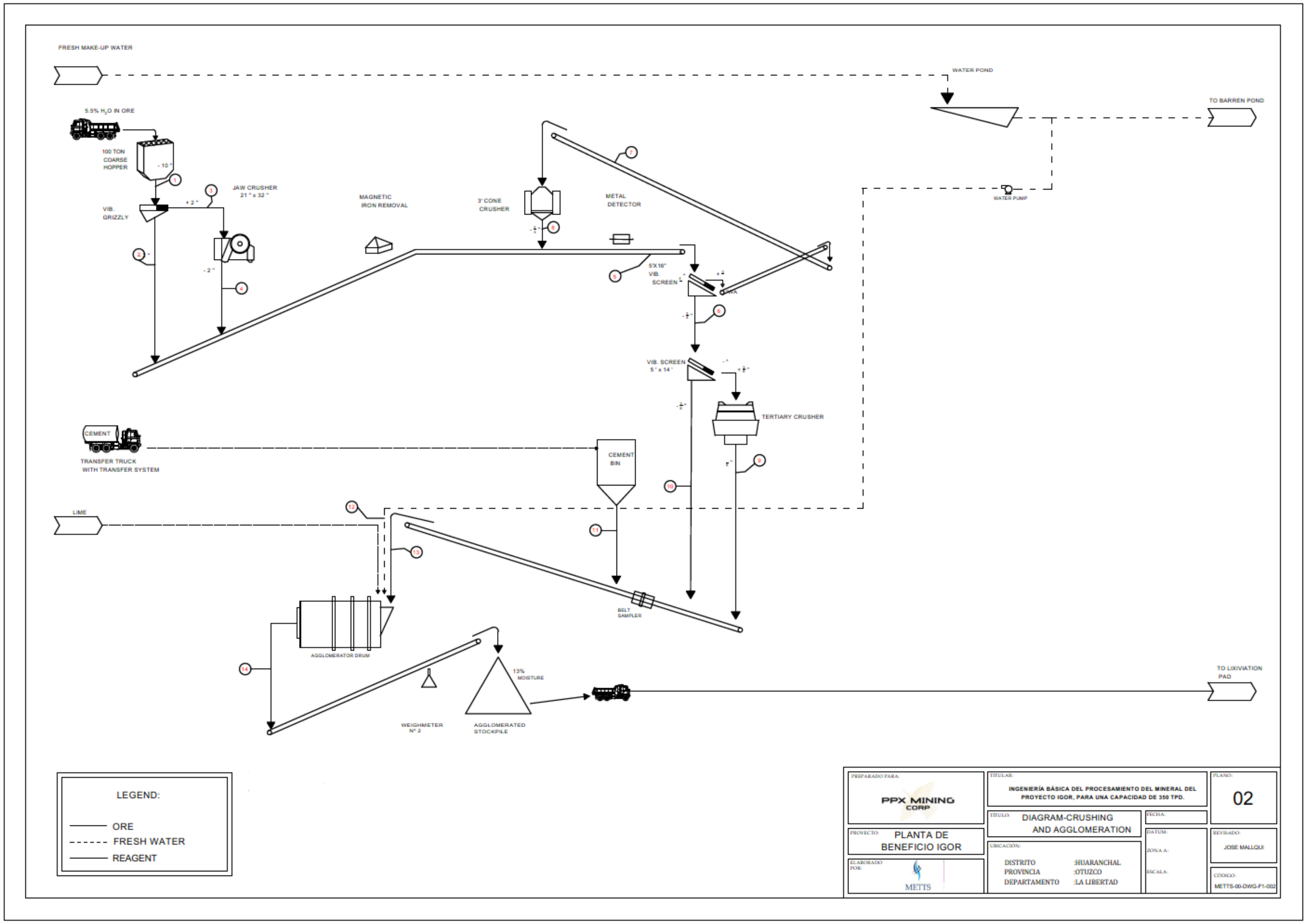


Figure 17.4 Heap-Leach (Lixiviation) Pads and Merrill-Crowe Circuit

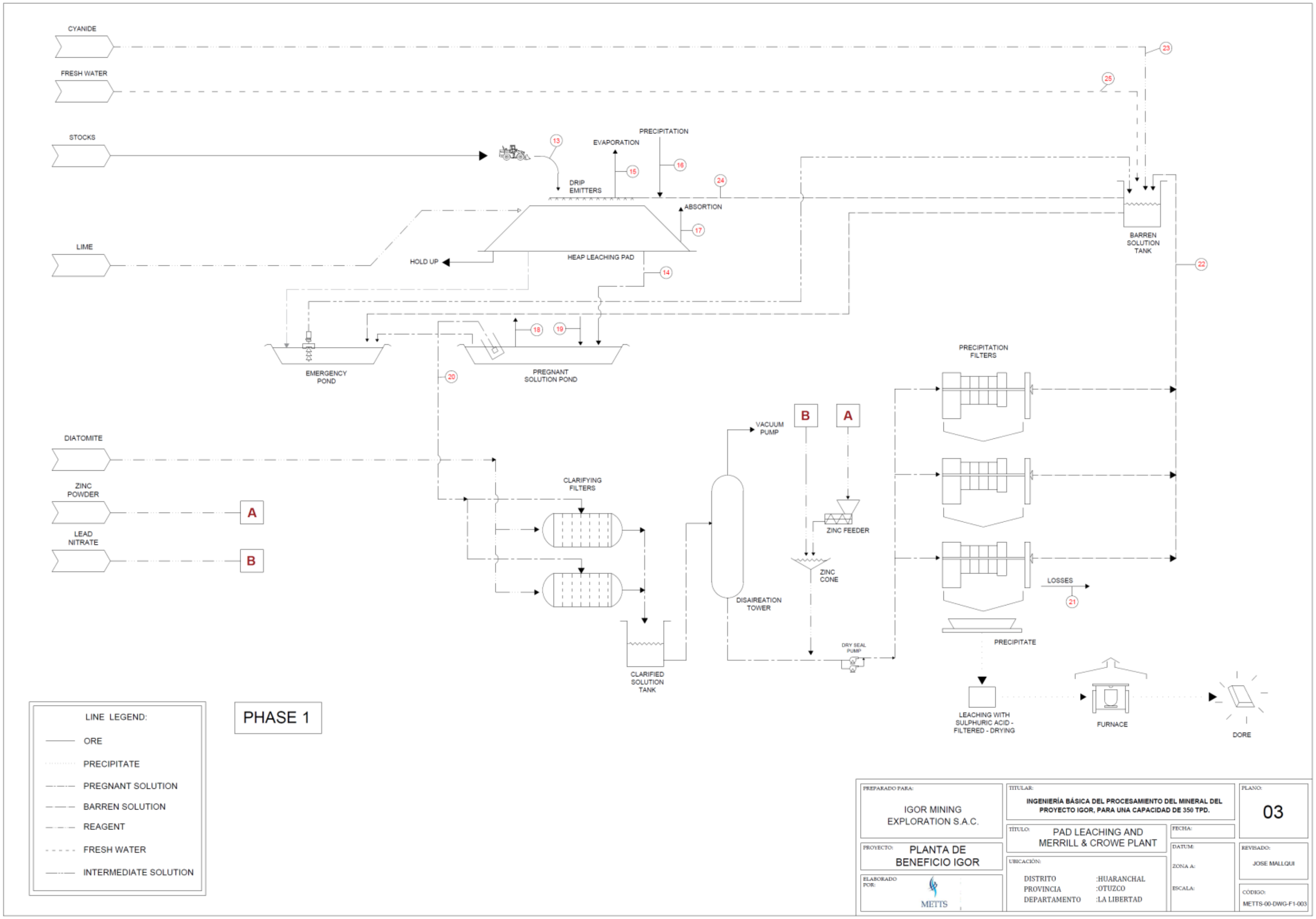
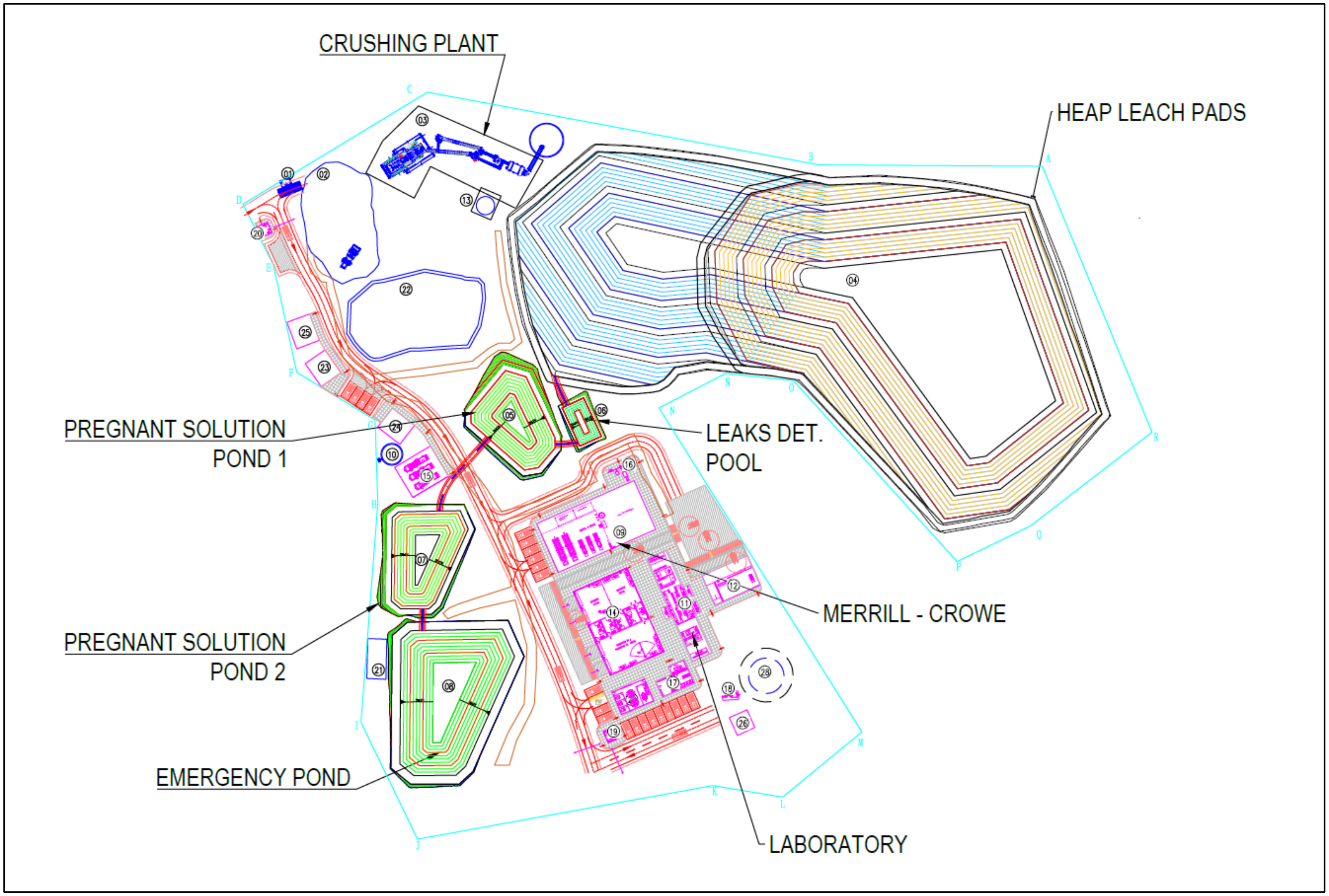


Figure 17.5 General Process Area Layout



17.1 Process and Plant Design

17.1.1 Supporting Data

METTS has used the PPX geological model as a basis for estimating ore production during the LOM and conducted a trade-off study that identified four options for processing the ore. There are two main mineralized structures: Callanquitas Este and Callanquitas Oeste. The mineralized material from Callanquitas Este was considered for the design of the Igor process facility. For the process design, the following head grades were supplied by PPX as shown in Table 17.3:

Table 17.3 Callanquitas Este Head Grade

Element	Head Grade (g/t)
Au	4.05
Ag	90.5

During the period of September-November 2017, METTS completed confirmatory metallurgical tests that defined parameters such as: mineral granulometry for feed to the leach pad, leaching period (days), leaching rate (L/hr/m^2), reagent consumption (lime, cement and cyanide), and gold and silver recoveries expected to be achieved. The following tasks were completed:

- Unit operations defined for crushing, cyanidation, Merrill-Crowe precipitation and melting for obtaining the doré (Au + Ag);
- Calculation of the material balance (water / solids);
- Metallurgical balance estimated for heap leaching and Merrill-Crowe extraction;
- Flow diagrams of the process were created;
- Energy balance was estimated;
- Organizational structure of plant operations was designed;
- Description of the infrastructure location criteria; and
- Confirmatory metallurgical tests.

17.1.2 Process Design Criteria

Based on the testwork and the conditions defined in the evaluation of the requirements, the plant design criteria are summarized in Table 17.4.

Table 17.4 Design Criteria for the Metallurgical Process

Item	Unit	Value	Source
General			
Operating days	d/year	360	Defined by METTS
Operating Hours per day – crushing	h/d	12	Defined by PPX
Operating hours per day – heap leach	h/d	24	Defined by METTS
Operating hours per day – Merrill-Crowe	h/d	24	Defined by PPX
Daily plant feed (nominal)	tpd	350	Defined by PPX
Daily plant feed (design)	tpd	500	Defined by PPX
Average material moisture	%	5.5	Defined by METTS
Utilization			
Crushing	%	84	Defined by METTS
Heap leach	%	100	Defined by METTS
Merrill-Crowe and refining	%	92	Defined by METTS
Water distribution system	%	95	Defined by METTS
Heap leach			
Gold extraction	%	80	Metallurgical test results
Silver extraction	%	33	Metallurgical test results
Operating time	year	8	Defined by PPX
Total mineral processed	tonnes	1,030,000	Defined by MDA
Total leach pad area	Ha	0.8	Defined by METTS
Merrill - Crowe			
Gold precipitation efficiency	%	98	Industrial Standard
Silver precipitation efficiency	%	98	Industrial Standard
Precipitate moisture	%	9 - 13	Industrial Standard

17.2 Process Area Working Schedule

The process equipment has been pre-selected for the conditions summarized in Table 17.5.

Table 17.5 Main Equipment Working Schedule

Component	Equipment Working Schedule	
Trucking Mine to Plant	Months per year	12
	Day per month	30
	Shift per day	2
	Hrs per shift	12
	Hours per day	24
Leaching Pad	Programmed hour per shift	12
	Hours per day	24
Mine – Plant Auxiliary Equipment	Month per year	12
	Days per month	30
	Shift per day	2

17.3 Crushing

Three stages of crushing are planned for the Callanquitas mineralized material prior to agglomeration and stacking on the heap-leach pad.

17.3.1 Primary Crushing

The ROM ore will be transported in trucks of 15m^3 with a net load of 20 tonnes, and this mineral will be discharged both in the coarse hopper and in the coarse stockpile depending on the availability of storage. The capacity of the ROM hopper will be 100 tonnes wet, and by means of a lower feeder the ore is discharged into a vibratory grizzly with 50mm opening (minimum). The oversize material will feed to a 21 inches x 32 inches jaw crusher and the undersize or screened material will fall to a conveyor belt No. 1. The jaw crusher has been selected to operate with a set opening set of 38 mm. The product of the jaw crusher will fall to conveyor belt No. 1.

Both the jaw crusher product and the grizzly undersize with 65% minus 25mm will go to a 5 feet x 16 feet vibrating screen of a single deck with a 19mm opening. The +19mm oversize returns, through conveyor belt N°2, to the 3 foot cone crusher. This unit works with a 19mm discharge opening and the crushed ore product of the conical crusher, whose granulometry is 60% minus 19mm, will unload to the conveyor belt N°1, joining with the product of the jaw crusher and undersize of the grizzly, to enter the vibrating screen again.

17.3.2 Fine Crushing

Discharge from the vibrating screen will be 100% less than19mm and it will enter a vibrating sieve N°2 of 9mm mesh. The oversize from vibratory sieve N°2 enters the vertical tertiary impact crusher ("VSI"), which discharges a product of 100% - 9.5 mm. This fine product falls to conveyor belt N°3 that joins with vibrating screen N°2 and both products go to the agglomeration system. The feed conveyor will include a conveyor belt weight meter for measuring material throughput.

17.3.3 Crushed-Ore Stockpile and Reclaiming

The crushed product from the crushing circuit will be stockpiled with a radial stacker. Two loading stations with hoppers and belt feeders will receive stockpiled material with the use of a front-end loader and discharge into a transfer reclaim conveyor that brings material over the conditioning section. The transfer conveyor will discharge into a trommel for mixing leach material with cement and lime.

17.3.4 Conditioning, Agglomeration and Transfer to Leach Pad

Lime will be added directly to the crushed product reclaim conveyor. The reclaim conveyor will discharge into a trommel for mixing material with cement and lime to complete the mineral conditioning for heap leaching. A 70-tonne lime silo with a reclaim device and screw feeder will allow metering the lime into the conveyor.

After the lime addition, the material will be passed through the agglomerating drum where it will be mixed with cement and water. The agglomerated ore will be stacked forming intermediate piles to allow for a curing period before transfer to the leach pad by trucks.

The material will be transferred to the heap area using two 30 tonne trucks. The leaching pads are to be formed using a front loader and/or excavator.

17.4 Design and Selection of Equipment

17.4.1 Primary, Secondary and Tertiary Crushers

The design of the primary and secondary crushing circuit was carried out based on a feed tonnage of 500 tpd, mineral moisture of 5.5% and 12 hours of work per day. This will allow incorporating additional mineral processing in the future.

Taking these factors into account, the designed hourly tonnage for the crushing plant would be 41.7tph. The design of the grizzly vibratory, primary jaw crusher, vibrating screen, conical secondary crusher and VSI was according to this tonnage.

17.4.1.1 Vibratory Grizzly

The ROM material is assumed to have a granulometry of feeding to the circuit of crushing of 100% minus 700mm and it is assumed that it has 40% at less than 50mm. In addition, it is known that the average humidity of the ROM material will be 5.5%, which can increase due to the presence of clays. Considering these characteristics, a vibratory grizzly with a minimum opening of 50mm will be considered.

17.4.1.2 Primary Crusher

With 60% of the ROM ore greater than 50mm as oversize from the vibratory grizzly, that is, 25tph design, a jaw crusher of 21 inches x 32 inches with a closed side setting (“CSS”) of 38.1mm is recommended. The average capacity would be 60tph according to the NORDBERG catalog, thus having an overproduction capacity of 240% of the treatment, giving the possibility of reducing the daily working hours or increasing the daily tonnage.

17.4.1.3 Secondary Screen

According to the arrangement of the circuit, the secondary crusher works in a closed circuit together with a secondary vibrating screen. The vibrating screen will have a screen opening of 19 mm. According to the material balance and considering the granulometry of entrance to the sieve, this would have a design mass rate of 90.2tph, for which a 5 foot x 16 foot vibrating screen will be adequate.

At a feed of 90.2tph and considering an efficiency of 90% in the classification, the minus 19 mm material would be produced at 41.7tph, and the material greater than 19mm (the circulating load) would be 48.6tph, thus having a closed circuit load of 54% that returns to the secondary crusher.

17.4.1.4 Secondary Crusher

Considering the feed rate to the secondary crusher as 48.6tph, a 0.914 meter (3-foot) standard conical crusher is recommended. This would have an average capacity of 64tph according to the NORDBERG catalog, thus having an overproduction capacity of 115%. The CSS can be closed to 13mm if a finer product is required.

17.4.1.5 Tertiary Screen

A 4 foot x 14 foot screen is recommended because there will be 41.7tph of product passing through the vibrating screen. This will supports average capacities of 60tph.

17.4.1.6 Tertiary Crusher

For the 25tph design feed rate to the tertiary crusher, a VSI is recommended, which would have an average capacity of 50tph according to METSO catalog. This would have an overproduction capacity of 200%.

17.4.1.7 Crushed Ore Agglomeration

The agglomerating unit, with overall dimensions of 1.8 meters diameter by 5.0 meters length, will receive material for mixing together with cement, at a rate of 3 to 5kg/t, lime, at the rate between 1 to 1.5kg/t, and fresh water.

17.4.1.8 Agglomerator Design

The product of the tertiary crusher will be 100% less than 9.5mm. This material will enter the agglomerating drum at a rate of 41.7tph. The design considers that the mineralized material enters with an average humidity of 5.5% and leaves the agglomerator with a humidity of approximately 13%. For the design of the agglomerator the following parameters were considered:

- Angle of repose of the mineral: 40°;
- Inclination of the drum: 5°;
- Percentage of critical drum speed: 30%; and
- L/D ratio: 3.0.

It was determined that the residence time of material in the agglomeration drum will be 38 seconds, considering a drum of 1.8 meters diameter x 5 meters long with a design capacity of 68tph.

17.4.1.9 Curing of Agglomerated Material

Curing of the agglomerated material will be done in the stockpile or during loading to the leach pad.

17.5 Heap Leaching - Pad Configuration and Operation

Mr. Burga considers that the next engineering phase should expand the design work for the heap leaching pads to incorporated details of the location and working strategy. The single, re-useable leach pad will consist of two sectors, each lined with a 90 cm overliner of clays and waterproof material, and between 1.5 to 2 mm of HDPE geomembrane to waterproof the soil. Stacking of agglomerated material will be done with a front loader or excavator on the two-story pad (the initial operation requires three lifts).

The design of the leach pad has been focused on the existing topographic characteristics in the project area and based on conventional leaching technology. The dimensions of the pad are based on the design criteria provided by METTS and complemented with the design recommendations and experience of other similar operations. These criteria are summarized in Table 17.6.

Table 17.6 Design Parameters for the Pad

Description	Unit	Value
Angle of repose of heap	°	35
Height of lift	m	5.1
Number of lifts		3 (Y1 and 2) 2 (Y3 to end)
Base of pad (L x W)	m x m	105 x 84

During the first two years, the first sector of the leach pads will have an area of approximately 0.8 Ha and a final height of 21 meters with an overall stacking slope of 1.90H:1.0V. The first sector will receive 126,000 tonnes per year and it will accumulate 252,000 tonnes of material. The total capacity, after three lifts, is estimated at 253,400 tonnes, which provides enough capacity for the first two years (Phase 1-Stage 1). After the first two years operation, the leached ore on sector 1 will be reclaimed for mine backfill at the rate of 200,000tpa.

The leach pad will be constructed to cover the full allocated area to form the first floor, with the leaching period estimated at 90 days, followed by five days of rinsing. The leach pad construction will be parallel with the leaching extraction, requiring special management of the operation.

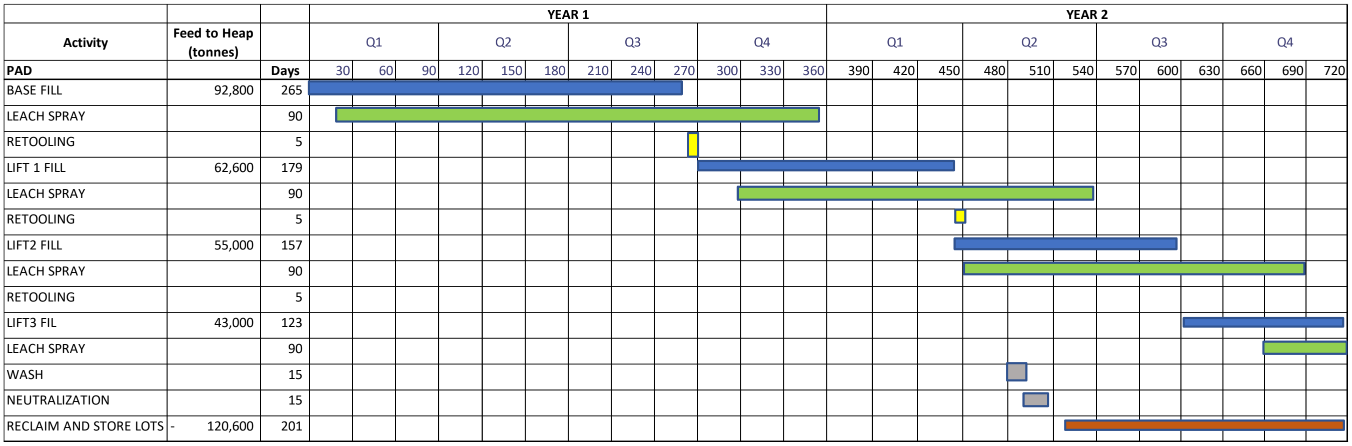
When the base of the first leach sector is complete, the surface will be prepared for the first lift followed by the second and third lifts. Prior to completion of stacking the first sector of the pad, the second sector should be operational to start stacking and leaching of agglomerated material.

At the start of the third year, the pad will be conditioned to receive 180,000 tonnes per year for leaching operation. In this phase, the operation is planned to allow reclaiming of 200,000 tonnes per year as mine backfill. Considering a 600tpd reclaiming rate, it will take 330 days to reclaim each sector of the pad.

The design concept assumes leaching of the ore will commence when enough of a heap is formed to create a uniform leaching area and maintain the flow of uniform pregnant solution. The heap height of 5.1 meters is considered adequate to maintain a high rate of percolation of the pregnant solution.

The cycle of stacking, leaching, rinsing and reclaiming sector 1 for years 1 and 2 is presented in Table 17.7. The material will be stacked in the base and three lifts to accommodate the two-year production.

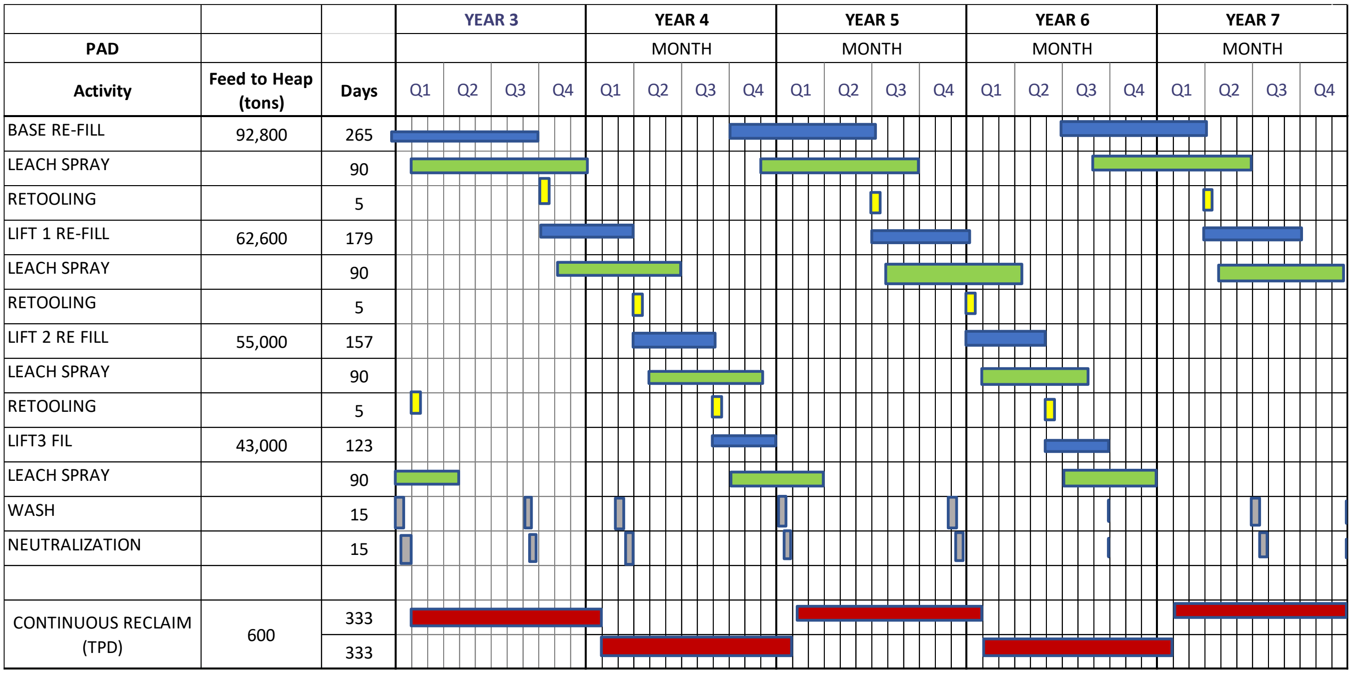
Table 17.7 Heap Leach Plan for Years 1 and 2



Activity	Feed to Heap (tonnes)	Days
PAD		Days
BASE FILL	92,800	265
LEACH SPRAY		90
RETOOLING		5
LIFT 1 FILL	62,600	179
LEACH SPRAY		90
RETOOLING		5
LIFT2 FILL	55,000	157
LEACH SPRAY		90
RETOOLING		5
LIFT3 FIL	43,000	123
LEACH SPRAY		90
WASH		15
NEUTRALIZATION		15
RECLAIM AND STORE LOTS	- 120,600	201

Operation of the leach pad for years 3 through the end of the mine life is presented in Table 17.8.

Table 17.8 Heap Leach Plan for Year 3 to End of Mine Life



PAD		
Activity	**Feed to Heap (tons)**	**Days**
BASE RE-FILL	92,800	265
LEACH SPRAY		90
RETOOLING		5
LIFT 1 RE-FILL	62,600	179
LEACH SPRAY		90
RETOOLING		5
LIFT 2 RE FILL	55,000	157
LEACH SPRAY		90
RETOOLING		5
LIFT3 FIL	43,000	123
LEACH SPRAY		90
WASH		15
NEUTRALIZATION		15
CONTINUOUS RECLAIM (TPD)	600	333
		333

During years 1 and 2, the leach solution will have a flow rate of 30 m^3/h. At the start of year 3, the flow rate will be adjusted to 43 m^3/h. The loading and recirculation of the leaching solution will be monitored to adjust the recovery rates.

After processing and stripping of gold and silver in the Merrill-Crowe facility, NaCN will be added to the barren solution to maintain the proper concentration. The barren solution will then be pumped to the barren solution tank to be transferred again to the heap irrigation system.

Lime will be added to the material in the stockpile and it can also be added during stacking to the pad. This must be done in such a way that the protective alkalinity greater than 10.5 pH is maintained inside the heap. The lime dosage system will be evaluated in the next stage of engineering where it will be decided if it is better to add as lime milk or in solid form.

The average humidity of the crushed ore should be on the order of 5.5%. The leached material is estimated to have a humidity of 8%. The effects of precipitation and evaporation will impact on the water balance in the leach pad. More details of the system will be given in the Detail Engineering stage.

17.5.1 Sand Separation

The heap-leach operation will include a sand separator pool with a capacity of 100m^3. It will have dimensions of 16.50 meters in length by 4.0 meters in width, and a depth of 1.50 meters. It will be lined with a 1.5 mm HDPE geomembrane on a low permeability floor of 300 mm thickness.

17.5.2 Pregnant Solution Ponds

The pregnant solution ponds 1 and 2 will have the following characteristics: a double lining system on a 300 mm floor layer of low permeability, and primary and secondary liners composed of 1.5 mm HDPE geomembrane. Pond 1 will have a capacity of 1,510m^3. Pond 2 will have a capacity of 1,500m^3.

Both pregnant solution ponds will have a leak detection system consisting of a geonet between the two geomembranes, designed to handle possible leaks in the primary geomembrane by directing it to a reservoir located in the south-east end of the pond. This requires that the bottom of the pond be designed with a minimum slope of 2% in the direction of the reservoir, where the leaked solution will be retained by solid pipes of HDPE SDR 17 of 300 mm, with holes of 5 mm spaced every 50 mm. These pipes will be connected to a pumping system and the leaked solution will be pumped and returned to the pregnant solution ponds. The lifting system consists of a solid HDPE SDR 17 pipe of 300 mm diameter and is located in a trench along the slope of the pond, with gravel filling and wrapping of 270 g/m^2 nonwoven geotextile. Inside this pipe there will be a submersible pump which will return the solution to the pond.

17.5.3 Emergency Pond

An emergency events pond will be located next to the pregnant solution pond 2 (Figure 17.5) and will have a capacity of approximately 4,000 m^3, to store the flows of emergency events from the leaching pad. The maximum operational elevation of the emergency events pond will be 3,548 meters. The total free edge shall be 1.0 meters, plus a safety berm of at least 0.5 meters around the edge of the pond. The pond has been designed considering an average slope of 2.0H:1.0V.

17.5.4 Solution Handling Pumps

Table 17.9 summarizes an estimate of the lengths and difference of dimensions that the solutions will travel for specifying the pumps to be used in the operation.

Table 17.9 Pregnant Solution, Spent and Contingency Pumps

Description	Length (m)	Differential H (m)	Max Flow (m^3/h)
Pregnant Solution Pump (to Merrill-Crowe plant)	20	1	150
Barren Solution Transfer Pump (to leaching pad)	120	9	150
Emergency Pond Portable Pump/contingency	-	60	100

17.6 Gold - Silver Recovery (Merrill Crowe) and Bullion Production

The pregnant leach solution ("PLS") will be collected through perforated drain tubes located at the bottom of the leach pad and flow into the PLS ponds. From there the PLS solution will be pumped to the Merrill-Crowe plant, wherein gold and silver will be precipitated from the solution, the precipitate will be filtered and dried, and then smelting to obtain gold-silver doré. The doré will be shipped off site for sale.

17.6.1 Clarification of Pregnant Solutions

The recovery process begins with the pressurized feeding of the PLS to the two clarifying filter presses. The process will be effective if total solution solids ("TSS") are reduced to values lower than 1 ppm. To obtain this level of solid removal, diatomaceous earth will be added to maintain the porosity of the cake in the clarifying filters. The clarifier filter cake will be manually discharged by means of a washing and cleaning sequence using fresh water or sweeping solution. The clarified PLS will be discharged to the clarified pregnant solution tank and from there it will be fed to the vacuum-packed tower. The Merrill-Crowe plant will receive 150 m^3/h of PLS as the design flow rate from this stage.

The vacuum-packed tower will have two vacuum pumps to reduce the dissolved oxygen content of the PLS to values less than 1 ppm. The purpose of reducing dissolved oxygen is so that it does not react with the reagents added to the tower outlet, especially with zinc powder, which is very susceptible to oxidation. The proposed vacuum tower could be 2.0 meters in diameter and 6.5 meters in height.

Zinc powder, lead nitrate and diatomaceous earth will be mixed with the clarified and de-aerated PLS via a feed pump, which will pump the mixture to the filter press system where the precipitate will be extracted. Three plate filter presses will be used. Two will be in operation and one will be in stand-by mode for the next cycle. Also, diatomaceous earth is used for the pre-layer of filters.

The precipitate of zinc, gold and silver will be discharged in a tray and will be leached with sulfuric acid to remove impurities before being smelted. A retort system will be used to remove the mercury present in the precipitate. The sweeping solution that is obtained from the precipitate filters will be recirculated to the sweep solution tank.

The precipitate or cake is estimated, according to the metallurgical balance, to have a composition of: 6.2% gold, 57.0% silver, and 36.8% zinc and other elements.

17.6.2 Merrill-Crowe Circuit Equipment

The list of the main equipment for the Merrill-Crowe plant is shown in Table 17.10. This list considers an estimated capacity to process a flow of up to 150 m^3/h.

Table 17.10 List of Main Merrill-Crowe Equipment

Item	Description	Quantity	Characteristics
1	Press clarifier filter	3	Ø 1500 mm. Horizontal, Pressure: 120 psi, carbon steel A-36, Filter area 90 m^2. # filters 24 sheets of polypropylene.
2	Vacuum tower	1	A-36 Ø = 1,200 x H = 6,500 mm, vacuum pressure of -22 "Hg. Upper and lower semi-elliptical cover. Manhole (2)
3	Volumetric dispenser zinc feeder	1	304 stainless steel end screw, flow rate: 0.4-4 liters/hour, flow = 5 kg/h, hopper with lid capacity 15 kg, variable speed 10-15 rpm, with signal 4- 20 mA, with 440-volt, 60 Hz motor, and electronic board.
4	Press precipitation filter model EP.	3	Pressure 100 psi, area 90 m^2, # chamber 35, cake thickness 30 mm, polypropylene plates 900 mm, standard recessed plates. Hydraulic closure
5	Tilting crucible furnace	1	Model A-300. Capacity 100 - 150 kg. Hydraulic turning
6	Slag storage	1	-

17.6.3 Compressed Air

The plant and instrumentation air will be supplied by a rotary screw type air compressor. The plant air will be stored in the main air accumulator tank. The plant air to operate the precipitate filters will be stored in a separate storage tank. A compressor and additional air accumulator tank will be located in the crushing plant to service the crusher. The instrumentation air for the plant will be stored in the accumulated instrumentation air and will be dried and filtered.

17.6.4 Cyanide Destruction

A SO_2-AIR cyanide destruction plant is planned to eliminate the NaCN from the excess solution collected in the emergency pond. The cyanide destruction circuit comprises mainly two 250 m^3 reactors, in parallel, in which SO_2 will be added probably in the form of sodium metabisulfite. Air will be injected to the reactors with a blower. If needed, some lime and copper sulfate could be added to the reactors.

17.6.5 Precipitate Wash System

The collected precipitate will contain metals such as copper, zinc, iron, and others. These impurities will need to be eliminated prior to smelting to produce the doré. Therefore, a precipitate washing system using concentrated sulfuric acid will be used to separate the impurities. The precipitate will be mixed with the sulfuric acid in a stainless-steel tank and then filtered, leaving the impurities in the solution as sulfates, and the gold and silver in the clean precipitate.

The cleaned precipitate will be discharged from the precipitate wash tank and filtered, separating the solution of sulphates with impurities and the gold and silver in the solids. Then the wet cake will be dried in the mercury recovery system and then will be mixed with flux and placed in cupels in the smelting furnace.

17.6.6 Mercury Recovery System

The mineralized material of the Callanquitas Este deposit contains mercury (3 ppm). This will require a mercury recovery system. To remove the mercury, the precipitate will be placed in trays of 30kg capacity and heated in the electric retort oven. The system will consist of a coated oven, two condensers and filters to trap the mercury, and a condensed mercury storage tank. The mercury will be transferred to iron bottles and shipped off site for disposal.

17.7 Process Reagents

Estimated reagent consumptions are based on metallurgical testing and the design criteria for the project. A list of required reagents and their calculated annual consumptions are presented in Table 17.11.

Table 17.11 Estimated Reagent Consumptions

Reagent	Consumption per Tonne Processed	Annual Consumption (kg)
Sodium Cyanide (NaCN)	0.18 (kg/t)	22,700
Lime	1.5 (kg/t)	189,000
Cement	4 (kg/t)	504,000
Diatomaceous earth	0.21 (kg/t)	26,500
Zinc powder	0.24 (kg/t)	30,200
Lead nitrate	1.5 (kg/t)	7,560
Sodium Metabisulfate	3.6 (g/t)	453
Anti fouling agent	10 (g/t)	1,260
Borax	0.01 (kg/t)	1,300
Sodium nitrate	0.02 (kg/t)	2,600
Sodium carbonate	0.02 (kg/t)	2,600
Diesel	0.85 (kg/t)	110,000

17.7.1 Sodium Cyanide (NaCN)

The NaCN mix and metering circuit will include two NaCN addition pumps, one operating and one for standby. This circuit will also include a cyanide transfer pump, a cyanide mix tank, a cyanide mix tank, dust containment box, a cyanide storage tank, cyanide bag hoist, and steel supports and grating for a monorail-type hoist for loading super sacks of cyanide briquettes into the cyanide mix tank. Cyanide will be delivered to the property in super sacks and stored in the fenced, locked, and illuminated reagent storage facility.

NaCN will be supplied in 1-tonne supersacks duly secured in a wooden container. The supersacks will be cut in a bag-breaking system and the cyanide pellets will fall into a preparation tank where fresh water (or the solution sweeps) will be used to dissolve the pellets. The NaCN solution will then be transferred to a cyanide storage tank. The cyanide solution will be mixed with a sweeping solution to obtain a 20% w/v solution. The cyanide solution will be dosed from the storage tank to the sweep tank after the Merrill-Crowe through a pressurized supply network. The cyanide mixing and storage area will have a container wall and be equipped with a vertical axis sump pump for easy cleaning. Spills will be directed to the sump pump and return to the preparation circuit.

17.7.2 Lime

This reagent will be used to control the pH in the agglomerating drum, heap-leach circuit, and cyanide destruction tank. The lime will be delivered in 1-tonne supersacks. The lime supersacks will be cut and stored in a lime silo and dispensed directly onto a conveyor belt. The lime delivery system will add lime to the crushed ore stream by a variable-speed feeder receiving instructions from the weighing scale on the final product conveyor belt from the crushing circuit. Lime delivery will be by bulk trucks and directly blown into the lime silo. The lime silo will be located near the crushing area.

17.7.3 Cement

Cement will be delivered in 25kg bags and stored in the cement storage facility. Cement will be added to the agglomerator vessel as needed.

17.7.4 Anti-scaling Agent

Anti-scaling agent will be delivered in 1,000-liter totes. Anti-scaling will be added to the barren and pregnant pump inlets via chemical addition pumps to mitigate pipe scale formation.

17.7.5 Zinc Powder

Zinc powder will be supplied in 50kg cylinders and loaded by hand into the zinc storage hopper. From there, the zinc will be dosed by a screw feeder to the PLS before the precipitate filters. Zinc powder will precipitate the gold and silver from the PLS. It is recommended that the zinc powder be of the smallest possible particle size. Smaller particle sizes have a greater surface area and therefore a better gold-silver precipitation efficiency.

Zinc powder is a material that oxidizes very easily. Therefore, it must be properly stored and properly closed to avoid oxidation and a decrease in the efficiency of gold-silver precipitation.

17.7.6 Lead Nitrate

Lead nitrate will be used to increase zinc activity and reduce the tendency to passivate layers of zinc hydroxide during precipitation. Lead tends to form a pseudo-alloy with gold (Pb-Au) on the surface of the zinc particle, causing the gold to be deposited preferentially over the lead due to the electromotive series, leaving more free surface on the zinc for the precipitation to continue.

Lead nitrate will be supplied in 25kg bags that will be manually loaded into the lead nitrate storage hopper. From there it is dosed by a screw type feeder into the PLS before the precipitate filters.

17.7.7 Sodium Metabisulfite (SMBS)

Discarded and/or surplus NaCN solutions will be treated with sodium metabisulfite ("SMBS") to reduce the levels of dissociable cyanide in weak acid. The SMBS will be supplied in 1-tonne bags, which will be placed on the bag cutter and discharged to the mixer tank. The SMBS powder generated during the opening of the bags will be extracted by means of an air extractor and discharged to the atmosphere. Fresh water will be added to the SMBS mixing tank to obtain a solution concentration of 20% w/v. The SMBS solution will then be pumped into the storage tank and through the cyanide destruction circuit via positive-displacement metering pumps.

The SMBS mixing and storage area will have a retaining wall and will be equipped with a vertical axis sump pump for easy cleaning. Spills will be directed to a sump pump and returned to the preparation process.

17.7.8 Copper Sulfate

Copper sulfate will be used as a catalyst in the destruction of cyanide from the INCO process. Copper sulfate will be supplied in 25kg bags. The bags will be carried manually to the bag cutter and unloaded into the preparation/storage tank. The copper sulphate will be mixed with fresh water until a concentration of 10% w/v is achieved. Copper sulfate will be dosed from the preparation/storage tank to the cyanide destruction tank.

17.7.9 Anti-fouling Agent

Anti-fouling agent prepared at 5% w/v will be supplied to the plant in iso-containers of 1.0 m^3 and pumped directly to the required dosing points by means of pneumatically operated diaphragm pumps.

17.7.10 Diatomaceous Earth (DE)

DE will be used for the pre-coating of clarifying filters and precipitate filters, and it also will be added to the filter feeding stream of the clarifier and filter feed ("body feed") to improve filtration rates and maintain the porosity of the filtered gold-silver-zinc precipitate cake. The DE will be supplied in 25kg bags. The pre-layer pulp will be prepared by mixing DE with barren solution in the pre-layer mixing tanks of the clarifying filter and precipitate filter up to a concentration of 0.5% solids. A "body feed" pulp will be prepared by mixing DE with a barren solution in a mixer tank to a concentration of 10% solids. The DE bags will be opened using bag cutters located at the top of each tank. The pulp will then be retained in the mixing tanks before distribution to the filters.

17.7.11 Melting Fluxes

The main fluxes required for the smelting of the precipitate cake to produce gold-silver doré will include sodium nitrate, sodium carbonate and borax. These reagents will be supplied in 25kg bags and mixed with the precipitate in the flux mixer as required before going into the smelting furnace.

17.8 Water Supply and Management

17.8.1 Fresh Water Supply

Fresh water will be supplied from the nearby approved river and stored in a water storage tank of 100m^3 for pumping to the processing plant (Figure 17.5). The water supply, water balance, and related infrastructure are summarized in Section 18.5.

17.8.2 Process Water and Heap Leaching

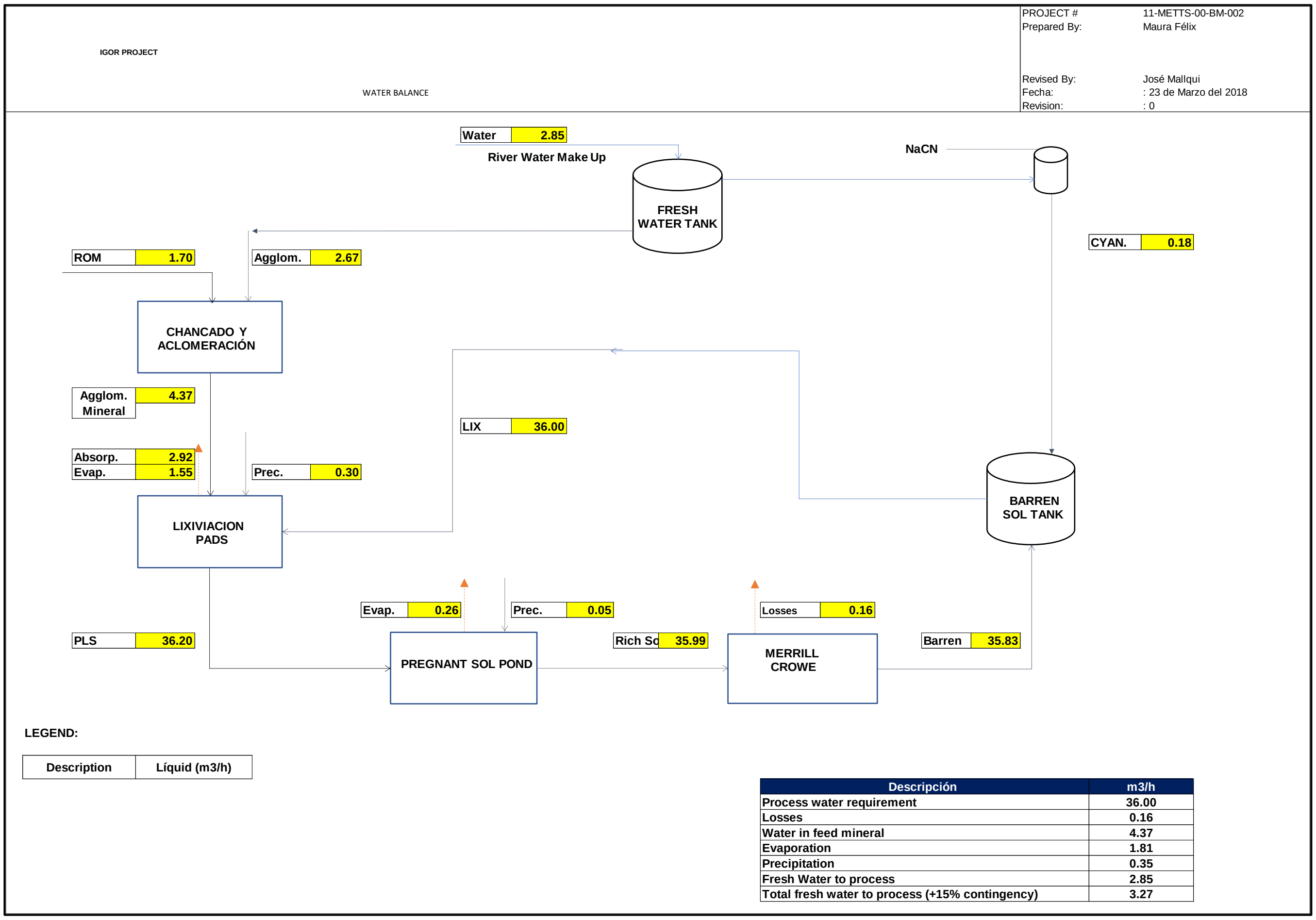
A water management flow sheet is illustrated in Figure 17.6. PPX plans to pump sufficient water from the river, and collect rain-water runoff, to meet the rate of 100m^3 per day estimated to operate the plant, crushers and heap-leach facilities.

The solution washed from the filter press in the Merrill Crowe system will have a cyanide concentration of 600 ppm and will be sent to the spent solution tank. There NaCN will be added bring the concentration to 1,500 ppm for return to the leach pad for irrigation. The barren solution can be used at various points throughout the process including water spray for the crushing area, preparation of diatomite, cyanide preparation, lime preparation, and clarifier filter washing. Antifouling agent will be added to the barren solution tank to avoid the formation of calcareous deposits due to the use of lime.

17.8.3 Water for Road Irrigation

For the LOM, a water truck with a capacity of 20m^3 will be used for the irrigation of access roads for dust suppression. In case of emergency, it can be used to provide water from the river to supply the process water system.

Figure 17.6 Process Water Management Illustration



Descripción	m3/h
Process water requirement	36.00
Losses	0.16
Water in feed mineral	4.37
Evaporation	1.81
Precipitation	0.35
Fresh Water to process	2.85
Total fresh water to process (+15% contingency)	3.27

18.0 PROJECT INFRASTRUCTURE (ITEM 18)

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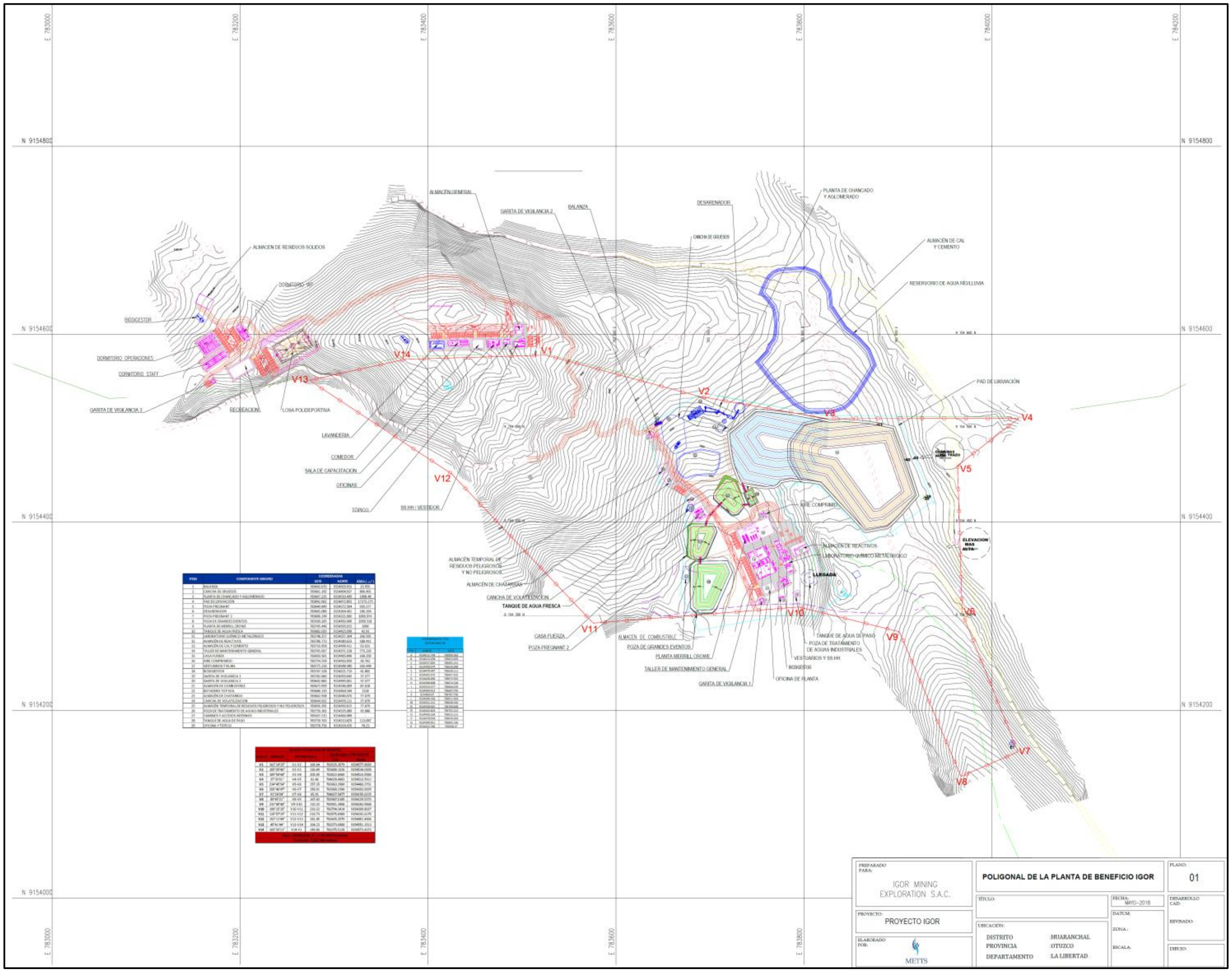
This section describes the infrastructure requirements for the process plant and auxiliary services operations. Mr. Burga reviewed the scope of proposed facilities and infrastructure for the project and considers that the outline presented satisfies technical requirements for this type of project. The infrastructure includes the supply of energy through generators, fuel storage, wastewater treatment, potable water supply, sewerage, main accesses and auxiliary plant, communication, security, plant buildings, management offices, dining rooms, workshops, and central warehouse.

18.1 Location and Components of the Infrastructure

A general plan of arrangement is shown in Figure 18.1.

Figure 18.1 General Plan of Arrangement, Igor Mine Project



18.2 Re-Useable Heap-Leach Pad

The re-useable leach pad will be approximately 105 meters by 84 meters as summarized in Table 18.1. The base of the pad will be lined with a 90 cm layer of over liner (clays and waterproof material) and between 1.5 to 2 mm of a layer of HDPE geomembrane to waterproof the soil.

Table 18.1 Heap-Leach Pad Design Parameters

Description	Unit	Value
Angle of repose	degrees	35
Height of lift per layer	meter	5.1
Number of layers		3 (Y1 and Y2) 2 (Y3 to end)
Base of pad (L x W)	meter x meter	105 x 84

The leach pad will have an area of approximately 0.88 Ha and a final height of 21 meters with an overall stacking slope of 1.90W:1.0H. Mr. Burga considers that the leaching pad area and the pad configuration requires confirmation during the next engineering phase to incorporate strategy for filling the heap leach and reclaiming leached material for mine backfill.

18.3 Power Supply and Distribution

The project considers the power supply using generators and a high-tension power line, installed by Hidrandina, the power provider, delivering to a sub-station that will feed the plant during the normal operation. The process power consumption is shown in Table 18.2.

Table 18.2 Process Power Consumption

Component	LOAD (KW)	Consumption (hr)	Voltage	Frequency	KW-hr
Merrill Crowe Plant	500	24	440	60	12,000
Refinery	50	24	440	60	1,200
Reagents	35	24	440	60	840
Crushing	421	8	440	60	3,368
Leach PAD	100	8	440	60	800
Ancillary equipment	123	24	440	60	2,952
Water pumps	150	12	440	60	1,800
Camp	150	16	440	60	2,400
TOTAL	1,529				

The energy distribution system for the project includes loads that can be grouped according to their importance. For the treatment plant circuit, a demand of 1,106 KW is estimated, distributed in the processes of crushing, leaching, Merrill-Crowe and other auxiliary processes. This energy circuit is estimated based on a treatment of 350 dry metric tons per day.

The circuit of minor loads (offices, dining rooms, etc.) is estimated at 238 KW. The safety factor includes general offices at 135 KW.

18.4 Roads

Access roads include the main access road to the facilities, the access road to the water reservoir, access to the processing plant and access to the dining room and offices. The main access road to the facilities connects the existing access road, which will be conveniently rehabilitated, and this access road connects the main components of the mining project. This access will be compacted gravel road construction.

Due to the need to interconnect the water dam with the other components, the construction of the road to the water reservoir has been proposed. This access will be compacted gravel road construction. Due to the need to interconnect the plant and complementary facilities with the other components, the construction of the process plant road is necessary to facilitate movement of personnel and vehicles. This access will be compacted gravel road construction.

18.5 Water Supply

The project water balance is summarized in Table 18.3.

Table 18.3 Water Balance Summary

Description	Value	Unit
Water consumption for agglomeration (nominal)	32	m^3/d
Leach Solution – Recirculated	30	m^3/h
Losses in Merrill-Crowe circuit (estimated)	0.03	m^3/h
Water consumption for cyanidation solution	0.21	m^3/h
Make water in barren pond (nominal)	1.44	m^3/h
Total fresh water to plant (+ 15%) (nominal)	82.2	m^3/d
Total fresh water consumption plant (+ 15%) (design)	***100***	***m^3/d***

18.5.1 Fresh Water Reservoir

This reservoir will have a capacity of 1,000m^3 to collect and store the water collected at river intake and pumped to the fresh water tank. This tank will also store rainwater that is collected by the drainage trench in the plant during the rainy season.

A river water pre-treatment system will receive the water pumped from the river. This system will include the reactor and the aluminum sulfate metering system and the hypochlorite metering system.

18.5.2 Potable Water

Potable water will be delivered to the plant storage area in large bottles. Bottled water dispensers will be located around the process area and the plant.

A potable water plant and a water purification plant will process water from the river and provide high quality water for the plant. A modular potable water plant with a capacity of 4,000 L/h is considered for permanent supply of potable water to the plant. This plant will include the multimedia filter, activated carbon filter, aluminum sulfate metering system, and chlorine metering for potable water production. Another, separate water purification system with 1,200 L/h capacity will include ultra-violet light (“UVL”) purification, and polishing filters to eliminate up to 99% of any bacteria, virus, protozoa, algae and yeast.

18.6 Sewage Treatment

Biodigesters will be installed for treatment of domestic used water (toilets, hand washing and showers of staff). It will have as a final disposal the irrigation of accesses and / or green areas. The water generated by these activities, after treatment with a biodigester, will comply with D.S. 003-2010-MINAM: MAXIMUM PERMISSIBLE LIMITS FOR THE EFFLUENTS OF DOMESTIC OR MUNICIPAL WASTEWATER TREATMENT PLANTS. In the washing hand water utilization, the contaminants that will be generated are the particulate material that could have adhered to the hands and soap residue. The bio-digester will be installed immediately after the authorization to start operations is given.

18.7 Sedimentation Pond for Industrial Water

This pond will receive effluent water from the Merrill-Crowe area, the casting area and floor drainage from the processing area. The settling of solids will be promoted by the use of flocculants. The facility includes a discharge pump to transfer clean water to the plant under the “zero effluent” concept.

18.8 Cyanide Water Treatment Plant

The amount of WAD cyanide dissolved in the solution, including weak and strong complexes, will be effectively controlled in the cyanide treatment plant to obtain a treated solution with cyanide below the permissible limits. The plant will be designed with a capacity of $20m^3/h$. The area defined for the installation is approximately $200m^2$.

The barren solutions entering the cyanide treatment plant will be taken at the exit of the barren solution pond, located at the Merrill-Crowe plant discharge, since it is at this point that the solution has the lowest concentrations of gold and silver, on the order of $0.027g/m^3$ of gold and $0.246g/m^3$ of silver, respectively. The cyanide laden stream will be treated with hydrogen peroxide and copper sulfate, and then any remaining solids will be precipitated in the sedimentation pond with the help of the flocculants.

The design also includes the removal of heavy metals. The heavy-metal removal process starts by adding the barren solution into the agitated reactor tanks where sodium bisulfite and coagulant will be added. The formed sediments will be separated from the liquid in a clarifying sedimentation cone. The solids will be transferred to a filtration package of geo-textile tubes for trapping the sludge, forming a permanent solid storage area.

Because of rainfall, it is possible that additional volumes of rainwater will be collected in the heap-leach pad and ponds, increasing the inventory of solution in the process facilities. This additional flow will report into the ponds designed for major and emergency events. In order to eliminate the possibility of spillage of solution, the design includes a discharge into the cyanide destruction plant. Sodium bisulfite will be used to eliminate the dissociable cyanide present in the weak acids stream exiting the plant.

18.9 Process Chemical and Metallurgical Laboratory

The project will have a chemical-metallurgical laboratory for the testing of solid samples (ore, precipitate, slag and doré), liquid samples from the process plant, and water and air samples for environmental monitoring purposes. This building will be of the fixed modular type and will have an approximate area of $300m^2$.

The reagent warehouse will be located inside the perimeter fence of the process plant and will have an approximate area of 190 m^2. This warehouse will store acids, fluxes, and will have areas for materials and tools.

The maintenance facilities building will be a steel structure with a metal roof located inside the perimeter fence of the process plant. It will have an area of approximately $620m^2$ and will include a workshop equipped with welding equipment, tools, compressed air and work benches.

18.10 Other Buildings

The plant administration building will be located near the maintenance building and will include plant administrative offices and a meeting room. It will be a modular building and will have an approximate area of $108m^2$, designed for approximately 12 to 15 people.

The process plant warehouse building, separate from the reagent warehouse, has been designed according to the storage and custody needs of materials and consumables of the process plant. The building will have an approximate area of $100m^2$ and will contain the warehouse offices. The plant warehouse will offer the storage of parts and spare parts of small and medium size, for the equipment of the process plant. There will be an area for the office of the logistics and maintenance manager. All the chemical compounds supplied to the project and which are exclusively used by the process plant will be previously checked in the process warehouse and then distributed to the reagent store.

18.11 Site Accommodations

The project camp buildings will be modular types including units for dormitory, laundry and services. There will be one 6.0 meter x 24.7 meter module to house 12 staff. There will be two 7.0 meter x18 meter modules for the general labor force. The modules will have common sanitary services. A VIP dormitory with space for up to 40 persons will be a modular unit of 850m^2. The units will be conditioned for the location with heating and hot water facilities.

18.12 Dining Room

There will be an external dining room located between the recreation and sports area, and the process plant. This dining room building will cover an approximate area of 150m^2 with a kitchen and dining area for approximately 40 people.

18.13 Fuel Storage

There will be a warehouse and tanks for the storage of fuels, which will be supplied with fuel from the trucks of the suppliers. The supply and replacement of the fuel will be completely done by the supplier chosen by the company. The same supplier will support the contract mining fuel facility which is to be provided by the contractor.

18.14 Communications

Off-site communications will consist of a satellite link system with internet-based communications, including telephones. On-site communication will be accomplished via two-way radios.

18.15 Volatilization Pool (Hydrocarbon Contaminated Soil)

A volatilization pool will be built to provide a place specifically designed to receive earth mixed with hydrocarbons. Contaminated soil will be aerated in a mixer favoring the evaporation of the hydrocarbon. The design includes an access zone, loading platform and the volatilization pool. The area of the pool will be 10 meters x 15 meters with a 0.5 meter high wall. A perimetrical channel 0.3 meters x 0.3 meters will prevent the ingress of runoff water into the pool. The floor will be waterproof and consist of 0.20 meters of clay material, and a geomembrane 1.5 mm in thickness. The area will have a galvanized roof for rain protection.

18.16 Solid Waste Storage

A solid waste management area will be located a few meters from the temporary storage for domestic waste. This facility will have an exterior area of 15 meters x 15 meters and an interior area of 225m^2. It will have a simple concrete floor 10cm thick, galvanized walls with reinforcements and / or wood frames with lumber supports, and the cover will be galvanized sheet-metal panels and translucent “fiberforte” sheets.

Compartments will be used to store different types of recyclable and non-recyclable waste. A government-approved solid waste management company will be retrieving and disposing of this waste on a monthly basis.

18.16.1 Hazardous Material Storage

There will be a storage area for waste materials that can contaminate the environment or that are hazardous to people, such as waste oils, batteries, and other chemicals which must be taken from the plant to places for further treatment. The perimeter will be surrounded by ACMA mesh fence and the floor waterproofed with high density polyethylene. The overall dimensions for this area are 9.0 meters by 6.0 meters for a total area of 54m^2.

18.16.2 Non-Hazardous Material Storage

A storage area will be constructed for different types of recyclable waste and not reusable material that will be held for future disposal. It will be located a few meters from the domestic sanitary landfill. This warehouse will have exterior dimensions of 15meters x 15 meters for an area of 225m^2. There will be a simple concrete floor, 10 cm thick; galvanized walls with wood frame reinforcement and lumber supports. The roof will be made from galvanized sheet-metal panels and translucent “fiberforte” sheets.

18.16.3 Intermediate Sanitary Waste Landfill

An intermediate sanitary landfill is planned that will allow temporary holding of waste produced from the camp, dining room and mine. It will be made up of trapezoidal trenches with a clay lining, a concrete leachate and a perimeter channel. The area of this landfill will be 1,000m^2 as considered in this PFS.

18.16.4 Temporary Storage for Domestic Waste

A temporary storage area for biodegradable waste products from the camp, dining room and mine will be located near the gatehouse 1. It has been designed for a useful life of 11 years. It will have an estimated area of 1,000m^2 for the PFS design.

18.16.5 Top Soil Storage Stockpile

A temporary top soil storage area will contain soil and related materials excavated during the construction of the heap-leach pad and process plant area. The top soil will be removed by machinery and will be deposited in the top soil deposit area, so that, once the operations are completed, it can be reused as a cover for the heaps.

18.17 Plant Security and Check Points

As a security measure, the processing plant will be completely enclosed by a wire mesh fence, equipped with barbed wire, which is intended to keep domestic and wild animals, and unauthorized personnel away from the area of operations. The security checkpoints will be located on the main access road and at the main entrance of the processing plant.

18.17.1 High Security Check Point

A maximum-security gate will be located inside the doré plant that will be fully insured with an inspection point for people entering and leaving.

The doré plant will include a safe, metal detector and surveillance cameras. The gold room would be designed to reduce personnel traffic during gold processing.

18.18 Truck Scale

A truck scale will be installed at the main entrance to control ore loads delivered to process plant.

18.19 Mining Facilities

With the exception of operations offices which will be located as part of the administration building, mining facilities will the responsibility of the contractor. This will include a workshop, laydown yard, and fuel tanks near the portal location. The fuel supplier for the plant will also provide fuel into the contractor’s fuel tanks.

Explosive magazines will also be provided by the contractor and will be located along the road between the plant and the portal.

19.0 MARKET STUDIES AND CONTRACTS (ITEM 19)

19.1 Metal Pricing

No market studies have been undertaken for this PFS. Gold-silver doré will be the commercial product from the Igor operation. Gold-silver doré from mining operations is readily sold on the global market to commercial smelters and refineries, and it is reasonable to assume that doré from the Igor project will also be salable.

To determine appropriate metal prices to be used for economic analysis and cutoff grades, Mr. Dyer has considered spot prices in the months prior to the Effective Date of this report and reviewed current metal prices used in recent NI 43-101 technical reports. The metal prices used for the economic evaluation in this PFS have been selected based on consensus prices as described by CIM guidelines in italics below, along with spot-price trends.

"Consensus Prices
The use of consensus prices obtained by collating the prices used by peers or as provided by industry observers, such as analysts for example, may be used in some cases. This methodology has the advantage of providing prices that are acceptable to a wide body of industry professionals (peers). The disadvantage is that sometimes these predictions can be consistently wrong for reasons beyond the QP's control. These prices are generally acceptable for most common commodities, major industrial minerals, and some minor minerals."

Metal prices used in several recently published technical reports that estimate reserves ranged from about $1,200 to $1,300 per ounce of gold, with the most current reports using $1,250 per ounce. Table 19.1 shows monthly gold prices compiled from Kitco.com. This shows that three-year and 12-month rolling-average gold prices have been on a fairly steady rise through July 2018, and then have decreased slightly. The monthly gold prices have stayed above $1,200 per ounce throughout the last 12 months with a high of $1,334 in April 2018 and a low of $1,198 in September 2018. The 12-month rolling average is $1,270 per ounce of gold and the 3-year rolling average is $1,253 per ounce. Based on gold prices recently used for technical reports, and the current trend in gold prices, a $1,250 gold price is used for the Igor economic model.

Table 19.1 Kitco Monthly Gold Prices (USD/oz Au – May 2017 to April 2018)

Month / Yr	Average	High	Low	3-Yr Avg	1-Yr Avg
Dec-17	$1,261.05	$1,291.00	$1,240.90	$1,221.86	$1,257.11
Jan-18	$1,331.67	$1,354.95	$1,311.00	$1,224.07	$1,268.70
Feb-18	$1,331.52	$1,352.45	$1,314.10	$1,226.97	$1,276.79
Mar-18	$1,324.66	$1,352.40	$1,307.75	$1,231.03	$1,284.59
Apr-18	$1,334.74	$1,351.45	$1,313.20	$1,234.83	$1,290.35
May-18	$1,303.03	$1,324.35	$1,288.30	$1,237.72	$1,295.19
Jun-18	$1,281.57	$1,302.75	$1,250.45	$1,240.50	$1,296.96
Jul-18	$1,238.53	$1,262.05	$1,217.55	$1,243.51	$1,297.16
Aug-18	$1,201.25	$1,219.00	$1,178.40	$1,245.84	$1,290.40
Sep-18	$1,198.47	$1,209.80	$1,185.40	$1,247.89	$1,280.69
Oct-18	$1,215.39	$1,235.95	$1,185.55	$1,249.45	$1,275.35
Nov-18	$1,220.95	$1,232.25	$1,202.10	$1,253.21	$1,270.24

Monthly average silver prices are presented in Table 19.2, which shows that the three-year and 12-month rolling-average silver price has been on a relatively steady decrease. The 12-month average at the end of April 2018 was $15.85 per ounce and the 3-year rolling average is $16.61 per ounce. Based on the silver price trend, a $16.50 per ounce silver price is used for the Igor economic model.

Table 19.2 Kitco Monthly Silver Prices (USD/oz Ag – May 2017 to April 2018)

Month / Yr	Average	High	Low	3-Yr Avg	1-Yr Avg
Dec-17	$ 16.16	$ 16.87	$ 15.71	$ 16.62	$ 17.05
Jan-18	$ 17.17	$ 17.52	$ 16.98	$ 16.62	$ 17.08
Feb-18	$ 16.66	$ 17.19	$ 16.35	$ 16.61	$ 16.98
Mar-18	$ 16.47	$ 16.65	$ 16.25	$ 16.62	$ 16.89
Apr-18	$ 16.61	$ 17.20	$ 16.28	$ 16.63	$ 16.77
May-18	$ 16.69	$ 17.13	$ 16.39	$ 16.63	$ 16.76
Jun-18	$ 16.52	$ 17.23	$ 16.03	$ 16.64	$ 16.73
Jul-18	$ 15.71	$ 16.21	$ 15.26	$ 16.66	$ 16.69
Aug-18	$ 15.01	$ 15.48	$ 14.60	$ 16.66	$ 16.53
Sep-18	$ 14.26	$ 14.53	$ 14.13	$ 16.65	$ 16.27
Oct-18	$ 14.59	$ 14.76	$ 14.33	$ 16.61	$ 16.07
Nov-18	$ 14.37	$ 14.82	$ 13.97	$ 16.61	$ 15.85

Mr. Dyer is not aware of other contractual obligations attributed to the project.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT (ITEM20)

PPX and its predecessors have performed exploration activities within the Igor property since 2005 and initiated bulk sampling and underground test mining of the Callanquitas mine from October of 2016 through 2018. Since 2016, the mineralized material has been shipped off site to a toll mill where it has been processed by grinding followed by cyanide tank-leaching and carbon-in-leach recovery. PPX is in the process of acquiring the environmental permit to construct and operate a cyanide heap-leach and Merrill-Crowe recovery system to produce gold and silver doré at a site within the Igor property (Figure 18.1).

The project under study includes exploration within the Igor property, operation of the Callanquitas underground mine, and a heap-leach process with Merrill-Crowe recovery plant. Various exploration campaigns were authorized by Peruvian government agencies under approved environmental management instruments and their modifications and updates since 2009.

The Callanquitas underground mine is located in the Igor 4 concession. Production from the underground will be transported by truck to the process area at variable rates of 115,000 tonnes per year to 161,000 tonnes per year over an estimated seven-year mine life. There it will be crushed, agglomerated and heap leached on a re-useable leach pad. The majority of the leached and agglomerated ore is planned to be trucked back to the underground and placed as back-fill.

20.1 Environmental Studies Conducted

Several environmental studies were undertaken or commissioned by PPX as follows:

- *Estudio de Impacto Ambiental semidetallado (EIA_sd), executed by BMB Consulting S.A. in 2008 (supported by ALS Laboratory Group and SGS Perú S.A.)*
- *First modification to EIA_sd (Primera Modificatoria del Estudio de Impacto Ambiental semidetallado), by BMB Consulting S.A. in 2011 (supported by ALS Laboratory Group and SGS Perú S.A.)*
- *First technical supporting report, referred as Primer Intrumento Técnico Sustentario (ITS), executed by Sienna Minerals S.A.C. in 2012-2013 (supported by Certimin S.A. in 2012-2013).*
- *Declaration of environmental impact referred to as Declaratoria de Impacto Ambiental, executed by Expediente Mineros S.A.C. in 2014-2015 (supported by Certimin S.A.)*
- *Declaration of environmental impact referred to as Declaración de Impacto Ambiental, executed by Asesores y Consultores Mineros S.A. in 2017-2018 (supported by Certimin S.A.)*
- *Estudio de Impacto Ambiental semidetallado (EIA_sd), executed by Asesores y Consultores Mineros S.A. in 2017-2018 (supported by Certimin S.A.)*
- *Certifícate of non existence of archeological remains referred to as Certificado de Inexistencias de Restos Arqueologicos (CIRA), executed by the Archeologist Moises Alejandro Rivero Iparraguirre in 2012-2013-2014 -2015.*

- *Certifícate of non existence of archeological remains referred to as Certificado de Inexistencias de Restos Arqueologicos (CIRA), executed by the Archeologist Jose Carcelen Silva in 2018.*

The following conclusions are based on the above studies:

The project area includes no Natural Protected Areas. The closest area is located 11 km away from the project and is known as the Natural Protected Area of Sunchubamba; hunting is primarily forbidden.

Water: Surface water is scarce. No permanent water flows were detected in the project area. Surface water flows are associated with the rainy season of November through March. The location of groundwater aquifers is below 3,100 masl.

In the sediments of the Tres Chorros, Cortaderas and Callanquitas streams, the metal content exceeds the limits according to the Interim Sediment Quality Guideline ("ISQG"), which provides the concentrations under which there is no adverse biological effect. There are no sediment quality standards; hence the ISQG is used as guide to identify the presence of adverse biological effect.

Air: Analyses of air quality have been carried out in the project area. Sulfur dioxide, nitrogen dioxide, particulate material less than 10 microns ("PM10"), particulate material less than 2.5 microns ("PM2.5"), carbon monoxide, lead, and hydrogen sulfide are meeting the standards of ECA Aire (DS Nº 003-2017- MINAM). Concentrations of PM2.5, PM10 and lead in the mine and the Igor exploration area are below the National Air Quality Standards of DS Nº 003-2017-MINAM, and the arsenic parameter is below the comparative regulations of the RM 315-96-EM / VMM. According to the 2017 monitoring, there is a concentration of breathable material ("PM").

Noise: In compliance with the current environmental regulations of Peru, PPX has conducted a study of environmental noise. It can be concluded that the noise levels for the industrial zone of the project area did not exceed the ambient environmental quality standards ("ECAs") for noise established in the DS 085-2003-PCM.

Fauna and Flora: Fauna and flora studies were conducted over the Igor project area. Endangered species according to DS N ° 043-2006-AG and international standards such as the Red List of Endangered Species of the IUCN and CITES (Convention on International Trade in Endangered Species of Wild Fauna and Flora) have not been reported.

Archeology: A survey was conducted and the existence of archeological remains were not found, except for proximity in the reservoir area and in the beneficiation plant area. The archaeological evidence in the reservoir includes the following: archaeological site "Serripe", with isolated archaeological elements called Muro I, Muro II and Muro III; and archaeological site Pampa de Igor. The plant area is under evaluation.

20.2 Mine and Process Waste, Tailings and Water Monitoring, Management and Disposal

The Callanquitas ore will be trucked 4.4 kilometers on an existing public road to the crushing and heap-leach process area. The width and gradient are suitable for the use of 20 tonne tippers. This road is in the area of the hamlets of Callanquitas and Igor. PPX will improve and maintain the road in good condition.

Waste material from underground development is planned to be used, in its entirety, as backfill in the Callanquitas mine. After heap leaching, the majority of ore from the Callanquitas underground workings will be trucked from the leach pad back to the mine and placed as backfill. It is estimated that approximately 430,000 tonnes of leached material will remain on the leach pad at the end of the LOM. Plans for mine closure, reclamation, and post-closure monitoring are summarized in 20.5.

Hazardous and non-hazardous industrial waste generated during the construction and operation of the project will initially be placed in bags inside cylinders with airtight lids. These will be delivered to a solid waste operating company ("OC-SW") duly authorized by MINAM for transportation and final disposal.

Water Management

Within the Igor project area, rainwater is managed with hydraulic infrastructure such as ditches, run-off drains and run-off ditches. These are implemented in access roads and drilling platforms. The water to be used in diamond drilling is minimal, since it is recirculated. The approval of the National Water Authority (ANA) has been obtained.

Underground water for the proposed mine is scarce during the rainy season and non-existent in dry seasons. Project water will be extracted from continuous sources, such as the Tres Chorros stream and the Huaranchal stream by means of pumps. This water use will be done in coordination with the inhabitants of the area because this water is also used for irrigating crops. Tres Chorros water will continue to be used for human consumption through purifiers, in order to increase the mining operation.

The Igor process plant will have a zero discharge philosophy. A closed circuit will reduce the need for makeup process water. There will be a water control process to manage contact of the water with the leach pad. Collectors and channels will be implemented to separate the contact and non-contact water. The river water will be pumped predominately in the wet season, while in the dry season the water will be drawn from the fresh water reservoir.

20.3 Permitting Requirements and Status

The Igor project has been granted all environmental and other permits required to conduct exploration work in the area; these include:

- Semidetailed Environmental Impact Assessment ("EIA-sd") effective through the end of year 2019, regulated by the General Directorate of Environmental and Mining Affairs;
- Water Use Permit for exploration activities, regulated by the National Water Authority;
- Certificate of Identification of Archaeological Remains ("CIRA"), regulated by the Ministry of Culture;
- Wastewater treatment certificate (issued by DIGESA);

- Approval to the modification of the EIA-sd for the Igor exploration work; and
- Approval of the ITS for the modification of the EIA-sd of the Igor exploration project.

Compliance with these permits is supervised by the Organismo de Evaluación y Fiscalización Ambiente ("OEFA").

20.3.1 Environmental Impact Statement ("DIA") for Exploitation

The project was granted environmental and other permits required for the exploitation of the Callanquitas mine as follows:

- Approval of the Environmental Impact Statement ("DIA");
- Approval of the CIRA;
- Approval of the Mining Plan and startup of activities;
- Approval of the Certificate of Mining Operation ("COM");
- Approval of the Permit for the Use of Explosives; and
- Approval of the license for the use of water for an industrial exploitation activity.

The DIA expires in 2020, but can be modified so that it may be extended for 5 more years. Compliance with these permits is supervised by GREMH-LL (Gobierno Regional de Energía, Minería y Hidrocarburos de La Libertad), the OEFA, and the Dirección General de Asuntos Ambientales Mineros ("DGAAM").

20.3.2 IGOR Beneficiation Plant EIA-sd

The Environmental File was submitted to the Ministry on Energy and Mining and has been approved. PPX held a public hearing and has received approval of the process plant. The archaeological evidence in areas close to the plant are rock walls forming rectangular and circular enclosures that will be monitored under the environmental and archaeological permits, including the approved plant site CIRA.

20.4 Social and Community Impacts

The Callanquitas and Igor hamlets are the two population centers within the project area. These have a constant population throughout the year of approximately 150 persons. Igor and Callanquitas are located 700 meters to the southeast and 1500 meters to the west, respectively, of the exploration area. The population is mainly engaged in agriculture, where the water used for this purpose is from the sources of the Huaranchal and Tres Chorros streams. The second occupation is livestock. Informal artisanal mining in the area is the third economic activity, with 17% of people engaged in this activity.

20.4.1 Social Sensitivity

The local population has adapted to the presence of PPX and the acceptance is partial. On the one hand they see it as an entity of development and opportunity for a better quality of life, due to the absence of the state in these population centers. There is also an opposition group with a negative general impression of old mining. With this group, PPX will have to work to achieve the possibility of long-term coexistence between mining and agriculture.

Electric power services exist, but not water and sewerage services. Water is provided by points of capture of groundwater, springs and blind silos.

20.4.2 Direct Social Area of influence

The area of direct social influence corresponds to the area where the general impacts of the project are direct and of greater intensity. This area is identified as Igor's hamlet.

Area of Indirect Social Influence

Huaranchal village center and the Pampa Grande, La Esperanza, La Quida and Tres Piedras hamlets have been considered areas of indirect social influence because they are populations close to the area of direct social influence of the project.

PPX has established a direct information policy with the inhabitants of the area, including town meetings, in anticipation of any permit extension to be granted for the Igor exploration and Mina Callanquitas project. If granted, the inhabitants from the Igor and Callanquitas areas shall be notified accordingly and consulted. PPX is focused on maintaining a harmonious atmosphere with the area inhabitants throughout the activities scheduled by the company.

During the life of the project, it is recommended to continue involving environmental authorities and neighboring populations to strengthen the relationship and facilitate communication regarding environmental impacts. It is essential to maintain good relations with neighboring communities to improve communications and facilitate environmental permits. It should be noted that PPX has an active community relations program and has established good relationships with local communities either directly or indirectly.

The mining operation will not cause an alteration in the lifestyle or the customs of the inhabitants that surround it; no cultural or anthropological changes are foreseen in the human groups indicated above. In turn, the project will generate jobs for the local population and thus improve their quality of life. Furthermore, in compliance with national legislation (Ministerial Resolution 304-2008-MEM/DM, article No. 11), the company undertakes the commitment to respect the social and cultural environment and to cooperate with the sustainable development of the communities located within the area of influence.

20.5 Mine Closure and Reclamation

The mine closure plan will focus on security, stabilization of land surfaces, subsequent use of facilities, structures and protection of the environment. The reforestation is considered ideal and favors the surrounding environment. A total of $3.8 million dollars is estimated by PPX for the cost of reclamation.

In Peru, there are clear and precise rules on the closure of mining facilities, which may include progressive closure and final closure. A summary of the objectives and closure activities are as follows:

- Ensure that the remaining facilities do not affect human health or degrade the environment;
- Ensure the maintenance of physical stability and that the areas affected by mining activities are in stable conditions at project closure;
- Ensure the maintenance of stability associated with long-term chemicals, in order to reduce the effects on biological diversity and avoid endangering public health and safety; and
- Ensure that environmental components, both surface and underground, are not affected as a result of closure.

To achieve the above-mentioned objectives, the following main activities are planned:

- The mine workings will be finally closed using mine waste, ensuring their physical and chemical stability. This will prevent third parties, informal miners, and animals access inside the mine and avoid accidents. On the adit plug, covers will be placed by adapting it to the profile of the contiguous natural land form, taking into consideration a slope of at most 1.5H: 1.0V, to allow the revegetation and water runoff and avoid stagnation of water;
- Civil infrastructures will be dismantled or demolished and removed from the mine site. Slopes will be regraded to provide drainage. Run-off and side ditches, as well as ditches at the foot of the slope will be maintained in order to divert runoff water;
- Recovery and closure activities will include the removal of all modules, power lines, pipelines and process components, securing rock and waste storage facilities, ensuring that the leach pad is chemically and structurally stabilized, and returning the area to the previous use of the land. As far as possible, the recovery will be carried out at the same time as the operations; and
- The leach pad will be modified by re-profiling and contouring of the surface. The remaining 430,000 tonnes of leached material will be covered with topsoil and planted with similar vegetation.

The mine closure plan is based on the environmental guide for the closure and abandonment of mines issued by the General Bureau of Environmental Affairs of the Ministry of Energy and Mines. This is in accordance with Peru's Law 28090, and the Regulations for the Closure of Mines – Executive Order (DS) 033-2005-EM. After the closure, it will be necessary to continue with the monitoring of all environmental and physical variables, in order to verify the proper performance of the plan, and if a contingent event should occur, adopt the necessary corrective measures. This monitoring will be the responsibility of PPX for five years.

The post-closure process must be carried out according to the approved schedules in the environmental studies of each environmental instrument:

- EIA-sd of the Igor exploration project: one year;
- EIS of the Callanquitas mine exploitation project: two years; and
- EIA-sd of the Igor process plant: 5 years

21.0 CAPITAL AND OPERATING COSTS (ITEM 21)

Mr. Dyer has compiled the capital and operating costs for Section 21. The author is responsible for the mining costs, and portions of the infrastructure costs. Mr. Ernie Burga is responsible for processing, infrastructure, and general and administrative ("G&A") costs.

Table 21.1 summarizes the project capital costs. Total initial capital is estimated to be $7.1 million. Sustaining costs of $4.6 million have been estimated, which include a refund of the cash contribution toward surety bonding. The total LOM capital cost estimate is $11.7 million. Note that negative quantities of capital in Table 21.1 reflect salvage of equipment at the end of the mine life.

Table 21.1 Capital Cost Summary (K USD)

	Units	Initial	Sustaining	Total
Plant & Infrastucture Direct Costs	K USD	$ 2,858	$ -	$ 2,858
Plant & Infrastucture Indirect Costs	K USD	$ 357	$ -	$ 357
Contingency - Plant & Infrastucture	K USD	$ 428	$ -	$ 428
Capitalized Development	K USD	$ 629	$ 5,416	$ 6,045
Pre-Production Mining	K USD	$ 382	$ -	$ 382
Other Mining Capital	K USD	$ 100	$ 90	$ 190
Contractor Adjustments	K USD	$ 2,374	$ -	$ 2,374
Salvage	K USD	$ -	$ (908)	$ (908)
Total Capital Cost	K USD	$ 7,128	$ 4,599	$ 11,726

Table 21.2 shows the estimated operating costs for the LOM. Total operating costs are $77.3 million, or $632 per ounce of gold produced over the life of mine.

Table 21.2 Operating Cost Summary (USD)

	Life-of-Mine Cost K USD	Cost per Tonne Processed	Cost per Oz AuEq
Expensed Mine Development	$ 2,520	$ 2.45	$ 21
UG Mining Costs	$ 50,846	$ 49.38	$ 416
Process Costs	$ 17,640	$ 17.13	$ 144
G&A Costs	$ 2,493	$ 2.42	$ 20
Reclamation Costs	$ 3,804	$ 3.69	$ 31
Total Operating Costs	$ 77,303	$ 75.08	$ 632

21.1 Capital Costs

The following sub-sections describe the capital-cost estimates for mining, process, and infrastructure.

21.1.1 Mining Capital

The mining capital is minimized due to the use of a mining contractor that provides the required mining equipment. This eliminates the need for equipment capital. The mining capital estimate includes capitalized development, pre-production mining, and mobilization/demobilization of the mining contractor. The annual mining capital estimate is shown in Table 21.3 with the total capital estimated to be $6,617,000 for the LOM. Estimate includes $6,045,00 for capitalized development, $382,000 for pre-production mining, and $190,000 for mobilization and demobilization.

Table 21.3 Annual Mining Capital

	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Capitalized Development	K USD	629	1,116	1,303	1,670	1,287	39	-	-	-	6,045
Pre-Production Mining	K USD	382	-	-	-	-	-	-	-	-	382
Mobilization/Demobilizaton	K USD	100	-	-	-	-	-	-	-	90	190
Total Mining Capital	K USD	$ 1,111	$ 1,116	$ 1,303	$ 1,670	$ 1,287	$ 39	$ -	$ -	$ 90	6,617

Capitalized development has been included based on contract mining rates for waste during development. This includes all of the main development with the exception of sub-level access which is included as expensed development (see Section 0). The unit costs assumed are shown in Table 21.4.

Table 21.4 Contractor Development Unit Cost

	Unit Cost	Units
Waste Development	$ 18.40	$/t Waste
Other Direct Costs	$ 8.20	$/t Waste
Indirect Costs	$ 5.40	$/t Waste
Explosives	$ 8.33	$/t Waste
Power	$ 1.23	$/t Waste
Total - Waste	$ 41.56	$/t Waste

The annual capitalized development cost estimate is shown in Table 21.5.

Table 21.5 Annual Capitalized Development

	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Total
Waste Development	K USD	$ 279	$ 494	$ 577	$ 739	$ 570	$ 17	$ -	$ 2,676
Other Direct Costs	K USD	$ 124	$ 220	$ 257	$ 329	$ 254	$ 8	$ -	$ 1,193
Indirect Costs	K USD	$ 82	$ 145	$ 169	$ 217	$ 167	$ 5	$ -	$ 785
Explosives	K USD	$ 126	$ 224	$ 261	$ 335	$ 258	$ 8	$ -	$ 1,212
Power	K USD	$ 19	$ 33	$ 39	$ 49	$ 38	$ 1	$ -	$ 179
Total Capitalized Development	K USD	$ 629	$ 1,116	$ 1,303	$ 1,670	$ 1,287	$ 39	$ -	$ 6,045

In addition to the items shown in Table 21.3, an additional contractor adjustment cost has been included in the amount of $2,374,000. This amount is included due to on-going negotiations with current contractor operations on site. PPX may or may not be obligated for this amount, but it is included as a worst-case position.

21.1.2 Process and Infrastructure Capital

The estimated total processing capital cost is $4.7 million as summarized in Table 21.6. This includes direct and indirect costs, and 10% contingency. The current capital cost for the processing plant is estimated at $3.64 million, including contingencies, and reflects the $1.07 million expended to date.

Capital estimates were based on pricing from vendors for main equipment. Installation costs were defined using a factor of 11% of the equipment cost that is suitable for installation of modular packages and low labor cost in the project site. Cost for Civil/Structural, piping, electrical and instrumentation were allocated as percentage of the equipment cost.

Table 21.6 Process Capital Cost Summary

Processing Direct Costs	**Original Estimate**	**Current Estimate**
Crushing and Grinding	$871,600	$ 654,000
Gold Recovery Plant	$577,400	$ 434,000
De-Oxygenation	$258,200	$ 194,000
Zinc Precipitation	$399,800	$ 300,000
Cyanide Addition	$217,200	$ 163,000
Smelting	$277,900	$ 209,000
Heap Leach Pad & Ponds	$281,600	$ 211,000
On-Site Infrastructure	$380,700	$ 286,000
Off-Site Infrastructure	$541,400	$ 407,000
Total Directs	$3,805,800	$ 2,858,000
Processing Indirect Costs		
EPCM	$114,200	$ 86,000
Contractor's Fees	$76,100	$ 57,000
Owners Costs	$114,200	$ 86,000
Transportation	$114,200	$ 86,000
First Fills	$57,100	$ 43,000
Total Indirects	$475,800	$ 358,000
Total Directs & Indirects	$4,281,600	$ 3,216,000
Contingency	$428,200	$ 428,200
Total Processing Capital	**$4,709,800**	**$ 3,644,200**

21.2 Operating Costs

Mr. Dyer compiled the operating cost estimate as summarized in Table 21.2. Mr. Dyer is responsible for the mine operating costs. While Mr. Ernie Burga is responsible for process and G&A operating costs, Mr. Dyer did compile the yearly costs based on fixed and variable costs, and has contributed to those sub-sections below.

21.2.1 Mine Operating Costs

Mine operating costs were estimated based on various contract mining estimates including the current contractor and updated quotations from other contractors. Costs were provided based on unit cost per tonne for mining of both ore and waste. The unit costs applied are shown in Table 21.7. The major consumables within the mining costs are electricity and fuel which have used $0.21 per kWh and $0.70 per liter respectively.

Table 21.7 Mine Operating Unit Costs

	Unit Cost	Units
Production Stoping	$ 27.80	$/t Ore
Waste Backfill	$ 2.10	$/t Ore
Other Direct Costs	$ 5.00	$/t Ore
Indirect Costs	$ 7.70	$/t Ore
Explosives	$ 5.55	$/t Ore
Power	$ 1.23	$/t Ore
Total - Ore	$ 49.38	$/t Ore
Expensed Dev	$ 9.56	$/t Ore
Cost for Cutoff	$ 58.94	$/t Ore
Waste Development	$ 18.40	$/t Waste
Other Direct Costs	$ 8.20	$/t Waste
Indirect Costs	$ 5.40	$/t Waste
Explosives	$ 8.33	$/t Waste
Power	$ 1.23	$/t Waste
Total - Waste	$ 41.56	$/t Waste

The yearly mine operating cost estimate is shown in Table 21.8 This shows the expensed and capital development tonnages. Capitalized development includes shafts, raises, and the main decline. Expensed development was used for operating costs and includes level development and any cross-cuts needed to access the ore on a level.

Table 21.8 Yearly Mine Operating Costs

Material Processed	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Ore Drives	K Tonnes	0	29	34	22	30	24	16	5	-	161
Ore Stoping	K Tonnes	-	86	119	135	119	137	127	146	-	868
Total Ore	K Tonnes	0	115	153	157	149	161	143	151	-	1,030
Expensed Development	K Tonnes	7	15	12	14	13	6	-	-	-	67
Capitalized Development	K Tonnes	15	27	31	40	31	1	-	-	-	145
Total Mined	K Tonnes	7	130	165	171	162	167	143	151	-	1,097
Backfill	K Tonnes	-	57	113	105	102	149	104	160	-	792
Production Mining Cost											
Production Stoping	K USD	$ 13	$ 3,204	$ 4,258	$ 4,356	$ 4,140	$ 4,477	$ 3,978	$ 4,197	$ -	$ 28,623
Waste Backfill	K USD	$ -	$ 121	$ 238	$ 221	$ 215	$ 313	$ 219	$ 337	$ -	$ 1,663
Other Direct Costs	K USD	$ 2	$ 576	$ 766	$ 784	$ 745	$ 805	$ 715	$ 755	$ -	$ 5,148
Indirect Costs	K USD	$ 4	$ 887	$ 1,179	$ 1,207	$ 1,147	$ 1,240	$ 1,102	$ 1,162	$ -	$ 7,928
Explosives	K USD	$ 3	$ 640	$ 850	$ 870	$ 827	$ 894	$ 794	$ 838	$ -	$ 5,714
Power	K USD	$ 1	$ 142	$ 188	$ 193	$ 183	$ 198	$ 176	$ 186	$ -	$ 1,266
Total - Ore	K USD	$ 22	$ 5,569	$ 7,479	$ 7,630	$ 7,256	$ 7,928	$ 6,985	$ 7,475	$ -	$ 50,343
Mine General Costs											
Personnel	K USD	$ 12	$ 48	$ 48	$ 48	$ 48	$ 48	$ 48	$ 48	$ -	$ 348
Supplies and Service	K USD	$ 7	$ 27	$ 27	$ 27	$ 27	$ 27	$ 27	$ 27	$ -	$ 196
Total General Services	K USD	$ 19	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ 75	$ -	$ 544
Total Mining Cost	K USD	$ 41	$ 5,644	$ 7,554	$ 7,705	$ 7,331	$ 8,003	$ 7,060	$ 7,550	$ -	$ 50,887
Mining Costs per tonne											
Ore Mining	$/t Ore	$ 47.28	$ 48.33	$ 48.83	$ 48.69	$ 48.72	$ 49.23	$ 48.81	$ 49.51	$ -	$ 48.90
Mine General Services	$/t Ore	$ 40.10	$ 0.65	$ 0.49	$ 0.48	$ 0.50	$ 0.47	$ 0.52	$ 0.50	$ -	$ 0.53
Total Ore Mining Cost	$/t Ore	$ 87.38	$ 48.98	$ 49.32	$ 49.17	$ 49.23	$ 49.69	$ 49.34	$ 50.01	$ -	$ 49.42
Expensed Development Mining Costs											
Waste Development	K USD	$ 125	$ 278	$ 218	$ 267	$ 243	$ 110	$ -	$ -	$ -	$ 1,241
Other Direct Costs	K USD	$ 56	$ 124	$ 97	$ 119	$ 108	$ 49	$ -	$ -	$ -	$ 553
Indirect Costs	K USD	$ 37	$ 82	$ 64	$ 78	$ 71	$ 32	$ -	$ -	$ -	$ 364
Explosives	K USD	$ 57	$ 126	$ 99	$ 121	$ 110	$ 50	$ -	$ -	$ -	$ 562
Power	K USD	$ 8	$ 19	$ 15	$ 18	$ 16	$ 7	$ -	$ -	$ -	$ 83
Total - Waste	K USD	$ 282	$ 628	$ 492	$ 602	$ 548	$ 249	$ -	$ -	$ -	$ 2,802
Expensed Development	$/t Ore	$ 603.89	$ 5.45	$ 3.21	$ 3.84	$ 3.68	$ 1.55	$ -	$ -	$ -	$ 2.72

21.2.2 Process Operating Costs

The production schedule assumes processing of 350 TPD during the initial two years of mining and then ramping up to 500 TPD. The parameters for the processing costs were split up into fixed and variable costs so that they could be applied by period based on the tonnage mined or the total cost per month as required. Table 21.9 shows the operating costs during the first two years of mining and Table 21.10 shows the process operating costs used throughout the remaining life of mine schedule.

Table 21.9 350 TPD Process Operating Cost

		OPEX SPLIT	
Item	Description	Fix cost	Variable cost
Process Plant Cost			
1	Staff involved in the process	$ 5.03	
2	Reagents and consumables		$ 4.76
3	Electric power		$ 6.83
4	Supply and auxiliary services		$ 0.40
Sub Total Operating Cost		$ 5.03	$ 11.99

Table 21.10   500 TPD Process Operating Cost

		OPEX SPLIT	
Item	Description	Fix cost	Variable cost
Process Plant Cost			
1	Staff involved in the process	$ 3.52	
2	Reagents and consumables		$ 6.79
3	Electric power		$ 5.72
4	Supply and auxiliary services		$ 0.57
Sub Total Operating Cost		$ 3.52	$ 13.08

Mr. Dyer applied the fixed and variable costs shown in Table 21.9 and Table 21.10 to produce the yearly operating cost that is used in the cash-flow model. Table 21.11 shows the yearly cost estimate.

Table 21.11 Yearly Process Operating Costs

Material Processed	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Leach Processed	K Tonnes	0	115	153	157	149	161	143	151	-	1,030
Gold Production	K Ozs Au	-	9	16	14	15	16	16	22	1	108
Silver Production	K Ozs Ag	-	33	64	98	162	246	170	342	22	1,137
Leaching Cost											
Plant Personnel	K USD	$ 53	$ 634	$ 634	$ 634	$ 634	$ 634	$ 634	$ 634	$ -	$ 4,488
Reagents and Consumables	K USD	$ 2	$ 549	$ 797	$ 1,064	$ 1,011	$ 1,094	$ 972	$ 1,025	$ -	$ 6,513
Power	K USD	$ 3	$ 787	$ 1,009	$ 896	$ 852	$ 921	$ 819	$ 864	$ -	$ 6,151
Suppy / Aux Service	K USD	$ 0	$ 46	$ 67	$ 89	$ 85	$ 92	$ 82	$ 86	$ -	$ 547
Total Leaching Costs	K USD	$ 58	$ 2,016	$ 2,506	$ 2,683	$ 2,581	$ 2,740	$ 2,505	$ 2,608	$ -	$ 17,699
Leaching Cost per Tonne											
Plant Personnel	$/t Leached	$ 112.95	$ 5.50	$ 4.14	$ 4.04	$ 4.25	$ 3.93	$ 4.43	$ 4.20	$ -	$ 4.36
Reagents and Consumables	$/t Leached	$ 4.76	$ 4.76	$ 5.20	$ 6.79	$ 6.79	$ 6.79	$ 6.79	$ 6.79	$ -	$ 6.33
Power	$/t Leached	$ 6.83	$ 6.83	$ 6.59	$ 5.72	$ 5.72	$ 5.72	$ 5.72	$ 5.72	$ -	$ 5.97
Suppy / Aux Service	$/t Leached	$ 0.40	$ 0.40	$ 0.44	$ 0.57	$ 0.57	$ 0.57	$ 0.57	$ 0.57	$ -	$ 0.53
Total Leaching Costs	$/t Leached	$ 124.94	$ 17.49	$ 16.36	$ 17.12	$ 17.33	$ 17.01	$ 17.51	$ 17.28	$ -	$ 17.19

Each cost center cost presented in Table 21.9 and Table 21.10 considers the following:

- The staff involved in the operation includes 38 management and supervision level personnel, production operators, plant maintenance and shift replacement personnel. The allocated salaries are in line with local Peruvian salary levels including social benefits. The same total staff for process operation is considered in both production levels; and

- The reagents and consumables costs are defined based on estimated unit consumption and unit costs obtained from reference data and prices from suppliers in the local market. For the 350 TPD operation, the reagent cost is presented in Table 21.12 and for the 500 TPD operation, the reagent cost is presented in Table 21.13

Table 21.12 Reagent Cost 350 TPD Process

Item	Description	Unitary consumption rate	Cost	Total
		Kg/t	$/kg	
	Cyanide	1	$ 2.20	$ 2.20
	Lime	0.5	$ 0.16	$ 0.08
	Portland Cement Type 1	4	$ 0.20	$ 0.80
	Diatomaceous earth	0.21	$ 0.90	$ 0.19
	Zinc powder	0.24	$ 3.03	$ 0.73
	Borax	0.01	$ 2.16	$ 0.02
	Sodium nitrate	0.02	$ 1.75	$ 0.04
	Sodium carbonate	0.02	$ 1.11	$ 0.02
	Diesel	0.85	$ 0.80	$ 0.68
	Total costo reactivos de Operacion ($/t)			$ 4.76

Table 21.13 Reagent Cost 500 TPD Process

Item	Description	Unitary consumption rate	Cost	Total
		Kg/t	$/kg	
	Cyanide	1.43	$ 2.20	$ 3.14
	Lime	0.71	$ 0.16	$ 0.11
	Portland Cement Type 1	5.71	$ 0.20	$ 1.14
	Diatomaceous earth	0.30	$ 0.90	$ 0.27
	Zinc powder	0.34	$ 3.03	$ 1.04
	Borax	0.01	$ 2.16	$ 0.03
	Sodium nitrate	0.03	$ 1.75	$ 0.05
	Sodium carbonate	0.03	$ 1.11	$ 0.03
	Diesel	1.21	$ 0.80	$ 0.97
	Total costo reactivos de Operacion ($/t)			$ 6.79

Electrical power consumption is estimated for the two production levels in Table 21.14 and Table 21.15. The cost of energy is considered to be 0.21$/kW.

Table 21.14 Process Electric Power Cost for 350 TPD Process

Area		Consumption	Operating Hours/day	Utilization	Index	TOTAL
		(kW)	Hrs		(kW-h/t)	($/t)
	Crushing	322	12	70%	7.73	$ 1.62
	Heap Leaching	100	24	100%	6.86	$ 1.44
	Merrill-Crowe	120	24	70%	5.76	$ 1.21
	Smelting	50	24	18%	0.60	$ 0.13
	Preparation Reagents	35	24	25%	0.60	$ 0.13
	Auxiliary services	158	24	100%	10.83	$ 2.28
	Water Supply	25	8	26%	0.15	$ 0.03
Operations Total Energy Cost		**($/t)**				**$ 6.83**

Table 21.15 Process Electric Power Cost for 500 TPD Process

Area		Consumption	Operating Hours/day	Utilization	Index	TOTAL
		(kW)	Hrs		(kW-h/t)	($/t)
	Crushing	322.0	17.1	70%	7.73	$ 1.62
	Heap Leaching	100.0	24.0	100%	4.80	$ 1.01
	Merrill-Crowe	120.0	24.0	100%	5.76	$ 1.21
	Smelting	50.0	24.0	25%	0.60	$ 0.13
	Preparation Reagents	35.0	24.0	36%	0.60	$ 0.13
	Auxiliary services	158.0	24.0	100%	7.58	$ 1.59
	Water Supply	25.0	8.0	37%	0.15	$ 0.03
Operations Total Energy Cost		**($/t)**				**$ 5.72**

Supplies and services cost has been allocated as 2% to cover related cost for these items.

21.2.3 General and Administration Costs

The G&A operating costs include the Lima office administrative services, support staff at the plant, technical support and consultants, and community support. The yearly cost estimate is shown in Table 21.16 below.

Table 21.16 Yearly G&A Costs

Material Processed	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Leach Processed	K Tonnes	0	115	153	157	149	161	143	151	-	1,030
Gold Production	K Ozs Au	-	9	16	14	15	16	16	22	1	108
Silver Production	K Ozs Ag	-	33	64	98	162	246	170	342	22	1,137
G&A Costs											
Lima Office - Admin Services	K USD	$ -	$ 39	$ 39	$ 39	$ 39	$ 39	$ 39	$ 39	$ 13	$ 286
Support staff at plant	K USD	$ -	$ 102	$ 102	$ 102	$ 102	$ 102	$ 102	$ 102	$ 34	$ 748
Technical support / consultants	K USD	$ -	$ 44	$ 44	$ 44	$ 44	$ 44	$ 44	$ 44	$ 15	$ 323
Community support	K USD	$ -	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 155	$ 52	$ 1,137
Total G&A	K USD	$ -	$ 340	$ 340	$ 340	$ 340	$ 340	$ 340	$ 340	$ 113	$ 2,493
G&A Cost per Tonne											
Lima Office - Admin Services	$/t Processed	$ -	$ 0.34	$ 0.25	$ 0.25	$ 0.26	$ 0.24	$ 0.27	$ 0.26	$ -	$ 0.28
Support staff at plant	$/t Processed	$ -	$ 0.89	$ 0.67	$ 0.65	$ 0.68	$ 0.63	$ 0.71	$ 0.68	$ -	$ 0.73
Technical support / consultants	$/t Processed	$ -	$ 0.38	$ 0.29	$ 0.28	$ 0.30	$ 0.27	$ 0.31	$ 0.29	$ -	$ 0.31
Community support	$/t Processed	$ -	$ 1.35	$ 1.01	$ 0.99	$ 1.04	$ 0.96	$ 1.08	$ 1.03	$ -	$ 1.10
Total G&A	$/t Processed	$ -	$ 2.95	$ 2.22	$ 2.17	$ 2.28	$ 2.11	$ 2.38	$ 2.25	$ -	$ 2.42

Additional detail for G&A costs, applicable to both the 350 TPD and 500 TPD cases is shown in Table 21.17.

Table 21.17 G&A Costs Details

Item	Description	Annual Cost		US$ per ton
		(US $)		
1	Lima Office (management, accounting, logistics, office)	$ 39,000.00		$ 0.31
2	Support staff at plant	$ 102,000.00		$ 0.80
3	Technical support / consultants	$ 44,000.00		$ 0.34
4	Community support	$ 155,000.00		$ 1.21
	Total G&A ($/t)	**340,000**		$ 2.66

21.2.4 Reclamation Costs

The total reclamation cost estimate was provided by PPX in the amount of $3.8 million dollars. This was applied over a 6-month period starting in month 89 after the end of production.

22.0 ECONOMIC ANALYSIS (ITEM 22)

Mr. Dyer performed an economic analysis based on the production schedule, operating and capital costs, metallurgical recoveries, and other information provided in previous sections. Both pre-tax and after-tax cash flows were estimated. The following sections discuss the buildup of the economic model.

22.1 Mining Physicals

The mining physicals are based on the mining and processing schedules. Gold and silver recoveries were applied based on the metallurgical recommendations of 80% for gold and 33% for silver. The recoveries were used to estimate the "recoverable" ounces and a lag time was applied to provide the timing of metal recoveries over a 4-month period. The applied lag times account for the percentage of the recoverable ounces credited to production on a month by month basis. This is accumulated as the produced ounces of gold and silver. The lag times used are shown in Table 22.1.

Table 22.1 Leach Lag Times

	Au	Ag
Month Placed	0%	0%
First Month	60%	60%
Second Month	25%	25%
Third Month	10%	10%
Forth Month	5%	5%
Fifth Month	0%	0%
Sixth Month	0%	0%
Total	100%	100%

Table 22.2 shows the mining and process production estimate used for the Igor cash-flow model.

Table 22.2 Yearly Mine and Process Physicals

Mining	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Ore Drives	K Tonnes	0	29	34	22	30	24	16	5	-	161
	g Au/t	2.51	2.83	2.77	3.22	3.17	2.54	3.13	3.52	-	2.94
	K Ozs Au	0	3	3	2	3	2	2	1	-	15
	g Ag/t	12.28	26.63	36.91	91.02	105.87	89.41	126.97	233.54	-	78.36
	K Ozs Ag	0	25	41	65	101	69	66	40	-	406
Stoping	K Tonnes	-	86	119	135	119	137	127	146	-	868
	g Au/t	-	3.87	4.27	3.72	3.94	4.21	4.63	5.26	-	4.31
	K Ozs Au	-	11	16	16	15	19	19	25	-	120
	g Ag/t	-	33.92	42.89	58.86	106.42	154.47	128.66	194.88	-	108.86
	K Ozs Ag	-	94	164	255	408	681	525	913	-	3,039
Total Mineralized	K Tonnes	0	115	153	157	149	161	143	151	-	1,030
	g Au/t	2.51	3.60	3.94	3.65	3.78	3.96	4.46	5.20	-	4.10
	K Ozs Au	0.0	13.4	19.4	18.4	18.1	20.5	20.5	25.3	-	136
	g Ag/t	12.28	32.07	41.55	63.41	106.31	144.80	128.47	196.24	-	104.08
	K Ozs Ag	0	119	205	319	509	750	591	952	-	3,445
Expensed Development	K Tonnes	7	15	12	14	13	6	-	-	-	67
Capital Development	K Tonnes	15	27	31	40	31	1	-	-	-	145
Total Mined	K Tonnes	22	157	196	211	193	168	143	151	-	1,242
Backfill	K Tonnes	-	57	113	105	102	149	104	160	-	792
Processing											
Heap Leach	K Tonnes	0	115	153	157	149	161	143	151	-	1,030
	g Au/t	2.51	3.60	3.94	3.65	3.78	3.96	4.46	5.20	-	4.10
	K Ozs Au	0	13	19	18	18	21	21	25	-	136
	Prod K Ozs Au	-	9	16	14	15	16	16	22	1	108
	g Ag/t	12.28	32.07	41.55	63.41	106.31	144.80	128.47	196.24	-	104.08
	K Ozs Ag	0	119	205	319	509	750	591	952	-	3,445
	Prod K Ozs Ag	-	33	64	98	162	246	170	342	22	1,137
Total Processed	K Tonnes	0	115	153	157	149	161	143	151	-	1,030
	g Au/t	2.51	3.60	3.94	3.65	3.78	3.96	4.46	5.20	-	4.10
	K Ozs Au	0	13	19	18	18	21	21	25	-	136
	Prod K Ozs Au	-	9	16	14	15	16	16	22	1	108
	g Ag/t	12.28	32.07	41.55	63.41	106.31	144.80	128.47	196.24	-	104.08
	K Ozs Ag	0	119	205	319	509	750	591	952	-	3,445
	Prod K Ozs Ag	-	33	64	98	162	246	170	342	22	1,137

22.2 Taxes

The production schedules shown in Table 22.2 were used to estimate revenues and costs for the operating cash-flow. Capital costs were subtracted from the operating cash-flow to estimate the pre-tax cash flow. Taxes were estimated considering corporate and Peruvian taxes. Mr. Dyer is not an expert in tax situations and has relied on PPX to provide the tax treatment methodologies.

The tax estimate starts with the net operating cash flow, and then reduces it based on deductions for tax pools and depreciation. The tax pools considered include:

- Losses Carried Forward – Includes $7.07 million recovered over the LOM;
- Capital Depreciation – Total initial capital less the development capital depreciated over a seven year period;
- Exploration Expenses – A total of $20.7 million depreciated over seven years; and
- Mine Site Development – Costs of $6.04 million deducted from remaining cash flow in years 2 through 5.

Table 22.3 shows the use of tax pools and depreciation applied for the tax calculation. The net taxable income was calculated based on the reduction of the operating cash flow by the values in Table 22.3. A tax rate of 29.4% was applied based on input from PPX.

Table 22.3 Tax Pools Used

Losses Carried Forward	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Opening	K USD	$ 4,668	$ 4,668	$ 3,345	$ 2,479	$ 1,718	$ 15	$ 400	$ 400			
End of 2017	K USD	$ 4,268										
2018 and Forward Estimate	K USD	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	$ 400	400		
Use of LCF	K USD		$ (1,723)	$ (1,266)	$ (1,162)	$ (2,103)	$ (15)	$ (400)	$ (400)	$ -		$ (7,068)
Closing	K USD	$ 4,668	$ 3,345	$ 2,479	$ 1,718	$ 15	$ 400	$ 400	$ 400	$ 400		
Capex												
Closing	K USD	$ 6,499	$ 6,499	$ 5,570	$ 4,642	$ 3,714	$ 2,785	$ 1,857	$ 928	$ -		
Usage	K USD			$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)		$ (6,499)
Exploration Expenses												
End of 2017	K USD	$ 18,705										
2018 Exploration	K USD	$ 2,000										
Useage	K USD			$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)		$ (20,705)
Minesite Development												
End of 2017	K USD	$ -										$ -
Opening	K USD	$ -	$ 629	$ 1,745	$ 362	$ 953	$ -	$ -	$ -	$ -	$ -	$ 3,689
Additions	K USD	$ 629	$ 1,116	$ 1,303	$ 1,670	$ 1,287	$ 39	$ -	$ -	$ -	$ -	$ 6,045
Usage	K USD			$ (2,687)	$ (1,079)	$ (2,240)	$ (39)	$ -	$ -	$ -	$ -	$ (6,045)
Closing	K USD	$ 629	$ 1,745	$ 362	$ 953	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,689

Peruvian royalty taxes are estimated based on information provided by PPX. The tax is a royalty on revenues based on operating margins. The estimated tax rate ("TE") is based on the Equation 3 and uses values from Table 22.4 Table 22.5 shows the resulting tax calculations.

Equation 3 Estimated Tax Rate

$$TE = \{[\sum (Ls - Li)\ j \times Tmg\ j\ ] + (MgO - Li)\ n \times Tmgn\}/MgO$$

Ls = Upper limit of the attached table

Li = Lower limit of the attached table

Tmg j = Marginal rate of the attached table

MgO = Operating margin in percent

j = Stretches from 1 to n-1

n = Number of the section where locate the operating margin

Table 22.4 Peruvian Tax Rates as Function of Operating Margin

N°			Margin Rate
	Li	Ls	
1	0	10%	1.00%
2	10%	15%	1.75%
3	15%	20%	2.50%
4	20%	25%	3.25%
5	25%	30%	4.00%
6	30%	35%	4.75%
7	35%	40%	5.50%
8	40%	45%	6.25%
9	45%	50%	7.00%
10	50%	55%	7.75%
11	55%	60%	8.50%
12	60%	65%	9.25%
13	65%	70%	10.00%
14	70%	75%	10.75%
15	75%	80%	11.50%
16			12.00%

Table 22.5 Yearly Peruvian Tax Calculations

Royalty payable calculations	**Units**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Total**
Gross revenues	K USD	$ -	$ 11,151	$ 20,545	$ 19,193	$ 20,915	$ 24,309	$ 22,053	$ 33,136	$ 1,522	$ 152,823
Operating Costs	K USD	$ -	$ (8,628)	$ (10,892)	$ (11,331)	$ (10,801)	$ (11,332)	$ (9,905)	$ (10,498)	$ (3,917)	$ (77,303)
Refining costs	K USD	$ -	$ (42)	$ (78)	$ (91)	$ (124)	$ (171)	$ (130)	$ (235)	$ (14)	$ (885)
Depletion (development costs)	K USD	$ -	$ (1,745)	$ (1,303)	$ (1,670)	$ (1,287)	$ (39)	$ -	$ -	$ -	$ (6,045)
Depreciation	K USD	$ -	$ -	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (6,499)
Operating profit for royalty	K USD	$ -	$ 736	$ 7,343	$ 5,173	$ 7,775	$ 11,838	$ 11,089	$ 21,474	$ (3,337)	$ 62,091
Operating margin %	%	0.0%	6.6%	35.7%	27.0%	37.2%	48.7%	50.3%	64.8%	-219.2%	40.6%
Royalty Rate Number	#	-	1	7	5	7	9	10	12	-	
Royalty rate	%	0.00%	1.00%	2.67%	2.05%	2.78%	3.61%	3.72%	4.79%	0.00%	

22.3 Cash-Flow Analysis

Mr. Dyer has estimated the pre-tax and post-tax cash flows as shown in Table 22.6. This shows a strong after-tax internal rate of return ("IRR") of 59%, which is reflective of the minimal continuing requirement for capital expenditure. The after-tax net present value ("NPV") at 5% is $30.1 million USD. The analysis is based on a gold price of $1,250 per ounce and a silver price of $16.50 per ounce.

Table 22.6 Igor Project Cash-Flow Evaluation

Revenues	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Payable Gold (99%)	K Ozs Au	-	8	16	14	15	16	15	22	1	-	107
Payable Silver (99%)	K Ozs Ag	-	33	63	97	160	244	168	339	21	-	1,126
Equivalent Ounces Au	K Ozs AuEq	-	9	16	15	17	19	18	27	1	-	122
Gross Revenue - Au	K USD	$ -	$ 10,608	$ 19,508	$ 17,588	$ 18,274	$ 20,285	$ 19,274	$ 27,546	$ 1,168	$ -	$ 134,251
Gross Revenue - Ag	K USD	$ -	$ 543	$ 1,037	$ 1,604	$ 2,641	$ 4,023	$ 2,779	$ 5,589	$ 355	$ -	$ 18,572
Refining Cost - Au	K USD	$ -	$ (25)	$ (47)	$ (42)	$ (44)	$ (49)	$ (46)	$ (66)	$ (3)	$ -	$ (322)
Refining Cost - Ag	K USD	$ -	$ (16)	$ (31)	$ (49)	$ (80)	$ (122)	$ (84)	$ (169)	$ (11)	$ -	$ (563)
NPR Royalty	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Revenue	K USD	$ -	$ 11,109	$ 20,467	$ 19,102	$ 20,792	$ 24,138	$ 21,922	$ 32,900	$ 1,509	$ -	$ 151,938
Operating Costs												
Expensed Mine Development	K USD	$ -	$ 628	$ 492	$ 602	$ 548	$ 249	$ -	$ -	$ -	$ -	$ 2,520
UG Mining Costs	K USD	$ -	$ 5,644	$ 7,554	$ 7,705	$ 7,331	$ 8,003	$ 7,060	$ 7,550	$ -	$ -	$ 50,846
Surface Rehandle	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Costs	K USD	$ -	$ 2,016	$ 2,506	$ 2,683	$ 2,581	$ 2,740	$ 2,505	$ 2,608	$ -	$ -	$ 17,640
G&A Costs	K USD	$ -	$ 340	$ 340	$ 340	$ 340	$ 340	$ 340	$ 340	$ 113	$ -	$ 2,493
Other Costs	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation Costs	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,804	$ -	$ 3,804
Total Operating Costs	K USD	$ -	$ 8,628	$ 10,892	$ 11,331	$ 10,801	$ 11,332	$ 9,905	$ 10,498	$ 3,917	$ -	$ 77,303
Stream Equivalent Ounces	K Ozs AuEq	-	0.9	1.6	1.5	1.7	1.9	1.8	2.7	0.1	-	12.2
RIVI Finance Payments	K USD	$ -	$ 758	$ 1,397	$ 1,305	$ 1,422	$ 1,653	$ 1,500	$ 2,253	$ 104	$ -	$ 10,392
Net Operating Cash Flow After RIVI	K USD	$ -	$ 9,386	$ 12,289	$ 12,636	$ 12,223	$ 12,985	$ 11,405	$ 12,751	$ 4,021	$ -	$ 87,695
Net Operating Cash Flow	K USD	$ -	$ 1,723	$ 8,178	$ 6,466	$ 8,569	$ 11,153	$ 10,517	$ 20,149	$ (2,512)	$ -	$ 64,243
Capital Costs												
Plant & Infrastucture Direct Costs	K USD	$ 2,858	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,858
Plant & Infrastucture Indirect Costs	K USD	$ 357	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 357
Contingency - Plant & Infrastucture	K USD	$ 428	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 428
Capitalized Development	K USD	$ 629	$ 1,116	$ 1,303	$ 1,670	$ 1,287	$ 39	$ -	$ -	$ -	$ -	$ 6,045
Other Mining Capital	K USD	$ 482	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 90	$ -	$ 572
Contractor Adjustments	K USD	$ 2,374	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,374
Other Owner's Capital	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Salage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (908)	$ -	$ (908)
Total Capital Cost	K USD	$ 7,128	$ 1,116	$ 1,303	$ 1,670	$ 1,287	$ 39	$ -	$ -	$ (818)	$ -	$ 11,726
Pre-Tax Cash Flow	K USD	$ (7,128)	$ 606	$ 6,875	$ 4,796	$ 7,281	$ 11,113	$ 10,517	$ 20,149	$ (1,694)	$ -	$ 52,516
Cumulative Pre-Tax Cash Flow	K USD	$ (7,128)	$ (6,522)	$ 353	$ 5,149	$ 12,431	$ 23,544	$ 34,061	$ 54,211	$ 52,516	$ -	
Pre-Tax IRR	%	64%										
Pre-Tax NPV (5%)	K USD	$ 37,664										
Pre-Tax NPV (8%)	K USD	$ 31,037										
Pre-Tax NPV (10%)	K USD	$ 27,335										
Tax Considerations												
Net Operating Cash Flow	K USD	$ -	$ 1,723	$ 8,178	$ 6,466	$ 8,569	$ 11,153	$ 10,517	$ 20,149	$ (2,512)	$ -	$ 64,243
Use of Tax Pools												
Exploration expenses	K USD	$ -	$ -	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ (2,958)	$ -	$ (20,705)
Capex	K USD	$ -	$ -	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ (928)	$ -	$ (6,499)
Contractor Adjustment	K USD			$ (339)	$ (339)	$ (339)	$ (339)	$ (339)	$ (339)	$ (339)		$ (2,374)
Loss Carried Forward	K USD		$ (1,723)	$ (1,266)	$ (1,162)	$ (2,103)	$ (15)	$ (400)	$ (400)			$ (7,068)
Mine site development costs	K USD	$ -	$ -	$ (2,687)	$ (1,079)	$ (2,240)	$ (39)	$ -	$ -			$ (6,045)
Net Taxable Income	K USD	$ -	$ -	$ -	$ -	$ -	$ 6,874	$ 5,892	$ 15,524			$ 28,290
Tax Payable (29.5%)	K USD	$ -	$ -	$ -	$ -	$ -	$ 2,028	$ 1,738	$ 4,580	$ -	$ -	$ 8,345
Peruvian Royalty Tax Payable	K USD	$ -	$ 7	$ 196	$ 106	$ 216	$ 428	$ 413	$ 1,030	$ -	$ -	$ 2,395
Net After-Tax Cash Flow	K USD	$ (7,128)	$ 599	$ 6,679	$ 4,690	$ 7,066	$ 8,658	$ 8,366	$ 14,540	$ (1,694)	$ -	$ 41,776
Cumulative After-Tax Cash Flow	K USD	$ (7,128)	$ (6,529)	$ 150	$ 4,840	$ 11,905	$ 20,563	$ 28,930	$ 43,470	$ 41,776	$ -	
After-Tax IRR	%	59%										
After-Tax NPV (5%)	K USD	$ 30,075										
After-Tax NPV (8%)	K USD	$ 24,817										
After-Tax NPV (10%)	K USD	$ 21,869										
After-tax Payback	Years	1.98										

22.4 Sensitivity

Sensitivity analysis was completed for metal prices, operating costs, and capital. These were done for both pre- and after-tax scenarios. The metal price sensitivity used a constant ratio between gold and silver prices of $1,250 gold to $16.50 silver. Table 22.7 shows the pre-tax sensitivity of the NPV and IRR to changes in gold prices from $1,100 to $1,400 in $25 increments. Table 22.8 shows the after-tax sensitivities to metal prices in the same range of gold prices.

Table 22.7 Pre-Tax Gold Price Sensitivity

Au Price	NPV (5%)	NPV (8%)	NPV (10%)	IRR
$ 1,100	$ 25,000	$ 20,122	$ 17,411	45%
$ 1,125	$ 27,111	$ 21,941	$ 19,065	48%
$ 1,150	$ 29,221	$ 23,760	$ 20,719	51%
$ 1,175	$ 31,332	$ 25,579	$ 22,373	55%
$ 1,200	$ 33,443	$ 27,399	$ 24,027	58%
$ 1,225	$ 35,554	$ 29,218	$ 25,681	61%
$ 1,250	$ 37,664	$ 31,037	$ 27,335	64%
$ 1,275	$ 39,775	$ 32,856	$ 28,989	67%
$ 1,300	$ 41,886	$ 34,675	$ 30,643	70%
$ 1,325	$ 43,996	$ 36,494	$ 32,297	73%
$ 1,350	$ 46,107	$ 38,313	$ 33,951	76%
$ 1,375	$ 48,218	$ 40,132	$ 35,605	79%
$ 1,400	$ 50,328	$ 41,951	$ 37,259	82%

Table 22.8 After-Tax Gold Price Sensitivity

Au Price	NPV (5%)	NPV (8%)	NPV (10%)	IRR
$ 1,100	$ 21,516	$ 17,326	$ 14,987	43%
$ 1,125	$ 22,950	$ 18,582	$ 16,141	46%
$ 1,150	$ 24,381	$ 19,834	$ 17,291	48%
$ 1,175	$ 25,808	$ 21,083	$ 18,439	51%
$ 1,200	$ 27,232	$ 22,330	$ 19,584	54%
$ 1,225	$ 28,655	$ 23,575	$ 20,727	57%
$ 1,250	$ 30,075	$ 24,817	$ 21,869	59%
$ 1,275	$ 31,492	$ 26,058	$ 23,008	62%
$ 1,300	$ 32,907	$ 27,297	$ 24,146	65%
$ 1,325	$ 34,319	$ 28,532	$ 25,281	68%
$ 1,350	$ 35,730	$ 29,767	$ 26,415	71%
$ 1,375	$ 37,139	$ 31,000	$ 27,548	73%
$ 1,400	$ 38,546	$ 32,232	$ 28,679	76%

Pre-tax operating and capital cost sensitivities are shown in Table 22.9 and Table 22.11 respectively. The after-tax operating and capital cost sensitivities are shown in Table 22.10 and Table 22.12

respectively. Sensitivities to pre-tax and after-tax changes to revenues, operating costs, and capital costs are shown graphically in Figure 22.1 and Figure 22.2 respectively. While mostly sensitive to metal prices, the sensitivity analyses show that the project cash flows still make a good return and profit despite the changes to revenues or costs that are shown.

Table 22.9 Pre-Tax Operating Cost Sensitivity

% of Value	NPV (5%)	NPV (8%)	NPV (10%)	IRR
80%	$ 49,693	$ 41,492	$ 36,894	84%
85%	$ 46,686	$ 38,878	$ 34,504	79%
90%	$ 43,679	$ 36,264	$ 32,114	74%
95%	$ 40,671	$ 33,650	$ 29,725	69%
100%	$ 37,664	$ 31,037	$ 27,335	64%
105%	$ 34,657	$ 28,423	$ 24,945	59%
110%	$ 31,650	$ 25,809	$ 22,555	54%
115%	$ 28,642	$ 23,196	$ 20,165	49%
120%	$ 25,635	$ 20,582	$ 17,775	44%

Table 22.10 After-Tax Operating Cost Sensitivity

% of Value	NPV (5%)	NPV (8%)	NPV (10%)	IRR
80%	$ 38,205	$ 32,000	$ 28,507	78%
85%	$ 36,190	$ 30,220	$ 26,862	73%
90%	$ 34,162	$ 28,428	$ 25,206	69%
95%	$ 32,124	$ 26,628	$ 23,542	64%
100%	$ 30,075	$ 24,817	$ 21,869	59%
105%	$ 28,014	$ 22,997	$ 20,187	55%
110%	$ 25,945	$ 21,169	$ 18,497	50%
115%	$ 23,863	$ 19,331	$ 16,798	46%
120%	$ 21,773	$ 17,484	$ 15,092	42%

Table 22.11 Pre-Tax Capital Cost Sensitivity

% of Value	NPV (5%)	NPV (8%)	NPV (10%)	IRR
80%	$ 39,827	$ 33,099	$ 29,334	78%
85%	$ 39,286	$ 32,583	$ 28,834	74%
90%	$ 38,745	$ 32,068	$ 28,334	70%
95%	$ 38,205	$ 31,552	$ 27,835	67%
100%	$ 37,664	$ 31,037	$ 27,335	64%
105%	$ 37,124	$ 30,521	$ 26,835	61%
110%	$ 36,583	$ 30,006	$ 26,335	58%
115%	$ 36,042	$ 29,490	$ 25,835	56%
120%	$ 35,502	$ 28,975	$ 25,335	54%

Table 22.12 After-Tax Capital Cost Sensitivity

% of Value	NPV (5%)	NPV (8%)	NPV (10%)	IRR
80%	$ 31,914	$ 26,609	$ 23,627	73%
85%	$ 31,454	$ 26,161	$ 23,188	69%
90%	$ 30,994	$ 25,713	$ 22,748	66%
95%	$ 30,535	$ 25,265	$ 22,308	62%
100%	$ 30,075	$ 24,817	$ 21,869	59%
105%	$ 29,615	$ 24,369	$ 21,429	57%
110%	$ 29,154	$ 23,921	$ 20,989	54%
115%	$ 28,694	$ 23,473	$ 20,549	52%
120%	$ 28,234	$ 23,024	$ 20,109	50%

Figure 22.1 Pre-Tax Sensitivity Chart

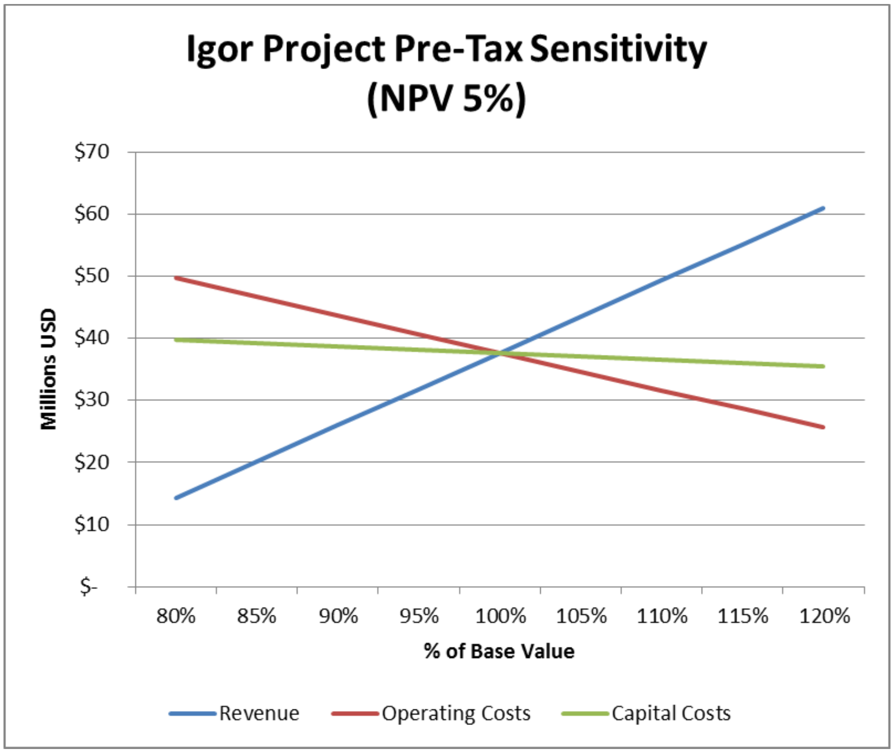
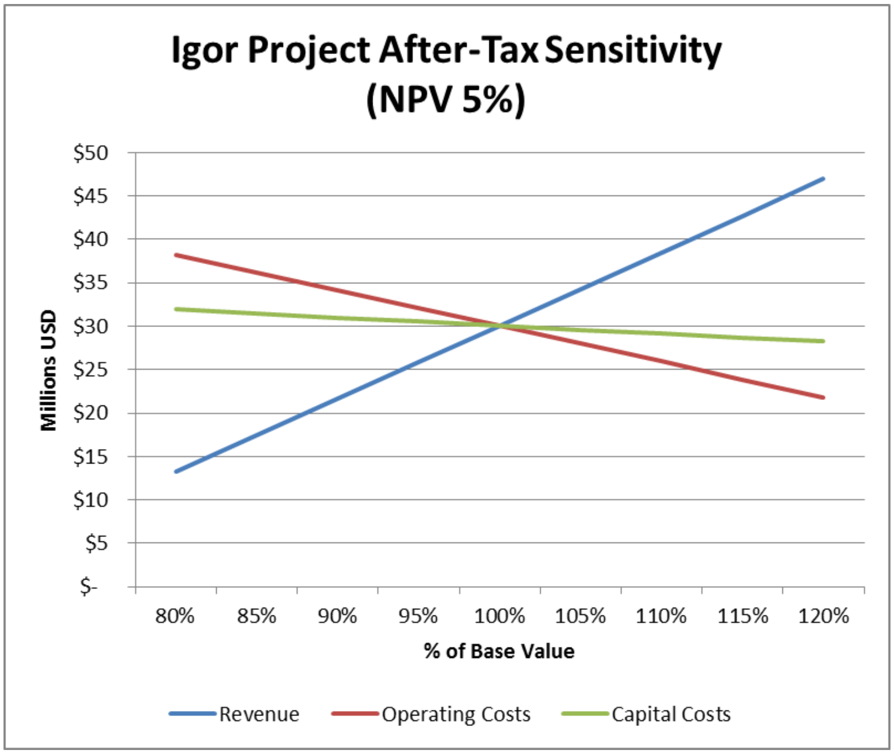


Figure 22.2 After-Tax Sensitivity Chart



23.0 ADJACENT PROPERTIES (ITEM 23)

The authors are not aware of any information regarding adjacent properties that is relevant to the resource and reserve estimates, and the PFS that are the subject of this report.

24.0 OTHER RELEVANT DATA AND INFORMATION (ITEM 24)

The authors are not aware of any additional information or explanation necessary to make this technical report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS (ITEM 25)

Based on the positive results of this PFS, the project should continue on a path to a production decision. There are opportunities to improve the project.

The authors have reviewed the project data, including the Igor drill-hole database and metallurgical test results. Mr. Tietz and Mr. Seamons visited the project site. The authors believe that the data provided by PPX, as well as the geological interpretations PPX has derived from the data, are generally an accurate and reasonable representation of the Igor project, and supports the use of these data in mineral resource and mineral reserve estimation.

Igor project gold and silver mineralization occurs primarily within narrow structurally-controlled breccias hosted within Mesozoic-age siltstone, sandstone, and quartzite. The breccia zones contain multiple stages of quartz, variably silicified hydrothermal breccias, and later tectonic breccia emplacement. Four target areas have been identified: Callanquitas, Tesoros, Domo, and Portachuelos.

The mineral resources stated within this technical report are all within the Callanquitas deposit which consists of two distinct north-trending structural zones which appear to merge at depth within the southern portion of the deposit area. The mineral zones (Callanquitas Oeste and Callanquitas Este) have a combined northerly strike of 1,400m, a drill defined depth of up to 500m, and average 1-2m in thickness.

The stated mineral resources are based on an underground mining scenario with heap-leach processing and are constrained by a 2.0g AuEq/t grade cut-off. The gold equivalent value is based on prices of US$1,350.00 per ounce gold and US$18.00 per ounce silver, and a 3:1 recovery difference between gold and silver. Measured and Indicated resources are limited to oxide material and total 1.47 million tonnes at an average grade of 4.72g Au/t and 109.5g Ag/t. This equates to 223,000 contained ounces of gold and 5,177,000 contained ounces of silver. The oxide Inferred resource is 0.34 million tonnes at an average grade of 4.58g Au/t and 124.6g Ag/t while the sulfide Inferred resource is 0.27 million tonnes at an average grade of 2.97 Au/t and 159.3g Ag/t.

Mineral reserves were determined using stope optimization, smoothing of underground mine designs, design of required development, and estimation of costs and cash flows. The cutoff grade value of 2.50g AuEq/t was used to define underground mineable material for reserves. Proven and Probable Reserves at Callanquitas, as summarized in Table 25.1 include only Measured and Indicated resource classifications within the final mine design at or above a cutoff grade of 2.5g AuEq/t.

Table 25.1 Igor Mine Mineral Reserves at 2.5g AuEq/t Cutoff

Type	Classification	K Tonnes	g Au/t	K Ozs Au	g Ag/t	K Ozs Ag	g AuEq/t	K Ozs AuEq
Oxide	Proven	8	4.56	1	32.3	8	4.74	1
	Probable	1,022	4.09	134	104.6	3,437	4.66	153
	Proven & Probable	**1,030**	**4.10**	**136**	**104.1**	**3,445**	**4.66**	**154**

The economic analysis of the mine was completed both on a pre-tax and after-tax basis using a $1,250 per ounce gold price and a silver price of $16.50 per ounce. The project has a strong after-tax internal rate of return ("IRR") of 59%, which is reflective of the minimal continuing requirement for capital expenditure. The after-tax net present value ("NPV") at 5% is $30.1 million USD.

26.0 RECOMMENDATIONS (ITEM 26)

This PFS indicates the Igor project is a viable mining and heap-leach processing operation, and work should continue on advancing the project to a production decision. The authors recommend a work program of $3.3 million as summarized in Table 26.1 and outlined below.

PPX should evaluate the possible addition of a conventional milling circuit to augment or replace the heap leach facility later in the mine life. This could potentially include a capacity expansion should resource and reserve drilling identify additional areas for mining to support higher capacity at the processing facility.

Given the open nature of the Callanquitas deposit and the possible extension of the deposit to the south and at depth, additional expansion and exploration core drilling, from the surface as well as underground, should be conducted to determine the ultimate dimensions of the Callanquitas deposit and its intersection with the Portachuelos target area. Although results from drilling in the Portachuelos area were not considered in this PFS, the intersection of these mineralized structures presents an attractive exploration target that should be investigated by additional exploration drilling.

Metallurgical testing: Additional metallurgical testing to better understand the deportment of silver is recommended. Sulfide-bearing gold and silver mineralization has been encountered both at depth within the Callanquitas deposit and at other target areas within the Igor Project**.** Should unoxidized mineralization prove to be voluminous, metallurgical testing will be required to determine an appropriate flow sheet, potential gold and silver recoveries, cost parameters, etc. to determine the economic potential of sulfide-bearing gold and silver mineralization.

Table 26.1 PPX Cost Estimate for the Recommended Program

Item	Est. Cost USD
Milling and CIL Scoping Study	$ 100,000
Expansion and Exploration Drilling	$ 3,000,000
Metallurgical Testing	$ 100,000
Resource Update	$ 100,000
Total:	**$ 3,300,000**

The authors believe that the Callanquitas mine and Igor project is a project of merit and warrants the proposed program and level of expenditures outlined above.

27.0 REFERENCES (ITEM 27)

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28.0 DATE AND SIGNATURE PAGE (ITEM 28)

Effective Date of report: December 3, 2018

The data on which the contained PFS is based was current as of the Effective Date.

Completion Date of report: January 18, 2019

Amended Date of report: January 4, 2022

“Paul Tietz”	January 4, 2022
Paul Tietz, C.P.G.	Date Signed
“ Thomas L. Dyer”	January 4, 2022
Thomas L. Dyer, P.E.	Date Signed
“ Joseph Seamons”	January 4, 2022
Joseph Seamons, P.E.	Date Signed
“ Diogenes Uceda”	January 4, 2022
Diogenes Uceda, P.Eng., PhD	Date Signed
“ Ernest Burga”	January 4, 2022
Ernest Burga, P.Eng.	Date Signed

29.0 CERTIFICATE OF QUALIFIED PERSONS (ITEM 29)

Paul Tietz, C.P.G.

I, Paul Tietz, C.P.G., do hereby certify here that I am currently employed as Senior Geologist by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502.

1. I graduated with a Bachelor of Science degree in Biology/Geology from the University of Rochester in 1977 and a Master of Science degree in Geology from the University of North Carolina, Chapel Hill in 1981. I also received a Master of Science degree in Geological Engineering from the University of Nevada, Reno in 2004. I have worked as a geologist for a total of 37 years since receiving my Master of Science degree in Geology.
2. I am a Certified Professional Geologist (#11004) with the American Institute of Professional Geologists. I have drilling and resource modeling experience in similar precious-metal deposits in Peru, the U.S.A., and Mexico.
3. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101. I am independent of PPX and its subsidiaries, as well as all previous operators, applying all of the tests in section 1.5 of National Instrument 43-101.
4. I am responsible for Sections 1.1, 1.2, 1.3, 1.5, 1.10, 2 through 12, 14, 20, 23, 24, and those parts of 25 and 26 pertaining to the project geology and mineral resource estimate, of this technical report titled Technical Report and Pre-Feasibility Study for the Callanquitas Gold-Silver Deposit, Igor Project, Región de La Libertad, Perú” for PPX Mining (“Technical Report”), and with an effective date of December 3, 2018.
5. I have had no previous involvement with this project. I visited the Igor project on January 12 – 15, 2018.
6. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, those parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. I hereby consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated January 4, 2022

“Paul Tietz”

Paul Tietz, C.P.G.
Print Name of Qualified Person

Joseph B Seamons, P.E.

I, Joseph B Seamons, P.E., do hereby certify here that I am currently employed as Senior Mining Engineer by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502.

1. I graduated with a Bachelor of Science degree in Mine Engineering from the University of Utah in 2011. I have worked as a mining engineer for a total of 8 years since my graduation.

2. I am a Registered Professional Engineer in the state of Nevada (#25482) and a Registered Member (#04152075) of the Society of Mining, Metallurgy and Exploration.

3. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of PPX and its subsidiaries, applying all of the tests in section 1.5 of National Instrument 43-101.

4. I am responsible for Sections 1.6, 1.7, 15, 16, and those parts of 25 and 26 pertaining to the mineral reserve estimate, of this technical report titled Technical Report and Pre-Feasibility Study for the Callanquitas Gold-Silver Deposit, Igor Project, Región de La Libertad, Perú" for PPX Mining ("Technical Report") and with an effective date of December 3, 2018.

5. I have had no previous involvement with this project. I visited the Igor project on June 17 to 22, 2018.

6. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, those parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

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

Dated January 4, 2022

"Joseph B Seamons"

Joseph B Seamons, P.E.
Print Name of Qualified Person
Thomas L. Dyer, P.E.

I, Thomas L. Dyer, PE, do hereby certify here that I am currently employed as Senior Engineer by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502.

1. I graduated with a Bachelor of Science degree in Mine Engineering from the South Dakota School of Mines and Technology in May of 1996. I have worked as a Mine Engineer for a total of 22 years since graduation.

2. I am a Registered Professional Engineer in the state of Nevada (#015729) and a Registered Member (#4029995) of the Society of Mining, Metallurgy and Exploration.

3. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4. I am independent of PPX and its subsidiaries, as well as all previous operators, applying all of the tests in section 1.5 of National Instrument 43-101.

5. I am responsible for Sections 1.8, 1.9, 19, 21, 22, and those parts of 25 and 26 pertaining to the economic analyses,, of this technical report titled *Technical Report and Pre-Feasibility Study for the Callanquitas Gold-Silver Deposit, Igor Project, Región de La Libertad, Perú"* for PPX Mining ("Technical Report"), and with an effective date of December 3, 2018.

6. I have had no previous involvement with this project. I have not visited the Igor project.

7. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, those parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

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

Dated January 4, 2022

"Thomas L. Dyer"

Thomas L. Dyer, P.E.
Print Name of Qualified Person

Diogenes A. Uceda, P.E.O. 90349069

I, Diogenes A. Uceda, P.Eng, MSc and PhD, residing at 606 Moonrock Avenue Sudbury, ON, P3E 5Z6 Canada, do hereby certify that:

1. I graduated with a Bachelor of Engineering (Aug 1968) and an Engineering degree (July 1976) in Metallurgy from the University of Engineering of Lima-Peru, a Master of Science degree in Metallurgical Engineering from Colorado School of Mines (May 1976) and a PhD from the University of Missouri (Dec 1988). I have practiced my profession continuously since 1967.
2. I am a Registered Member in good standing of the Professional Engineers of Ontario (90349069). I have worked as a process engineer, senior process engineer and process manager for extractive metallurgy, pyrometallurgy, hydrometallurgy, electrometallurgy, mineral processing and quality and environmental projects and operations in Canada, Australia, the Dominican Republic, and Peru.
3. I have read National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association as defined in NI 43-101 and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I am independent of PPX Mining Corporation and its related companies, as independence is described in Section 1.5 of NI 43-101.
5. I am one of the authors of this Technical Report titled "Technical Report and Pre-Feasibility Study for the Callanquitas Gold-Silver Deposit, Igor Project, Región de La Libertad, Perú" with an effective of December 3, 2018.
6. I am responsible for Sections 1.4, 13, 17, and those parts of 25 and 26 pertaining to the metallurgy and recovery methods analyses, of this technical report. . This technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.
7. I have not visited the project site and I have had no prior involvement with the Igor property that is the subject of this Technical Report.
8. At the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I hereby consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated January 4, 2022

Diogenes A. Uceda
Diogenes A. Uceda, P.Eng 90349069

Ernest Burga, P. Eng. Reg Number: 6067011

I, Mr. Ernest Burga, B.Sc., P. Eng., residing at 3385 Aubrey Rd., Mississauga, Ontario, L5L 5E3, do hereby certify that:

1. I am the President of Andeburg Consulting Services Inc. and I graduated with a Bachelor of Science degree in Mechanical Engineering from the University of Engineering, Lima, Peru in 1966. I have practiced my profession continuously in Canada since 1978.
2. I am a Registered Member in good standing of the Professional Engineers of Ontario (6067011).
3. I have read National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association as defined in NI 43-101 and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101. My summarized career experience is as follows:
   - Maintenance Engineer, Backus and Johnston Brewery of Peru 1966-1975
   - Design Mechanical Engineer, Cambrian Engineering Group 1975-1978
   - Design Mechanical Engineer, Reid Crowther Bendy 1979-1981
   - Lead Mechanical Engineer, Cambrian Engineering Group 1981-1987
   - Project Engineer, Hydro Metallurgical Division- HG. Engineering 1988-2003
   - Lead Mechanical Engineer, AMEC Americas 2003-2005
   - Sr. Mechanical Engineer, SNC Lavalin Ltd. 2005-2009
   - President, Andeburg Consulting Services Inc.- 2004 to present
   - Contracted Mechanical Engineer – P&E Mining Consultants Inc. 2009 to present
4. I am independent of PPX Mining Corp. and its related companies, as independence is described in Section 1.5 of NI 43-101.
5. I am one of the authors of this Technical Report titled *Technical Report and Pre-Feasibility Study (PFS) for the Igor Project, La Libertad, Peru,* prepared for PPX Mining Corp., with an effective date of December 3, 2018. I am responsible for co-authoring Sections 17, 18, 21.1.2, 21.2.2, 21.2.3 and all parts of the Summary, Interpretations and Conclusions, and Recommendations which pertain to those Sections. This technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.
6. I have not visited the project site.
7. At the effective date of this Technical Report, 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.
8. I hereby consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this January 4, 2022

"Ernest Burga"
Ernest Burga, P. Eng. Ontario (6067011)