NI43-101 Technical Report
Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico

Prepared for:

HELIOSTAR



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1 Summary

1.1 Introduction

The Cerro del Gallo (CdG) Project, part of the San Antón property, is located in the state of Guanajuato, Mexico, and is wholly owned by San Antón de Las Minas SA de CV (San Antón) a resident Mexican company owned by Heliostar Metals Ltd., a Canadian company with corporate offices in Vancouver, British Columbia, Canada.

At the request of Heliostar, this Technical Report was prepared by Kappes, Cassiday and Associates (KCA), Mine Technical Services (MTS), Hard Rock Consulting LLC (HRC), AECOM, Stantec and Heliostar with input from other consultant groups.

This Technical Report presents the results of a Pre-Feasibility Study (PFS) on the Cerro del Gallo Project and has been prepared in accordance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' current "Standards of Disclosure for Mineral Projects" under the provisions of National Instrument 43-101 (NI 43-101), Companion Policy 43-101 CP and Form 43-101F1.

The Cerro del Gallo Project considers open pit mining of approximately 92 million tonnes of ore with an estimated grade of 0.56 grams per tonne (g/t) gold, 13.2 g/t silver and 0.09% copper. Ore from the pit will be crushed to 80% passing 5mm, agglomerated with cement, conveyor stacked onto a heap leach pad and leached using a low concentration sodium cyanide solution. Pregnant solution from the heap leach will be processed in a Sulfidization, Acidification, Recirculation, Thickening (SART) plant where a copper - silver precipitate product will be produced, followed by a carbon adsorption-desorption-recovery (ADR) plant to produce a doré product.

The processing throughput for the Project is 6 million tonnes of ore per year after the second year, with the first year at 5.6 million tonnes. The scope of the PFS includes a mine production schedule, as well as costing for all process components and infrastructure required for the operation. This report is based on the oxide, mixed oxide, mixed sulfide and sulfide portions of the Proven and Probable Reserves for the Project.

1.2 Property Description and Location

The CdG deposit is located in the state of Guanajuato in central Mexico, approximately 30 kilometers east of Guanajuato City and 55 kilometers east of the international airport of Leon in an active mining district. The property is accessible by road, rail and air services. Additionally, there is availability of a skilled local workforce, grid power, water, sealed roads, equipment suppliers and established transport routes.

Figure 1.1: Location Map of the Cerro del Gallo Property in Guanajuato



Source: Heliostar (2025)

1.3 Ownership

The San Antón Property covers privately owned land. Heliostar, through their wholly owned subsidiary San Antón de Las Minas SA de CV, owns the portion on which the project described in this report will be built. There are no ejidos (community owned lands) present in the San Antón de las Minas community.

San Antón de Las Minas SA de CV owns freehold title and has surface rights to land totaling 445 hectares, including the CdG Project, a core shack warehouse in San Antón (6,927 m^2), and the Dolores Shaft office (29,209 m^2). Land access and compensation agreements have been obtained with the relevant landowners for access and exploration.

The CdG Project is subject to a Net Smelter Return (NSR) royalty of 4% on one mineral concession and 3% on five other concessions. The overall average is calculated to be 3.75% for the reserves outlined in this study.

1.4 Geology and Mineralization

The CdG deposit is located in central Mexico within the Mesa Central physiographic province that includes the Guerrero Composite Terrane. The Guerrero Composite Terrane is characterized by submarine and subaerial volcanic and sedimentary successions that range in age from Jurassic to Middle–Late Cretaceous.

The oldest rocks in the CdG region are a deformed and regionally metamorphosed volcano-sedimentary sequence of Triassic to Cretaceous age. Consejo de Recursos Minerales referred to these rocks as the Esperanza Formation, described as carbonaceous and calcareous shale interbedded with arenite, limestone and andesite to basalt flows, all weakly metamorphosed to phyllite, slate and marble.

In the CdG Project area, the Esperanza Formation consists of layered argillaceous and silty argillaceous sedimentary rocks, and fragmental volcanic rocks of intermediate composition, including ash tuffs, lithic to crystal tuffs and some volcanic breccias and agglomerates. In the Project area, the Esperanza Formation is locally surrounded by Tertiary age rhyolitic flows, rhyolitic tuffs, trachyte-andesite and andesites.

At CdG, mineralization is hosted in both felsic intrusive and surrounding volcano-sedimentary wall-rock of the Esperanza Formation. Mineralization occurs as disseminations and as fracture-controlled veins extending from 200 to 400 m outward from the mineralizing intrusive complex.

The strongest gold mineralization at CdG is associated with intense quartz stockwork veins and silicification within a wall-rock annulus around the outer limits of the felsic stock. The system loses intensity outward from this annulus with a decrease in stockwork and quartz veining density.

There are less than 2% by volume of sulfide minerals in the host rock. Gold-copper mineralization is zoned concentrically around the felsic intrusive with higher grade gold mineralization proximal to and within an outer annulus of the intrusion. The highest copper grades are found outward from the gold zone. Zinc mineralization is locally anomalous outside the copper zone. Metal zonation boundaries are gradational and there is an overlap in the gold-copper zone and the copper-zinc zone. Silver mineralization occurs with the gold-copper mineralization as well as later structurally controlled epithermal vein system that overprints the intrusive related copper-gold system.

1.5 Drilling and Sample Analysis

All drill data utilized in this Technical Report to calculate the current resource and reserves were generated by previous owners, Cerro Resources (Cerro) between 2004 and 2008, and Primero Mining Corp (Primero) in 2013. Argonaut performed limited drilling in 2018, solely for the purposes of obtaining metallurgical samples for test work. The drilling campaigns are summarized in **Error! Reference source not found.Error! Reference source not found.** below.

Table 1.1: Drilling Campaigns by Year, Company and Type

Company	Year	RC		DD		Total	
		Holes	*Meters*	*Holes*	*Meters*	*Holes*	*Meters*
	2004	3	335.28	-	-	3	335.28
	2005	80	14,884.92	9	5,402.95	89	20,287.87
Cerro	2006	109	24,005.97	31	12,935.80	140	36,941.77
	2007	63	13,838.33	30	15,452.17	93	29,290.50
	2008	27	7,036.30	7	3,569.05	34	10,605.35
Primero	2013	-	-	51	12,829.16	51	12,829.16
Argonaut	2018	-	-	18	1,484.50	18	1,484.50
Total (04-08)		282	60,100.80	77	37,359.97	359	97,460.77
Total		**282**	**60,100.80**	**146**	**51,673.63**	**428**	**111,774.43**
% DH Type of Total		65.9%	53.8%	34.1%	46.2%	100.0%	100.0%

Source: Argonaut (2019)

All core drilling from 2004 through 2008 by Cerro utilized oriented core and was performed by an Atlas Copco CS-1500 truck-mounted core drill. Normally the hole was started with HQ (63.5 mm) sized core and, if necessary, reduced to NQ (47.6 mm) sized core.

From 2013 through 2014, under Primero, the core drilling program was performed by Major Drilling with a Major 50 truck mounted drill. These holes were positioned in areas requiring closer spaced drilling, often because the original RC holes were not completed

and were lost in mineralization. Primero's drilling also included PQ (85 mm) sized core holes to twin mineralized RC drill holes. Core handling and logging procedures for Primero were identical to those of Cerro.

In 2018 Argonaut conducted a core drilling program using PQ size core. This program, totaling 1,484.5 meters, was designed to obtain mineralized material for metallurgical studies. Argonaut utilized a UDR 200 drill rig supplied by Major Drilling. This core drilling program was intended entirely to produce metallurgical test samples. As such, Argonaut twinned existing drill holes, and sent whole PQ core to the metallurgical testing lab.

Standard core handling procedures initiated by Cerro Resources were utilized throughout all three core drilling campaigns at CdG. Core cutting was done by saw, followed by industry standard sampling procedures. Argonaut, however, did not sample the core for standard assays as noted above. Logging criteria included: lithology, alteration, degree of oxidation, and mineralization. Drill hole data was recorded on handwritten logs onto a pre-printed log sheet template database and eventually merged with assay results. Drill data also included hole identification, coordinates (in NAD27 for Mexico), depth, and sample number, including duplicates, blanks and standards.

1.6 Mineral Processing and Metallurgical Test Work

The laboratory testing program included comminution and flotation studies along with column, bottle roll and agitated (shake) cyanide leach tests. The leach tests were differentiated by crush size and type (conventional and HPGR). Historical test work included laboratory work prior to 2015 performed by others. The most recent test work includes results from 2018 and 2019 performed by KCA.

Historical test work was conducted on three main ore types: Weathered/Oxide, Mixed, and Fresh/Sulfide. The current test work differentiates the Mixed category into two separate material types: Mixed Oxide and Mixed Sulfide. In total there are four separate material ore type categories that were considered in the recent test work and in this study: Oxide, Mixed Oxide, Mixed Sulfide and Sulfide.

The projected field gold, silver and copper recoveries are summarized in

Table 1.2**Error! Reference source not found.**, and the estimated cyanide and cement consumptions presented in Table 1.3.

Table 1.2: Cerro del Gallo Recoveries by Material Type

Ore Type (HPGR)	Feed Distribution LOM Average	Projected Field Recoveries		
		Au, %	Ag, %	Cu%
Weathered (Oxide)	9.2%	74	60	17
Mixed Oxide	5.8%	68	73	62
Mixed Sulfide	38.0%	61	58	73
Fresh (Sulfide)	47.0%	53	35	59
Life of Mine (LOM) Average		59	49	62

Source: KCA (2025)

Table 1.3: Cerro del Gallo Reagent Consumptions by Material Type

Ore Type (HPGR)	Feed Distribution LOM Average	Projected Field Reagents, kg/t		
		NaCN	Cement	Lime
Weathered (Oxide)	9.2%	0.87	10.0	–
Mixed Oxide	5.8%	0.34	10.0	–
Mixed Sulfide	38.0%	0.87	10.0	–
Fresh (Sulfide)	47.0%	0.60	10.0	–
Life of Mine (LOM) Average		0.71	10.0	–

Source: KCA (2025)

1.7 Mineral Resources

Mine Technical Services (MTS) was contracted to prepare an updated mineral resource estimate for the CdG Project. The QP for the Mineral Resource is Mr. Timothy O. Kuhl, Principal Geologist with MTS. Mr. Kuhl constructed a geology model that includes lithology domains, oxidation domains and grade shell domains based on audited database provided by Heliostar.

Data analysis was completed with histograms, probability plots, box plots and contract profiles. Grade estimates were completed for gold, silver and copper by lithology domain.

Design and economic parameters were used to calculate cutoff grades for the different oxidation material types. A Lerch-Grosman pit optimization was used to define a mineral resource shell that was used for the declaration of the CdG mineral resources.

The Mineral Resource commodity prices used were $2,500/oz (gold, $30.50/oz (Silver) and $4.60/lb (copper).

Table 1.4 summarizes the undiluted Indicated, and Inferred Mineral Resources constrained to the US$2,500 Au-EQ conceptual pit. The Mineral Resource tabulation is defined using NSR cutoff for Oxide ($11.81/t), Mixed Oxide ($10.66.t), Mixed Sulfide ($11.81/t) and Sulfide ($11.23/t). The stated resources are inclusive of reserves.

Table 1.4: Cerro del Gallo Mineral Resources

Classification	Material Type	NSR Cutoff	Tonnes (kt)	Grade				Contained Metal			
				Au g/t	Ag g/t	Cu %	AuEq g/t	Gold (koz)	Silver (koz)	Copper (t)	AuEq (koz)
Indicated	Oxide	$11.81	10,733	0.41	17.92	0.09	0.60	141	6,184	9,659	207
	Mix Oxide	$10.66	13,613	0.28	11.12	0.08	0.50	123	4,867	10,890	219
	Mix Sulfide	$11.81	70,066	0.40	13.70	0.09	0.68	901	30,862	63,060	1,532
	Sulfide	$11.23	145,572	0.38	13.77	0.11	0.62	1778	64,447	160,129	2,902
	Total		**239,984**	**0.38**	**13.78**	**0.10**	**0.63**	**2,944**	**106,359**	**243,739**	**4,859**
Inferred	Oxide	$11.81	2,042	0.19	21.08	0.09	0.40	12	1,384	1,838	26
	Mix Oxide	$10.66	1,604	0.14	16.12	0.07	0.40	7	831	1,123	21
	Mix Sulfide	$11.81	10,501	0.28	13.75	0.11	0.57	95	4,642	11,552	192
	Sulfide	$11.23	10,300	0.33	11.74	0.07	0.51	109	3,888	7,210	169
	Total		**24,448**	**0.28**	**13.67**	**0.09**	**0.52**	**224**	**10,746**	**21,722**	**408**

Notes to accompany Mineral Resources table:

1. Mineral Resources are reported within a resource shell constrained by the property boundary using the 2014 CIM Definition Standards.
2. Mineral Resources have an effective date of 31 July 2025. The Qualified Person for the estimate is Mr. Timothy O. Kuhl, Reg Mem SME and Principal Geologist with Mine Technical Services.
3. An NSR is used for reporting Mineral Resources by material type. NSR cutoffs of $11.81 for Oxide, $10.66 for Mixed Oxide, $11.81 for Mixed Sulfide and $11.23 for Sulfide were used. The NSR is determined based on estimated processing costs of US$9.10/t, general and administrative costs of US$0.90t, production taxes and royalty costs of US$1.40/t. Metal prices of US$2,500/oz Au, US$30.50/oz Ag, and US$4.60/lb Cu were used in calculating the NSR. In addition, a gold recovery of 74%, a silver recovery of 60% and a copper recovery of 17% were used for Oxide material, a gold recovery of 68%, a silver recovery of 73% and a copper recovery of 62% were used for Mixed Oxide material, a gold recovery of 61%, a silver recovery of 58% and a copper recovery of 73% were used for Mixed Sulfide material and a gold recovery of 53%, a silver recovery of 35% and a copper recovery of 59% were used for Sulfide material in the NSR calculation.
4. Based on the stated metal prices and recoveries the gold equivalent grades were calculated as AuEq = Au Grade + (((Cu Price in US$/lb * 22.0462 * Cu Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Cu Grade) + (((Ag Price in US$/g * Ag Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Ag Grade). The average overall payables from the smelter and refineries were estimated at 98.8% for gold, 90.1% for silver and 88.2% for copper.
5. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
6. Tonnage and grade estimates are in metric units.
7. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

1.8 Mineral Reserves

Mineral Reserves were estimated for the Cerro de Gallo Project which is planned to be an open pit operation with a heap leach pad. Pregnant solution from the heap leach will be processed in a Sulfidization, Acidification, Recirculation, Thickening (SART) plant where a copper - silver precipitate product will be produced, followed by a carbon adsorption-desorption-recovery (ADR) plant to produce a doré product.

The Mineral Resources for the deposit were evaluated using an Lerchs–Grossmann (L–G) pit optimizer to generate optimized pit shells. Pit shells were generated based on varying metal prices, with base prices of US$2,200/oz Au, US$26.50/oz Ag and US$4.00/lb Cu. Starting with the current topography, a total of 60 pit shells were generated to determine optimal break points for developing pit phases and for determining the ultimate final pit phase for the deposit.

The Mineral Reserves are based on three phases of pit designs developed by the QP for the pre-feasibility study. The open pit designs completed on the deposit were evaluated with the updated Indicated Mineral Resources and were demonstrated to be economically viable, therefore Indicated Mineral Resources within the pit designs have been converted to Probable Reserves. All Inferred material was classified as waste and scheduled to the appropriate waste rock storage facility (WRSF).

The Mineral Reserve estimates are reported using the 2014 CIM Definition Standards. The QP for the estimate is Mr. Jeffrey Choquette P.E., of Hard Rock Consulting. Mineral Reserves are reported at the point of delivery to the process plant and have an effective date of 31 July, 2025. The Mineral Reserves are reported by material type using a $12.50/t NSR cut-off inside the final pit designs.

Gold equivalency (AuEq) grades were calculated using ratios of metal prices and metal recoveries in the following equation.

- AuEq = Au Grade + (((Cu Price in US$/lb * 22.0462 * Cu Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Cu Grade) + (((Ag Price in US$/g * Ag Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Ag Grade). The average overall payables from the smelter and refineries were estimated at 98.8% for gold, 90.1% for silver and 88.2% for copper

The mineral reserve statement is presented in Table 1.5.

Table 1.5 Mineral Reserve Statement

Classification	Material Type	Tonnes (kt)	Grade				Contained Metal			
			Au g/t	Ag g/t	Cu%	AuEq g/t	Gold (koz)	Silver (koz)	Copper (t)	AuEq (koz)
Probable	Oxide	9,198	0.46	18.46	0.08	0.65	137	5,459	7,714	193
Probable	Mix Oxide	4,411	0.42	10.74	0.09	0.64	59	1,524	4,115	91
Probable	Mix Sulfide	38,761	0.50	15.26	0.10	0.80	629	19,020	37,354	995
Probable	Sulfide	39,524	0.53	15.00	0.12	0.78	670	19,064	45,557	997
Probable	**Total**	**91,893**	**0.51**	**15.25**	**0.10**	**0.77**	**1,495**	**45,066**	**94,740**	**2,275**

Notes to accompany Mineral Reserves table:

1. Mineral Reserves are reported at the point of delivery to the process plant, using the 2014 CIM Definition Standards.
2. Mineral Reserves have an effective date of 31 July 2025. The Qualified Person for the estimate is Mr. Jeffrey Choquette, P.E., of Hard Rock Consulting.
3. An NSR cutoff of $12.50/t was used for reporting the Mineral Reserves which is based on estimated processing costs of US$9.10/t, general and administrative costs of US$0.90t, production taxes and royalty costs of US$1.40/t. Metal prices of US$2,200/oz Au, US$26.50/oz Ag, and US$4.00/lb Cu were used in calculating the NSR. In addition, a gold recovery of 74%, a silver recovery of 60% and a copper recovery of 17% were used for Oxide material, a gold recovery of 68%, a silver recovery of 73% and a copper recovery of 62% were used for Mixed Oxide material, a gold recovery of 61%, a silver recovery of 58% and a copper recovery of 73% were used for Mixed Sulfide material and a gold recovery of 53%, a silver recovery of 35% and a copper recovery of 59% were used for Sulfide material in the NSR calculation.
4. Based on the stated metal prices and recoveries the gold equivalent grades were calculated as AuEq = Au Grade + (((Cu Price in US$/lb * 22.0462 * Cu Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Cu Grade) + (((Ag Price in US$/g * Ag Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Ag Grade). The average overall payables from the smelter and refineries were estimated at 98.8% for gold, 90.1% for silver and 88.2% for copper.
5. Mineral Reserves are reported within the ultimate reserve pit design.
6. Tonnage and grade estimates are in metric units.
7. Mineral Reserve tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

1.9 Mining Methods

The CdG deposit contains mineralization at or near the surface that is ideal for open pit mining methods. The mine plan includes a conventional truck and loader open pit operation. The major production unit operations will include drilling, blasting, loading, hauling, and dumping. These activities are planned to be completed with a contractor fleet.

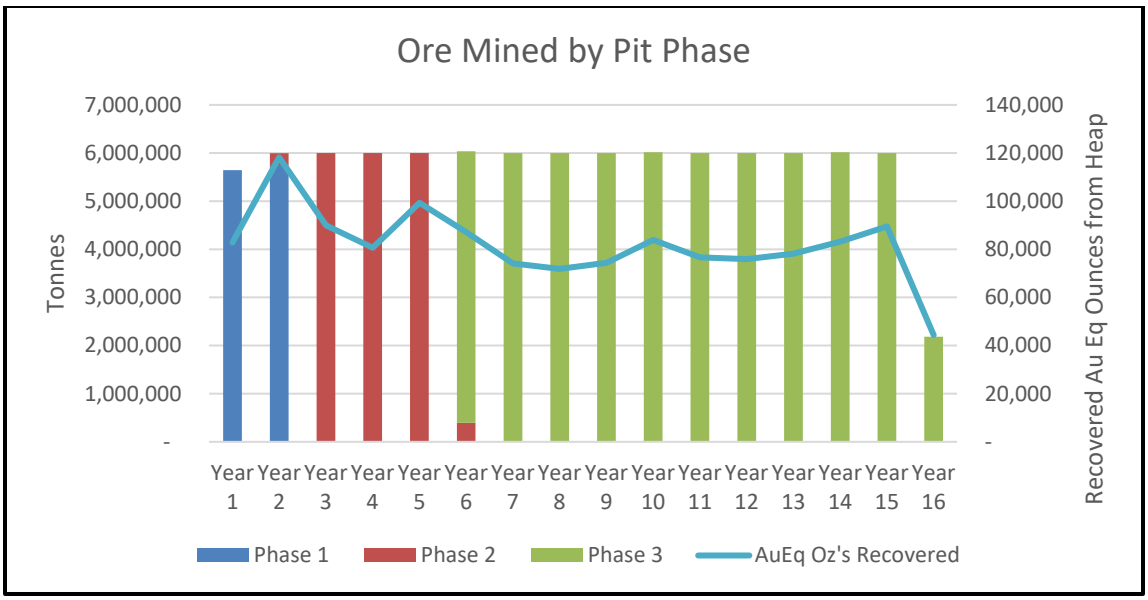
The processing throughput for the Project is 6 million tonnes of ore per year after the second year, with the first year at 5.6 million tonnes. The mine plan is based on Probable Mineral Reserves only. A total of 91.9 Mt will be processed during the 15.3-year mine life

with an overall LOM plan strip ratio of 0.32:1. A total of 29.3 Mt will be sent to the WRSF located east of the pit and north of the heap leach pad. The WRSF has sufficient capacity for the LOM plan.

The final pit design was limited to a US$1,650/oz AuEq pit shell due to the currently designed heap leach pad capacity. The pit slopes were designed by geotechnical sector which are shaped by complex geological and structural conditions. The deposit consists of a barren granodiorite core capped with silica, surrounded by welded ash flow tuff and volcaniclastic sediments primarily composed of shale. Mineralization occurs mainly in a ring around the granodiorite, which results in a configuration of three individual pit bottoms and seven design sectors distributed across three structural domains. The pit was designed into three phases for the Mineral Reserve evaluation with the first phase targeting the higher-grade ore on the southwest side of the deposit and the second phase targeted a high-grade zone on the northwest side of the deposit.

Production of mineralized material from the open pits is driven by the nominal ore crusher capacity rate of 16,667 t/d, which is equivalent to 6.0 Mt/a. During the mining of pit ore, the peak mineralized material and waste production is capped at approximately 11.5 Mt/a, with an overall average production rate of 22,775 t/d. Stripping ratios are very low, however during years 6 and 7 the strip ratios do rise to 0.86 and 0.75 respectively. Reserves were categorized using oxide, mixed oxide, mixed sulfide, and fresh material categories. The mine plan is scheduled in phases to provide the highest valued ore to the leach pad in the early years. The annual ore production schedule by pit phase is shown in **Error! Reference source not found.**.

Figure 1.2: Annual Ore Schedule Forecast



Note: Figure prepared by Hard Rock Consulting, 2025

1.10 Heap Leach Recovery Methods

Test work results have indicated that the CdG ore is amenable to heap leaching for the recovery of gold, silver and copper.

The CdG ore contains cyanide soluble copper that results in high cyanide consumption during the heap leaching process. A SART plant is included that releases cyanide associated with the copper-cyanide complex, allowing a significant portion of it to be recycled back to the leach process as free cyanide, and produces a copper-silver precipitate that can be sold.

The ore will be mined by standard open-pit mining methods, fine crushed using a 3-stage system incorporating jaw, cone, and high-pressure-grinding-roll (HPGR) crushers, agglomerated with cement and conveyor stacked on the heap leach pad in 8-meter lifts. The pad will be constructed in four phases and will hold approximately 92 million tonnes. The heap leach pad will have a composite liner consisting of clay/geosynthetic clay liner (GCL) and smooth/textured HDPE.

Ore will be single-stage leached with a dilute cyanide solution at a high solution application rate for the first 40 days and a lower application rate for the remaining 80 days for a total leach cycle of 120 days. The gold, silver, and copper bearing solution will be collected in the pregnant solution pond and pumped to the SART plant. Pregnant solution will be acidified with sulfuric acid, then copper and silver will be precipitated as sulfides by the addition of sodium sulfide. The precipitate will be thickened and filtered to produce a copper-silver filter cake for shipment to a smelter.

The barren solution from the SART plant will be processed in a carbon adsorption-desorption-recovery (ADR) plant to recover gold. The gold will be periodically stripped from the carbon using a desorption process. The gold will be plated on stainless steel cathodes, removed by washing, filtered, dried and then smelted to produce a doré bar.

Based on the total ore reserve of approximately 92 million tonnes and an established processing rate of 16,667 tonnes per day (first year at 5.6 million tonnes, and 6 million tonnes per year thereafter), the project has an estimated life of about 15.5 years.

1.11 Infrastructure

Existing infrastructure for CdG includes a site office, dirt and gravel roads, and limited power line throughout the Project site. Internet and cellular communications are currently available, though these systems will need to be expanded for operations.

Access to the Project site is by the paved Mexican Highway 110, then a secondary dirt road that goes to San Antón de las Minas community. A private road will enter into the mine property approximately 2 km before reaching the San Antón de las Minas community. This road will provide access to the administration offices, mine, process plant and other Project facilities. An existing community road will be relocated. Internal site roads will be established to serve as mine haul roads, service roads and in-plant roads which connect the facilities for access purposes

1.11.1 Power

Power supply to the CdG Project will initially be generated by on site natural gas generator units operating, with an additional unit on standby. It is assumed that in Year 1 of operations, power supply from a 115KV power line will be available by connecting to the national grid and power generation at site will no longer be needed. Two of the temporary generators and their associated fuel tanks will remain at the project to be utilized as emergency power backup for the process plants.

1.11.2 Water

The project will require water supply for the following uses:

- Mining operations for dust control, drilling, etc.;
- Crushing for dust control;
- Makeup water for the heap leach;
- Process plant and laboratory;
- Modular offices and other site facilities.

The Company has a permitted and active water concession volume of 640,000 m^3/year. The project is expected to consume an annual average of 628,000 – 900,000 m^3 per year, depending on the amount of leach pad in the circuit. A hydrogeologic study is planned to support environmental permitting and water supply needs.

Solution from the heap leach pad will drain to a pregnant solution pond, where it will be pumped through the processing facility to recover precious metals and then pumped back to the leach pad in a continuous cycle. An emergency event solution pond will be located adjacent to the pregnant solution pond to allow containment of excess process solution during precipitation events, which will add additional water to the contained system. A satellite pond will also be included where excess water can be transferred for containment. Process water requirements are first met by pumping collected waters from the emergency event ponds; after that resource is exhausted, make-up requirements will be met by well water. Raw water for the project will be pumped directly from the water

wells to raw water tanks located next to the administration area and a secondary water tank located near the crushing area.

Potable water will be bottled and delivered to the project site.

The raw water tank located near the administration area will be a dual-purpose tank. A portion of this tank will be designated for fire water use.

1.11.3 Project Buildings

The project facilities will be supplied in the form of a modular office and shipping containers will be used for warehouse storage. The refinery will be constructed with masonry walls and a structural steel roof while the motor control centers (MCC) are assumed to be modified shipping containers. Prefabricated steel buildings will be used for the administration building, laboratory, guard house, clinic, dining room, and a training room.

1.11.4 Security

Access to the project will be limited by perimeter fencing around the entire site. A guardhouse at the primary entry point to the project will serve as a security check point that will be manned 24 hours per day, 7 days a week for identification control, random checks, drug and alcohol monitoring and vehicle check-in/out. A security contractor will be used for general site security and protection of mine assets.

1.12 Waste Disposal

1.12.1 Sewage

Wastewater and sewage will be handled by subsurface local septic tanks and centralized leach-fields.

1.12.2 Solid Waste

Solid waste will be disposed of in a manner complying with local regulations.

1.13 Environmental Studies, Permitting and Social Impact

The Cerro del Gallo Project (Project) is in a rural, historic mining district and near the historic San Antón de las Minas underground mine. Mexico has established environmental laws and regulations that apply to the development, construction, operation and closure of mining projects. Of particular importance are the air, surface water, groundwater and mining wastes monitoring programs. The Project will require

various authorizations, licenses and permits for Project construction, operation, and closure. Comprehensive environmental and socioeconomic baseline studies will be required to support permit applications.

Environmental baseline studies carried out by the previous owner indicated that the Project is in a region of high flora biodiversity. Surface water quality generally meets Mexican permissible limits for discharge standards. Groundwater quality does not meet potable water quality standards, and elevated arsenic and lead, possibly associated with mineralization, were noted in some water samples. The Qualified Person (QP) notes that Mexico does not have water quality standards for naturally-occurring groundwater; the standards are related to discharge to water bodies and drinking water usage. There are no established protected areas within the Project or in the immediate area, nor any Indigenous rightsholders; however, a determination is pending as to whether there is a federally protected drainage.

The mineralization for the Project includes sulfide minerals that have the potential to produce acidic conditions and metals leaching. Geochemistry studies of the future mining wastes (that is, spent ore and waste rock) have been carried out based on static and kinetic testing to characterize potential environmental impacts during operations and post-closure. The results indicated that most of the waste rock and all of the spent ore will be classified as a hazardous waste and require special handling based on Mexican regulations for mining wastes. Leaching of some metals may be problematic related to some waste rock lithologies and the spent ore may discharge elevated concentrations of arsenic, cadmium, and selenium. During operations, leachate from the heap leach facility (HLF) will be managed with a leachate collection system. The waste rock storage facility (WRSF) design includes a seepage collection system. Options to manage leachate and seepage, particularly for the post-closure period, will be evaluated as the Project advances.

An environmental permitting process was started by a previous owner, but has lapsed. The Company plans to restart the permitting process for the construction of the Project with a new application. The Company plans to use most of the existing data, but new data will also be collected to support the permit application to construct the Project. There are multiple engineering and design factors that will need to be developed to mitigate potential environmental impacts, particularly associated with the management of potential acid generation and metals leaching from the waste rock.

No known factors to preclude a successful permitting effort have been identified; however, the permitting process in Mexico has been significantly delayed under the current political administration. The length and the effort of the permitting process can be difficult to predict

due to the political regime in Mexico. In 2023, Mexican laws were revised regarding the mining sector. A proposed bill to ban open-pit mining is under review by the government. This Technical Report reflects the current understanding of the laws that affect the Company, including environmental permitting and compliance. The QP notes that the current and revised laws are subject to ongoing interpretation and that in many instances the revised laws require associated regulations that have not been promulgated.

The closure plan is at a conceptual stage and includes a variety of assumptions. A conceptual closure scenario has been developed for future mining operations based on the Life of Mine plan. The estimated closure cost is about US$132M. Closure costs could increase as closure planning advances.

The Company has initiated a social program with a variety of activities in 2025 to develop liaises with the communities and government. The local community is generally in favor of the Project. The Project is in a Mexican state that has water shortages, and there may be scrutiny on water quality and availability from interested parties within and outside of the local community, in particular, from the expat community and nongovernmental agencies in San Miguel de Allende. The permitting process could be impacted by these interested parties as well as by delays in permitting by the Mexican environmental agency.

No material issues related to the environmental, permitting or social aspects of the Project are noted by the QP.

1.14 Capital and Operating Costs

Capital and operating costs for the mining, process and general and administration components of the CdG Project were estimated by KCA and HRC with significant input from Heliostar. Costs for the mining components are based on contract mining. The estimated costs are considered to have an accuracy of +/-20%. Pre-production capital costs required for the CdG Project by area are presented in

Table 1.6 and are expressed in 3rd quarter 2025 US dollars.

Table 1.6 Summary of Pre-Production Capital Costs by Area

Plant Totals Costs	Total Supply Cost US$	Install US$	Grand Total US$
Total Direct Costs	**$97,620,900**	**$28,176,965**	**$125,797,865**
Sub Total with Spare Parts			**$131,466,306**
Contingency	$22,057,581		$22,282,581
Total Direct Costs with Contingency			**$153,748,887**
Mining Costs			**$1,400,000**
Indirect Costs			**$6,465,600**
Other Owner's Costs			**$3,549,690**
EPCM			**$13,817,150**
Sub Total Costs Pre-Production			**$178,981,327**
Initial Fills			**$869,747**
Working Capital (60 days)			**$15,630,000**
TOTAL COSTS (excluding IVA)			**$195,481,074**

Source: KCA (2025)

The total Life of Mine (LOM) capital cost for the Project is US$355.6 million, excluding IVA (value added tax) or other taxes; all IVA is applied to all capital costs at 16% and is assumed to be fully refundable. Table 1.7 the capital requirements for the CdG Project.

Table 1.7 Capital Cost Summary - LOM

Description	Cost (US$M)
Pre-Production Capital	$ 178.9
Initial Fills	$ 0.9
Sustaining Capital – Mine & Process	$ 43.9
Closure	$ 131.9
Total excluding IVA	**$ 355.6**

Source: KCA (2025)

The average life of mine operating cost for the Project is US$13.82 per tonne of ore processed. Table 1.8 presents the LOM operating cost requirements for the CdG Project.

The operating costs presented are based upon the ownership of all process production equipment and site facilities, including the onsite laboratory. The owner will employ and direct all process operations, maintenance and support personnel for all site activities.

Table 1.8 LOM Operating Cost Summary

Description	LOM Cost (US$/t Processed)
Mine	$ 3.79
Process & Support Services	$ 9.12
Site G & A	$ 0.91
Total	**$13.82**

Source: KCA (2025)

The operating costs were estimated based on 3rd quarter 2025 US dollars and are presented with no added contingency based upon the design and operating criteria described in this report. IVA is not included in the operating cost estimate.

Closure and reclamation will include chemically stabilizing the heap and waste rock storage facility, physically stabilizing these facilities, control of surface waters and removal or re-purposing of all other project facilities. The open pit is expected to stay dry. It is assumed that low permeability material will be placed over the heap and waste rock storage facilities.

1.15 Economic Analysis

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates;
- Assumed commodity prices and exchange rates;
- Proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution;
- Sustaining costs and proposed operating costs;
- Assumptions as to closure costs and closure requirements;
- Assumptions as to environmental, permitting and social risks.
- Additional risks to the forward-looking information include:
- Changes to costs of production from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade or recovery rates;

- Geotechnical or hydrogeological considerations during mining being different from what was assumed;
- Failure of plant, equipment or processes to operate as anticipated;
- Ability to maintain the social license to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates;
- Changes to tax rates.

The Project has been evaluated using a constant US dollar, after-tax discounted cashflow methodology based on a 5% discount rate. For personnel costs and material sourced in Mexico, an exchange rate of 18.7 pesos per US dollar was assumed. This valuation method requires projecting material balances estimated from operations and calculating economic analysis. Cashflows are calculated from sales of metal, less cash outflows such as operating costs, capital costs, working capital changes, royalties, any applicable taxes and reclamation costs. Resulting annual cash flows are used to calculate the net present value (NPV) and internal rate of return (IRR) of the Project. Tax calculations involve complex variables that can only be accurately determined during operations, and as such, the actual post-tax results may differ from those estimated.

The economic analysis was performed assuming a base case gold selling price of US$2,300/oz. Gold metal prices are elevated in the economic analysis to reflect current metal price trends. The base case silver selling price is assumed to be US$27/oz and the base case copper price is assumed to be $4.00/lb.

No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- No pre-production period is planned as mineralized material outcrops at surface with very little waste stripping required, a 0.08:1 strip ratio is planned for year 1;
- The production life is forecast at 15.3 years;
- Cost estimates are in constant Q3 2025 US dollars for capital and operating costs, with no inflation or escalation factors considered;
- Results are based on 100% ownership with a 1% government NSR on revenue from gold and silver production;
- Capital costs are funded with 100% equity (no financing assumed);
- All cash flows are discounted to the start of the construction period using a mid-period discounting convention;
- All metal products will be sold in the same year they are produced;

- Project revenue will be derived from the sale of gold and silver doré and a copper and silver precipitate from the SART plant.

The Project was assumed to be subject to the following tax regimes:

- The Mexican corporate income tax system (Federal Income Tax) consists of 30% income tax. Federal income tax is applied on Project income after deductions of eligible expenses including depreciation of assets, earthworks and indirect construction costs, exploration costs, special mining tax, extraordinary mining duty and any losses carried forward;
- Mining tax in Mexico (Special Mining Tax) consists of 8.5% on earnings before interest, taxes, depreciation, and amortization. The special mining duty is applied on Project income after deduction of eligible exploration, earthworks and indirect costs expenses. Income subject to the special mining tax does not allow deductions for depreciation or allow losses carried forward.

At the assumed metal prices, total payments are estimated to be US$442.8 M over the LOM.

For the economic model, value-added tax is not considered in the capital or operating cost estimate as it is assumed that value-added tax paid will be credited in the year that it occurs (net zero impact).

Royalties payable for the CdG include a 1.0% royalty due to the Mexican government as an “Extraordinary Mining Duty”. The 1.0% extraordinary mining duty represents US$25.5 M over the LOM and is included in the Project economics. Royalties are also payable to current claim owners which average 3.7% for the current reserve. The mining claims royalty represents US$108.3 M over the LOM and is also included in the Project economics.

The financial analysis for the Project shows an after-tax net present value at a discount rate of 5% of US$423.9 M, an after-tax internal rate of return of 33.1%, and a payback period of 2.28 years.

Table 1.9 summarizes the projected cashflow; net present value at varying rates; internal rate of return; years of positive cash flows to repay the negative cash flow (payback period); multiple of positive cash flows compared to the maximum negative cash flow (payback multiple) for the project on both an after-tax and before-tax basis.

The projected total lifespan of the Project is 15.3 years. Approximately 1.496 Moz of gold, 45.067 Moz of silver and 94.7 kt of copper are projected to be mined, with 0.888 Moz gold, 22.208 Moz of silver and 58.5kt of copper recovered and produced for sale.

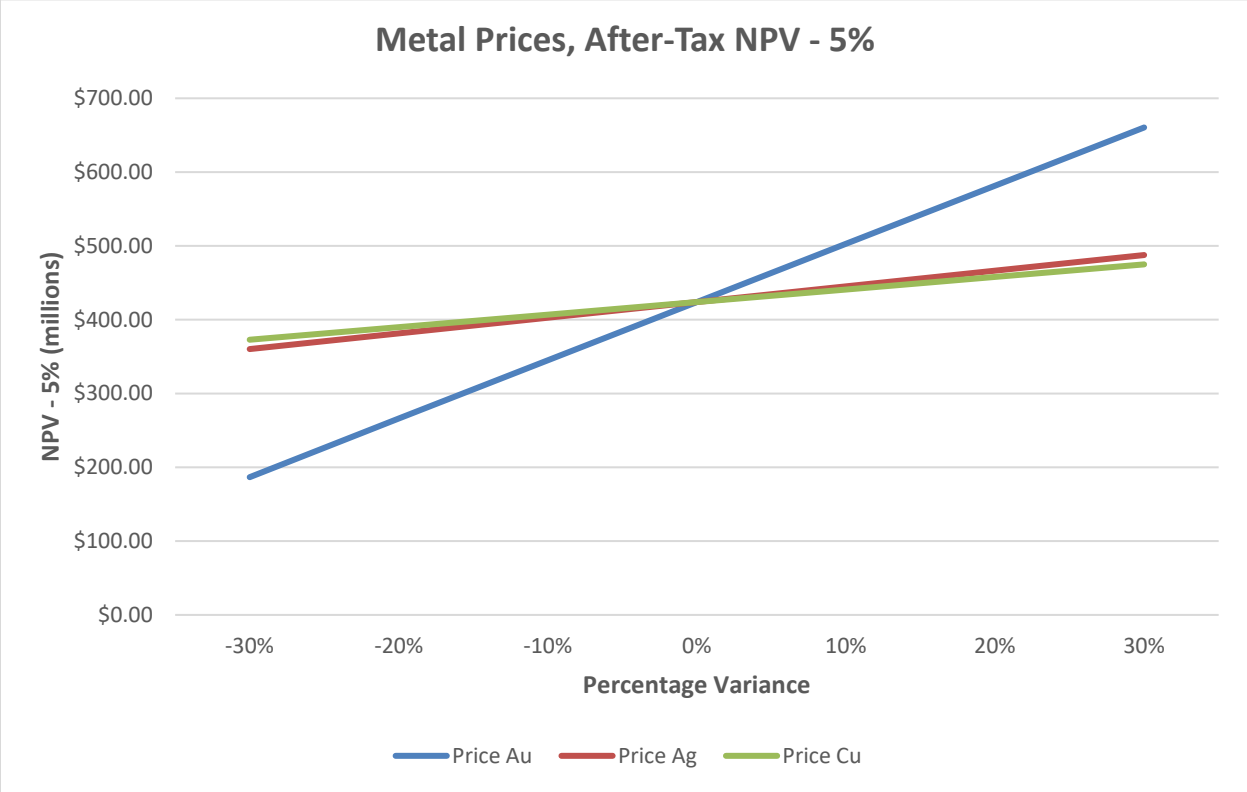
Table 1.9: Summary Economic Results

Project Valuation Overview	Units	After Tax	Before Tax
Total cashflow	US$ M	724.08	1,166.86
NPV @ 5.0% (base case)	***US$ M***	***423.92***	***699.38***
NPV @ 7.5%;	US$ M	326.76	549.45
NPV @ 10.0%;	US$ M	252.23	434.91
Internal rate of return	***%***	***33.1***	***44.9***
Payback period	Years	2.28	1.78
Payback multiple		4.51	6.72
Total initial capital	US$ M	195.48	195.48

A sensitivity analysis was performed on the base case to determine Project sensitivity to gold, silver and copper prices and grade, operating costs and capital costs.

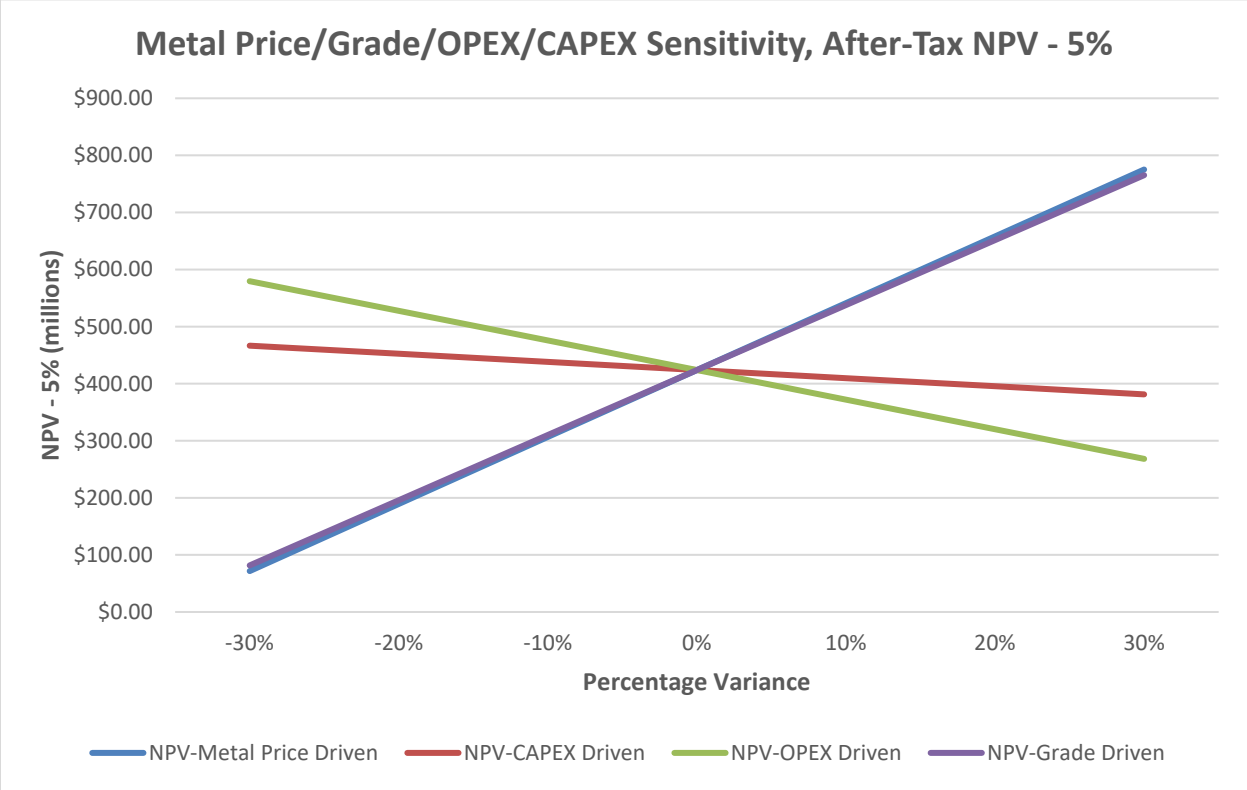
The Project is most sensitive to changes in the gold price and grade. It is less sensitive to the silver and copper prices as they only represent approximately 17.9% and 15.1% of the gross revenue respectively. It is also less sensitive to operating cost changes, and least sensitive to changes in capital costs as illustrated in Figure 1.3 and Figure 1.4.

Figure 1.3: Metal Price Sensitivity Analysis



Note: Figure prepared by Hard Rock Consulting, 2025.

Figure 1.4: Project Metal Price, Grade, Operating Cost and Capital Cost Sensitivity Analysis



1.16 Conclusions

The work that has been completed to date has demonstrated that the CdG open pit mine and heap leach facility is a technically feasible and economically viable Project. It is the conclusion of the QP's that the work completed in preparation of this technical report included adequate detail and information to declare Mineral Reserves. Standard industry practices, equipment and design methods were used in the PFS.

- The base case for the Project has been developed with sufficient detail to underpin a decision to continue to move the Project through subsequent stages of development.

- The production schedule targeted a consistent total mine tonnage of approximately six million tonnes per year after the first year. The first year is approximately five and a half million tonnes. At this rate the expected mine life will be about 15.3 years.

- The mine life and total tonnes processed are limited by land constraints for the heap leach and waste dump facilities and metallurgical testing at depth.

- For the four material types, results of the metallurgical testing show average gold recoveries in the range of 58% to 74%, silver recoveries range from 40% to 79%, and copper recoveries range from 22% to 59%.

- Tertiary crushing with two stages of conventional crushing and one tertiary stage HPGR crushing is used to produce the final crush size of 80% passing 4 to 6 mm.

- Based on the production schedule and field adjusted recovery for the four material types projected gold, silver and copper recoveries are 60%, 52% and 43%, respectively.

- Copper and silver will be recovered in a SART plant by sulfide precipitation; gold and some silver will be recovered by conventional carbon adsorption.

- During operations, control of cyanide levels and careful monitoring of copper in pregnant and barren leach solutions will be required to minimize cyanide consumption and maximize gold and silver recoveries.

- Cement for agglomeration constitutes a significant portion of the operating costs. Careful monitoring of heap permeability and ongoing test programs at site will be required.

- The CdG Project requires supplies and infrastructure items such as administration buildings, laboratory, warehouse, roads, powerline and MCC, water wells and water line, reagent storage, a complete service truck shop; and dining area.

- The economic analysis indicates that the profitability of the potential operation will be driven by gold price, operating costs and capital costs. Given the lower grade nature of the deposit and the strip ratio, well over half of the revenues are consumed by the operating costs. Therefore, a focus on controlling costs will be important in maintaining robust project economics.

- The PFS design of the heap leach facility (HLF) consists of a geomembrane lined pad and process ponds and is intended to operate as a zero-discharge system to groundwater and surface water. The HLF is designed with a geomembrane lined satellite pond that is available to store solution if necessary, during above average climate conditions.

- The HLF provides suitable capacity to store and leach the 92 Mt of ore identified through this study. The ultimate HLF configuration is geotechnically stable and all stability sections analyzed meet the minimum factors of safety shown in the design criteria.

- For the planned HLF design, the 92 Mt of ore storage capacity requires a maximum heap height of 80 meters. The heap leach pad, as conceptually designed, is at its limit with respect to storage capacity for the proposed site, constrained by surrounding topography and the planned location of the proposed pit and the waste rock disposal area (WRD). If additional ore is identified, beyond the 92 Mt, it will be difficult and likely economically prohibitive to increase the capacity of this leach pad.

- The PFS design of the WRD has an excess capacity to store 54.4 Mt of waste rock with only 29.3Mt being planned to be stored in the facility for this study. An additional 3.4 Mt of waste rock is planned for use as structural fills for construction of the HLF. The ultimate WRD configuration is geotechnically stable and all stability sections analyzed meet the minimum factors of safety.

- For the planned WRD design, the 55 Mt of waste rock storage capacity requires a maximum waste rock dump height of 100 meters. The WRD, as conceptually designed, is at its limit with respect to storage capacity for the proposed site, constrained by the property boundary to the north and east, the pit to the west, and the HLF to the south.

- The WRD, as designed, is progressively reclaimed during operations to reduce the amount of contact water that may need to be managed during operations. The HLF as designed is rinsed in an effort to remove unwanted constituents from the ore, which may include metals and cyanide prior to receiving the closure cover. The HLF and WRD closure cover system will be selected to limit the infiltration as needed to meet the environmental compliance criteria.

1.17 Recommendations

The following recommendations have been made to advance the CdG Project:

- A feasibility study should be conducted if the Owner wants to reduce the outlined risks and increase the accuracies of the cost estimates.

- A detailed geotechnical characterization of the available clays within the property limits of the CDG mine should be completed to evaluate if there are sufficient clayey materials on site to be used as the low permeable soil liner bedding beneath the leach pad and ponds geomembrane liner.

- The close proximity of the proposed HLF to the proposed pit may pose a stability risk of the pit walls to the Phase 3 heap leach pad expansion as it is currently proposed. Further stability analyses will be needed to determine if the risk is acceptable or if further stabilization efforts will be needed.

- If the geochemical investigation identified ARD (acid rock drainage) as a potential issue, then it is recommended that in-situ infiltration testing throughout the WRD foundation is performed to characterize the permeability of the foundation soils.

- The 2014 PH Consultores climate study should be updated.

- To facilitate feasibility and detailed design of the HLF and WRD, additional geotechnical field investigation and laboratory testing programs are recommended.

- Further consideration should be given to evaluating the hydrogeologic conditions that exist at the open pit, HLF and WRD sites.

- The hydrogeologic consultant, IDEAS, recommends a detailed groundwater investigation to determine suitable locations for additional water wells.

- Monitoring of existing stream flows should be considered to measure sediment transportation in existing streams.

- Additional effort should be made in developing a feasible, yet proactive reclamation plan for the HLF and WRD.

- Develop a trade-off study for the closure cover system to identify the optimal cover system with respect to cost and functionality.

- Conduct additional SART tests to optimize reagent requirements and better define penalty elements associated with the final product.

- Conduct additional static and kinetic acid base accounting tests to better define the rate of the contained sulfides in both the ore and waste to turn acidic.

Discussions and costs on the recommendations are presented in Section 26.

2 Introduction and Terms of Reference

2.1 Technical Report Preparation

Heliostar Metals (Heliostar) commissioned Kappes, Cassiday & Associates (KCA) to prepare a Pre-Feasibility Study for the Cerro del Gallo Project. This Technical Report has been prepared in accordance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' current "Standards of Disclosure for Mineral Projects" under the provisions of National Instrument 43-101 (NI 43-101), Companion Policy 43-101 CP and Form 43-101F1 and supersedes a Technical Report prepared by Argonaut dated 31 January 2020 titled "Pre-Feasibility Study NI43-101 Technical Report Cerro del Gallo Heap Leach Project Guanajuato, Mexico".

This Technical Report is issued to Heliostar Metals. Heliostar Metals is listed on the TSX V Exchange (TSX V: HSTR) and holds a 100% interest in the CdG deposit. Heliostar is a producing company with two gold and silver mines currently in production in Mexico: San Agustin and La Colorada.

This study commenced during April 2025 and has an effective date of 11 December 2025.

The development option considered for the CdG project is an open pit mine utilizing a three-stage fine crushing circuit with tertiary high pressure grinding roll (HPGR) crushing, conventional heap leaching, sulfidization-acidification-recirculation-thickening for copper and silver recovery, and an adsorption, desorption recovery plant for gold recovery from pregnant heap leach solution.

Heliostar has worked closely with the contributing companies, in addition to being a direct contributor, to develop this Technical Report.

2.2 Qualified Persons and Site Visits

Travis Manning, PE Utah, of KCA was the overall author of the Report and was responsible for processing and infrastructure and associated costs. Mr. Manning is an independent Qualified Person under NI 43-101. Mr. Manning visited the site on 23 July 2025. During the visit, Mr. Manning inspected drill core, reviewed locations of process and infrastructure facilities and met with local personnel to discuss the Project.

Todd Minard, P.E. (Nevada and California), of AECOM is responsible for the design of the heap leach facility (HLF) and waste rock dump (WRD). Mr. Minard visited the site in February 2014, October 2019 and 23 July 2025. While on site Mr. Minard conducted visual examination of the HLF and WRD ground conditions and identified candidate liner bedding sources that will be evaluated in future geotechnical investigation.

Dr. Marvin Silva, Ph.D., P.E., P.Eng.(Alberta and BC), of Tailings Heap Solutions LLC, is responsible for the pit slope geotechnical assessment, including review of geotechnical data and interpretation relevant to open pit slope design and stability. Dr. Silva is an independent Qualified Person under NI 43-101. Dr. Silva visited the site on 23 July 2025. During the site visit, Dr. Silva inspected representative rock outcrops and pit wall exposures where accessible, reviewed drill core and available geotechnical logging records, and met with site personnel to discuss geological and geotechnical conditions, drilling and sampling procedures, and pit slope performance considerations.

Dr. Ted Eggleston, Ph.D., RM SME, reviewed data collection including drilling, surveying, geological logging, geotechnical logging, sampling, sample preparation and analysis, QA/QC, data storage, and geological interpretations. Dr. Eggleston is an independent Qualified Person under NI 43-101. He visited the site from June 18 to June 20, 2025. During that site visit, Dr. Eggleston reviewed core, drill sites (including collar orientations), collected data from the records at the exploration office that were not available elsewhere, and reviewed the exploration work done, and interpretations, with company personnel.

Timothy O. Kuhl, RM SME, constructed the geological model and completed the mineral resource estimation. Mr. Kuhl is an independent Qualified Person under NI 43-101. He visited the site from June 18 to June 20, 2025. During that site visit, Mr. Kuhl reviewed core, drill sites, reviewed records at the exploration office that were not available elsewhere, and reviewed and interpretations with company personnel.

Jeffery Choquette, P.E. Montana, of HRC is responsible for the Mineral Reserves, mining methods and economic analysis. Mr. Choquette is an independent Qualified Person under NI 43-101. Mr. Choquette visited the site on 23 July 2025. During the visit, Mr. Choquette inspected drill core, reviewed locations of open pit, WRSF, process and infrastructure facilities and met with local personnel to discuss the Project.

Dawn Garcia, CPG, of Stantec is responsible for environmental studies, permitting and social impact. Ms. Garcia visited the site during the days of July 23 and 24, 2025. During this time, she met with site personnel, heard a presentation by Company staff regarding the Project, viewed the site from an overlook, visited the coreshack and saw examples of

core, visited the local communities, saw the water supply well for the community of San Anton de las Minas, and visited the city of Dolores Hidalgo.

The effective date of the Mineral Reserve is July 31, 2025. The effective date of this Technical Report is December 11, 2025.

2.3 Units of Measure

All currency amounts in this Technical Report are stated in terms of U.S. dollars (US$) unless otherwise stated. Units of measure are metric, unless otherwise specified. The costs were estimated based on quotes and cost data as of 4th quarter 2025.

Units of measure and abbreviations that may occur in this Technical Report are summarized in .

Table 2.1 Cerro del Gallo Units of Measure and Abbreviations

Abbreviation	Description
ALS	ALS Chemex laboratory
AQ	Core diameter (usually ~ 2.7 cm diameter)
Au	Gold
AuEq / AuEqV	Gold equivalent
Ag	Silver
BWI	Bond ball mill work index
$Ca(OH)_2$	Calcium hydroxide, hydrated lime
Cdn$	Canadian currency (dollars)
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
cm^3	Cubic centimeter
cm^2/s	Centimeter per second
CV	Coefficient of variation
DDH	Diamond drill hole (core)
g or gms	Gram
G&A	General and administrative
g/L	Grams per liter
g/t or g/mt	Grams per metric tonne
Grd-Thk	Grade times thickness product
Ha	Hectare
HDPE	High-density polyethylene
HQ	Drill core diameter (~ 63.5 mm diameter)
ICP	Inductively coupled plasma analytical method
ICP-AES	Inductively coupled plasma analytical method
ID^2	Inverse distance squared
ID^3	Inverse distance cubed
kg/t or kg/mt	Kilogram per tonne
km	Kilometer
km^2	Square kilometers
kW	Kilowatt
kWh	Kilowatt-hour
lb	Pounds
LLDPE	Linear-low-density polyethylene
$L/Hr/m^2$	Liters per hour per square meter
m	Meter
M	Million
Mμ	Microns
m^2	Square meters
m^3	Cubic meters
m^3/hr	Cubic meters per hour
masl	Meters above sea level
mm	Millimeter
mg	Milligram
MXN	Mexican currency (peso)
NaCN	Sodium cyanide
NN	Nearest neighbor model

Abbreviation	Description
Nm^3/hr	Normal cubic meters per hour
NQ	Drill core diameter (~ 47.6 mm diameter)
NSR	Net smelter return
oz	Troy ounce approximately (31.1035 grams)
PEA	Preliminary economic assessment
ppb	Parts per billion
ppm	Parts per million
ppmv	Parts per million by volume
PQ	Drill core diameter (~ 85.0 mm diameter)
QA/QC	Quality assurance/quality control
QQ	Quantile-quantile plot
RC	Reverse circulation drilling method
RPD	Relative percent difference
RQD	Rock quality designation
SMU	Selective mining unit
scfm	Standard cubic feet per minute
t	Metric tonne (1,000 kg)
Mt	Million tonnes
tpy or t/y	Tonnes per year
t/d	Tonnes per day
t/h	Tonnes per hour
t/m^3	Tonnes per cubic meter
US$ or USD	US currency (dollars)
US$M	US currency (dollars) in millions
UTM	Universal Transverse Mercator
%	Percent

Source: KCA (2025)

3 Reliance on Other Experts

KCA, HRC and Stantec are not experts in Mexican legal, civil, environmental or tax matters. The authors of this report have relied on information from Heliostar Metals pertaining to:

- the assessment of the legal validity of mining claims or concessions; private lands, mineral rights, royalties, property agreements;
- environmental and permitting legal issues;
- social and community impacts of development of the Project;
- guidance on actions and policies needed to ensure that Heliostar obtains and maintains social license to operate the Project; and,
- appropriate tax implications.

4 Property Description and Location

4.1 Location

The CdG Project is wholly owned by Heliostar Metals, a Canadian company with corporate offices in Vancouver, British Columbia, Canada.

The CdG Project is located in the state of Guanajuato, approximately 270 km northwest of Mexico City. The Project is roughly centered on latitude 21° 04’ 28” N, longitude 101° 01’ 38” W and lies mainly within the Municipality of Dolores Hidalgo with some concessions extending westerly into the Guanajuato municipality. The project location is presented in Figure 4.1. The CdG Project lies within the San Antón de las Minas mining district and is only 23 km east-northeast of the historic Guanajuato Mining District which has recorded production of over 1.4 billion ounces of silver and 6.5 million ounces of gold.

Production records for the San Antón district date back to the 1860’s but there was mining activity much earlier during Spanish colonial times with production coming from epithermal vein systems. The district lies within the central-southern segment of the world-class Mexican Gold-Silver Belt.

The CdG Project is located on the Servicio Geológico Mexicano (SGM) and Instituto Nacional de Estadística Geografía e Informática (INEGI) Guanajuato 1:50,000 scale topographic/geologic map (F14-C43), and Guanajuato 1:250,000 scale topographic/geologic sheet 14-7. The coordinate system used for all maps and sections in this report is Universal Transverse Mercator (UTM) WGS84 Zone 14, Northern Hemisphere.

Figure 4.1: Location Map of the Cerro del Gallo Property in Guanajuato



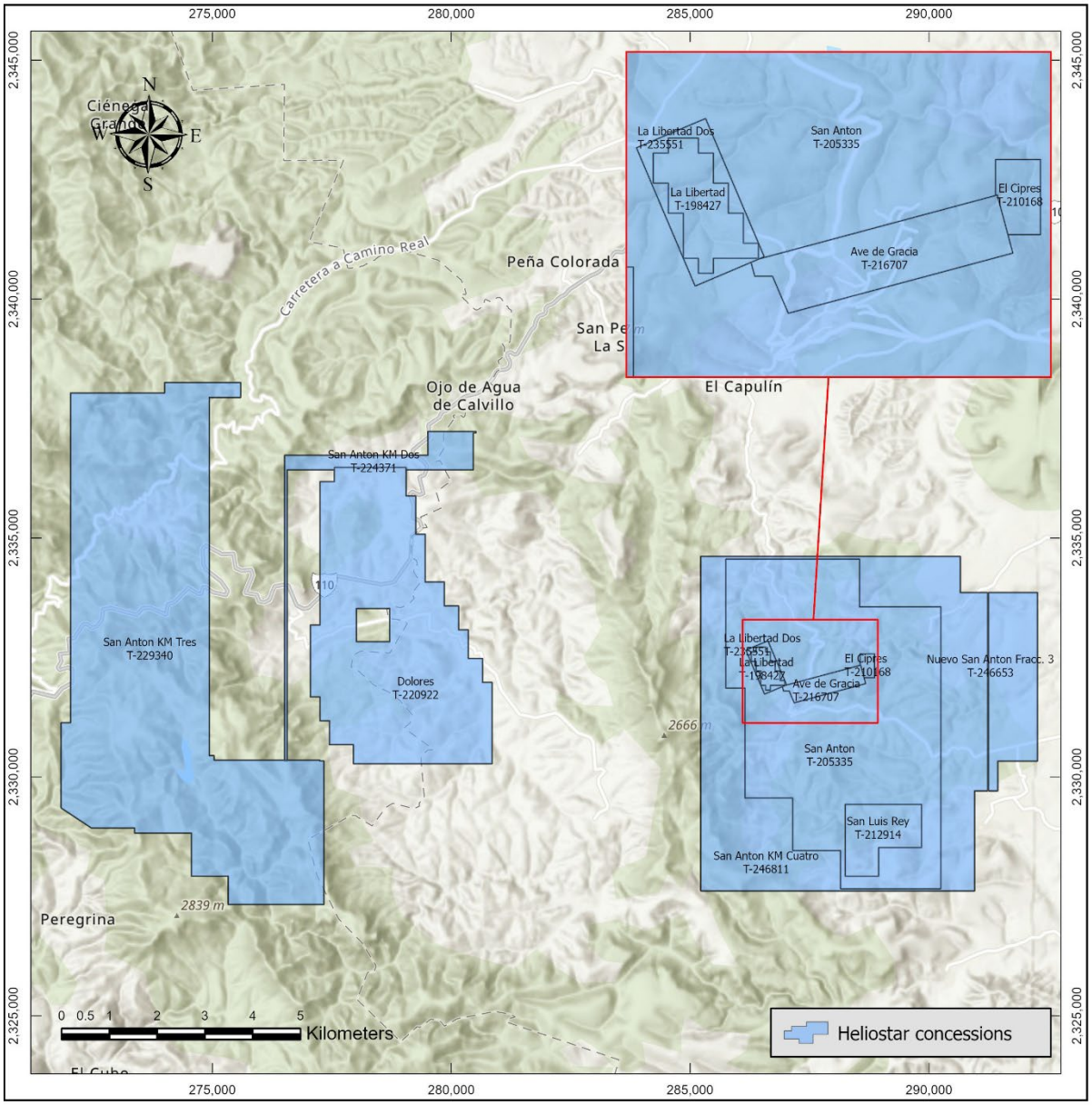
Source: Heliostar (2026)

4.2 Concessions

The following information on the mineral titles are provided by Heliostar Metals. Heliostar currently has title to 15 concessions totaling 16,877.91 Ha, as listed in the following Figure 4.2 and Table 4.1. Of these, 4 concessions are in the process of being withdrawn.

Concession taxes are due each year in the months of January and July. The total holding costs for 2026 translates to approximately US$296,000. Mining concessions in Mexico also have a minimum annual investment which in 2019 at CdG will total approximately US$2,160,000. Taxes and investment requirements are calibrated based on surface area and the age of the concessions.

Figure 4.2: Heliostar Concessions



Source: Heliostar (2026)

Table 4.1 Concession Status

Concession Name	Title Number	Surface (Hectares)	Grant Date	Expiry Date
La Libertad	198427	32	26/11/1993	25/11/2043
San Antón	205335	2,201	08/08/1997	07/08/2047
El Ciprés	210168	14	10/09/1999	09/09/2049
San Luis Rey	212914	186	13/02/2001	12/02/2051
Ave de Gracia	216707	64	17/05/2002	16/05/2052
Dolores	220922	1,665	28/10/2003	13/02/2052
San Antón KM Dos	224371	189	29/04/2005	28/04/2055
San Antón KM Tres	229340	3,462	11/04/2007	10/04/2057
La Libertad Dos	235551	18	19/01/2010	18/01/2060
Nuevo San Antón Fraccion 1*	246651	685	16/10/2018	26/10/2048
Nuevo San Antón Fraccion 2*	246652	135	27/10/1998	26/10/2048
Nuevo San Antón Fraccion 3	246653	372	16/10/2018	26/01/2048
San Antón KM Cuatro	246811	1,602	21/08/2019	20/12/2059
San Antón KM Fraccion 1*	256141	5,869	05/04/2005	04/04/2055
San Antón KM Fraccion 2*	256141	385	05/04/2005	04/04/2055

*** Concession in process of withdrawal**

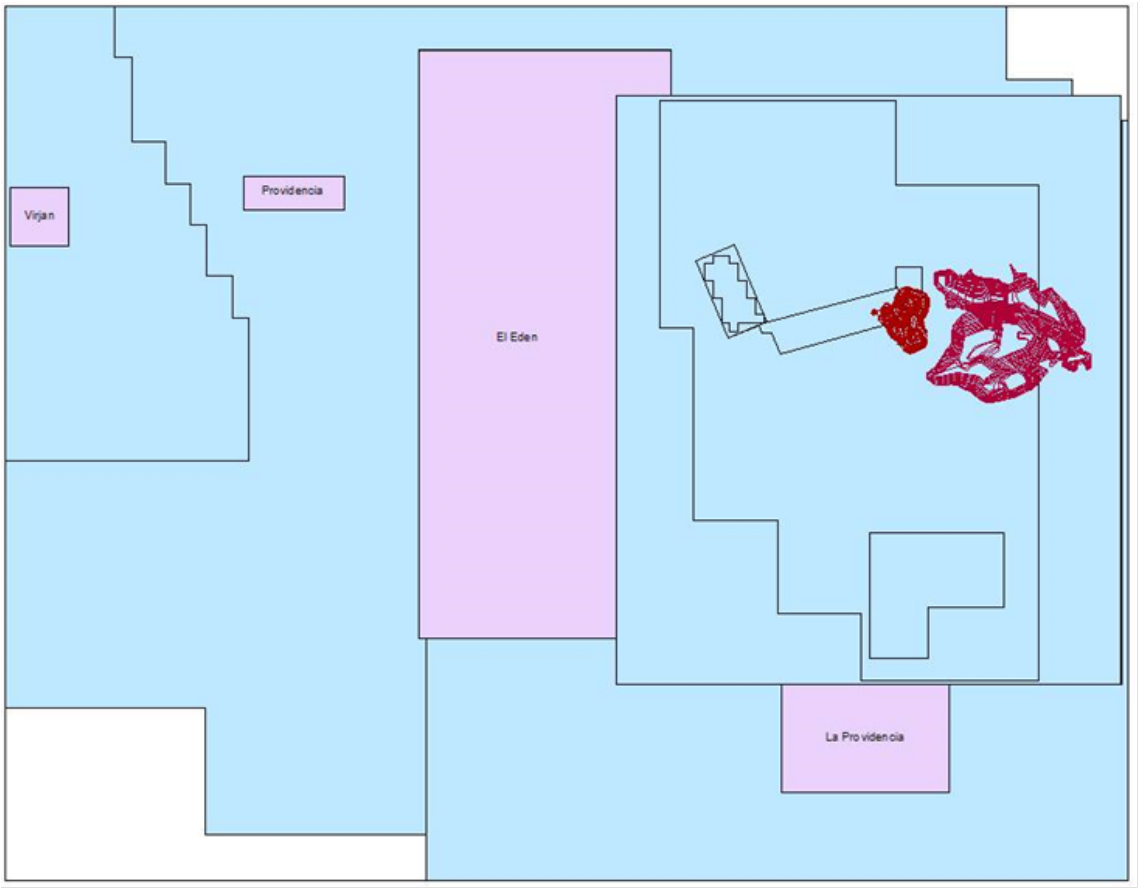
4.3 Inholdings

Within the boundary belonging to San Antón de las Minas S.A. de S.V. (SAM), as presented in Figure 4.3, there exist some small-scale mining claims. In total there are four mining lots, which include:

- El Eden Title 212009 on behalf of the concessionaire Compañía Minera El Cubo SA de C.V.
  - Surface area of 1675.8 hectares
- La Providencia Title 211859 on behalf of the concessionaire Compañía Minera El Cubo SA de C.V.
  - Surface area of 254.6 hectares
- Virgan Title 214424 on behalf of the concessionaire Compañía Minera El Cubo SA de C.V.
  - Surface area of 49 hectares
- Providencia Title 211859 on behalf of the concessionaire Lauro Gonzalez Ibarra
  - Surface area of 48 hectares

Heliostar has not done any prospecting work in any of the adjacent areas.

Figure 4.3: Cerro del Gallo Adjacent Properties



Source: PEA (2020)

4.4 Net Smelter Return Royalties

Some concessions making up the CdG Project are subject to Net Smelter Return (NSR) royalties as follows:

The San Antón concession T-205335 is subject to a 4% NSR royalty to Servicio Geológico Mexicano (SGM).

La Libertad T-198427, Nuevo San Antón T-208424, El Cipres T-210168, Ave De Gracia T-216707 and Dolores (T-220922) concessions are subject to a 3% NSR royalty to Nanjo Royalties S. de R.L. de C.V.

The overall average is calculated to be 3.7% for the reserves outlined in this study.

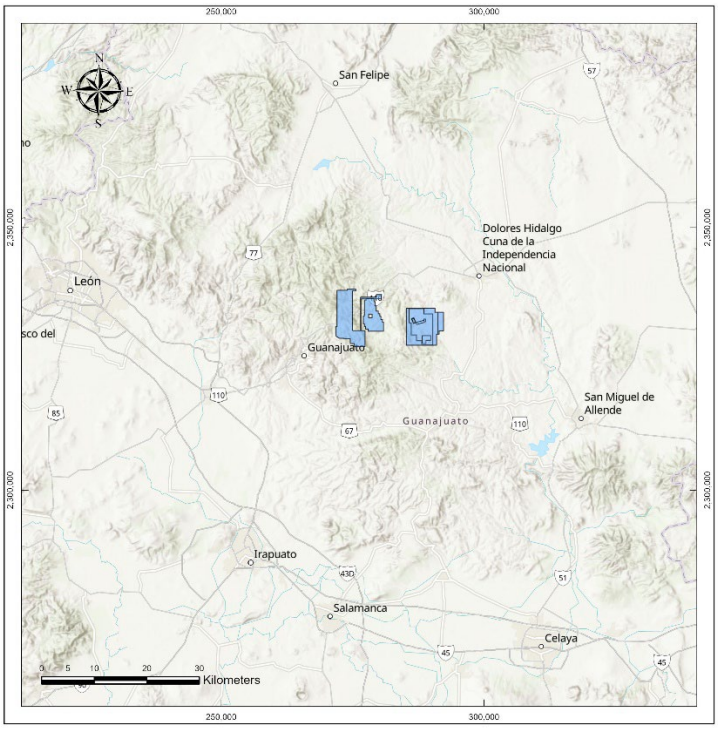
5 Accessibility, Climate, Local Resources and Physiography

5.1 Accessibility

The CdG Project is located in the central region of Guanajuato state, approximately mid-way between the cities of Guanajuato and Dolores Hidalgo (Figure 5.1). The Project lies 23 km east northeast of Guanajuato, the state capital, and 270 km northwest of Mexico City. The Project has year-round unrestricted access provided by a network of well-maintained roads.

The principal access route to the area is by paved Federal Highway 110, then 17 km of well-maintained all-weather gravel road. Travel time from Dolores Hidalgo requires approximately 30 minutes and approximately one hour from Guanajuato. There is an international airport at the city of Leon, which is 30 minutes from Guanajuato.

Figure 5.1: Cerro del Gallo Accessibility



Source: Heliostar (2026)

The CdG Project is located within a region of well-established infrastructure. The property area is well serviced by road, rail, and air services, power and water supplies, and skilled

and semi-skilled local labor. Surrounding cities can provide nearly all of the services required for a major mining operation.

Guanajuato municipality has a population of around 153,000 while Dolores Hidalgo has approximately 135,000 inhabitants. The small community of San Antón de las Minas, which is adjacent to the Project, has a population of nearly 500 and is a reliable source of semi-skilled and unskilled labor to assist in project support.

5.2 Physiography

The CdG Project is situated in the physiographic region known as the Sierra Madre Oriental. Elevations in the area range from 1,800 meters to 2,670 meters, with the top of CdG having an elevation of 2,310 meters. CdG is located along the flank of the San António Mountain Range which trends generally 20° NW. CdG itself is a distinct conical-shaped hill within an otherwise undulating landscape, as seen in Figure 5.2.

Figure 5.2: View looking Southwest at Cerro del Gallo



Source: PEA (2021)

5.3 Climate

The municipality of Dolores Hidalgo is classified as semi-arid. The climate is generally dry with sporadic rainstorms in the summer months. Average annual precipitation is 586 mm mainly from June to September. The monthly average, 1 in 100 wet year, and 1 in 100 dry year precipitation depths are summarized in Table 5.1. The winter months are

cool and dry. Temperatures range from a maximum of 36.5°C in summer to a minimum of 3.8°C in winter. The average yearly temperature is 17.4°C. The mine will be operated year-round.

Table 5.1 Cerro del Gallo Average and Extreme Precipitation

Month	Average Precipitation	1 in 100 Wet Year	1 in 100 Dry Year
Jan	15.00	30.51	0.00
Feb	11.80	24.00	0.00
Mar	8.40	17.09	0.00
Apr	10.80	21.97	0.00
May	33.80	68.75	0.00
Jun	99.60	202.60	0.00
Jul	138.10	280.91	0.00
Aug	108.90	221.52	0.00
Sep	102.80	209.11	0.00
Oct	37.10	75.47	0.00
Nov	10.30	20.95	0.00
Dec	9.60	19.53	0.00
Annual	**586.20 mm**	**1,192.40 mm**	**0.00 mm**

Source: (PHCA 2014)

6 History

6.1 Project History

Within the San Antón de las Minas district (CdG Project) there is ample evidence of widespread prospecting activity and limited underground production, dating back to Colonial times. Little information is available for these early activities. The history during modern times is summarized as follows:

In early 1977, as part of a combined federal and state government program to encourage new investment in the mining industry, the Consejo de Recursos Minerales (CRM, today Servicio Geológico Mexicano), the Mexican equivalent of a Department of Mines, began an assessment of the San Antón de las Minas area. This activity included an assessment of the Carmen-Providencia epithermal vein system and regional geological mapping, stream-sediment geochemistry, prospect evaluation, and airborne magnetics. The CRM were probably the first group to recognize intrusive related disseminated copper and gold mineralization at CdG.

Starting in 1982, the Sociedad Cooperativa Minero Metalúrgica Santa Fe de Guanajuato S.C.L. (Cooperativa) conducted exploration activities in the area with a focus on the districts epithermal vein systems. This included evaluating the Carmen-Providencia vein system where they rehabilitated the Dolores shaft to the 90-meter level, re-opened the La Mora adit, and commenced development of the Carmen adit along a section of the Carmen vein near the Dolores shaft. Reportedly, two truckloads of ore totaling 20.9 tons grading 82 g/t silver and 2.15 g/t gold were mined. In 1983, the Cooperativa conducted the first drilling at CdG with the completion of six core holes for a total of 1,571 meters. This work focused on veins and only select vein intervals were assayed for gold and silver.

In 1994, the Cooperativa sold two small contiguous claim blocks to Luismin S.A. de C.V. (Luismin). These concessions covered portions of CdG and the Ave de Gracia epithermal vein system. Luismin's work included drilling 15 holes at CdG totaling 3,551 meters. Luismin was the first company to focus on the potential for an open pit mining operation at CdG. Subsequently, Luismin acquired five additional mining concessions referred to as La Libertad, Nuevo San Antón, El Cipres, Ave de Gracia and Dolores.

During 2002, Wheaton River Minerals (Wheaton) acquired the Luismin gold/silver properties in Mexico, including the San Antón Project. Two years later, in 2004, Goldcorp

acquired Wheaton which, at the time, held a 49% interest in CdG with San Antón holding 51%.

In 2004, Wheaton entered into a Joint Venture Agreement with the Australian based company, Kings Minerals NL to form the operating company San Antón de las Minas S.A. de C.V. (San Antón). The Agreement gave Kings Minerals the right to earn a 51% interest in the project. This was accomplished by reimbursing Wheaton US$510,000 and expending US$3,000,000 over the first two years of the venture commencing in July of 2004. These expenditures were completed in November 2005, giving Kings Minerals a 51% interest in San Antón. During 2004, Wheaton also purchased the San Antón concession from Cooperativa. Later, after the joint venture with Cerro Resources was formed, an additional five mining concessions were granted, as follows: San Antón KM, San Antón KM Dos, San Antón KM Tres, San Antón KM Cuatro and La Libertad Dos. The San Luis Rey concession was also purchased.

In December of 2006, Kings Minerals completed a business combination agreement with a publicly listed Canadian Company called Andaurex Industries (Andaurex), which resulted in a new entity called San Antón Resources Corporation. San Antón Resources Corporation was subsequently listed on the Toronto Stock Exchange. The main asset of this Company was the San Antón Project. As the result of this combination, Kings Minerals held a 71% interest in the 51% ownership of the San Antón Project under the San Antón JV with Goldcorp. In September of 2010, a business combination was completed with Andaurex whereby Kings Minerals acquired all of the San Antón common shares it did not already own, giving Kings Minerals 100% ownership of its portion of the San Antón Joint Venture. Prior to this, in 2008, the Company's name on the Toronto Exchange was changed to Cerro Resources. At this time, Cerro Resources held a 65.7% interest in the San Antón JV, with the remaining 34.3% owned by Goldcorp. Goldcorp's original 49% interest was diluted to 34.3% due to them electing not to participate in the 2007 and 2008 exploration programs funded 100% by Cerro Resources.

Cerro Resources held the property until 2011 and carried out programs of reverse circulation and core drilling. Several geochemical sampling programs were also completed. While they held the property, Cerro Resources and their predecessor company San Antón completed 280 RC holes totaling 59,595 meters and 74 core holes totaling 36,384 meters.

An early focus of Cerro Resources was exploring several epithermal vein systems in the district. Drilling on these was distributed as follows: 18 holes (4,069 meters) into the Dolores area (Dolores shaft), 26 holes (6,158 meters) into the Empalizada area, and 14 holes (4,337 meters) into the Espiritu Santo area. A further 13 holes (1,768 meters) were

drilled at San Luis Rey. Most holes tested the upper level of the epithermal vein systems to a depth of 150 to 300 meters.

Although Cerro Resources did test some vein systems, their main focus was the evaluation of the CdG copper-gold system. This work included completion of technical reports in 2008, 2011 and 2012. The 2008 report is titled: *Technical Report on the Cerro del Gallo deposit within the San Antón Property Mexico, Prepared by San Antón Resource Corporation Inc. Ottawa, Canada.* This report envisioned a very large operation that included mining the entire copper-gold porphyry system and construction of a flotation mill. The M&I stated resource in this 2008 report totaled 461 million tonnes, grading 0.27 g/t Au, 11 g/t Ag and 0.11% Cu.

The 2011 report titled: *Technical Report, Feasibility Study and Preliminary Assessment, Cerro del Gallo Project, Guanajuato, Mexico* (Feasibility level study on the heap leach and a Preliminary Assessment on a milling scenario), superseded previous works and focused only on the gold and silver resource. The Report envisioned a total open-pit mine-life of 14 years with the first seven years being a heap leach operation followed by construction of a CIL mill to process the un-oxidized mineralized material.

In May of 2012, Cerro Resources completed another report titled: *Technical Report, First Stage Heap Leach Feasibility Study, Cerro del Gallo Gold Silver Project, Guanajuato, Mexico.* This report re- examined the economics of an open pit heap leach operation and did not include the additional study for a milling operation, as was presented in the 2011 Technical Report.

In May of 2013, Primero Mining Corp. (Primero) announced the completion of a transaction with Cerro Resources whereby Primero acquired all of the issued and outstanding ordinary shares of Cerro Resources, thereby taking ownership of Cerro Resources 69.2% interest in the CdG Project. In December of 2013, Primero also announced that it had made arrangements to acquire Goldcorp's then 30.8% interest in the project.

Primero considered CdG as a potential long-life project and, in 2013, announced approximately 1.0 million ounces of gold-equivalent proven and probable reserves and 1.6 million ounces of gold-equivalent measured and indicated resources (exclusive of reserves). The heap leachable proven and probable reserves measured 32.2 million tonnes grading 0.69 g/t gold for 712,000 ounces. This tonnage also contained 15 million ounces of silver grading 14 g/t. Primero's objective was to put the project into production in 2015. In 2013 Primero completed 50 core holes totaling 12,623 meters in the resource area. Of these, 13 were PQ sized twin holes for metallurgical study and 19 were infill

holes. The rest were for condemnation and to test for acid generation. In general, the fill-in drilling confirmed the existing mineralization model and the resource did not materially change. The metallurgical drilling, among other things, identified the complexity of the oxide distribution.

A large part of Primero's focus during their time with the project was to better classify the degree of oxidation through the mixed-zone and predict heap leach recoveries. Primero did not complete a Technical Report, but in 2014 the consulting company BBA was engaged by Primero to complete a fairly extensive metallurgical program and authored a Technical Report titled "Cerro del Gallo Metallurgical Test work 2014." In general, this work assigned different heap leach recoveries depending on ore type, with near surface weathered ore being 76%, high-oxidized mixed-ore at 60% and low-oxidized mixed-ore at 50%. In comparison, a 2012 resource study by Cerro Resources gave all of the mixed ore a gold recovery of 64%.

In August of 2015, Primero engaged JDS Energy and Mining (JDS) to conduct an Open Pit Optimization study utilizing the above recoveries for the various ore types. This work was a relatively conservative look at the previous resource and mine-plan with an emphasis on the strongly oxidized weathered portion of the deposit and select areas of the mixed ore that lies immediately beneath the weathered zone. The JDS study resulted in a drop in grade and contained ounces but did incorporate a more selective mining plan that significantly decreased the strip ratio and lessened the risks of mining less oxidized material with low recoveries.

Primero also went through the exercise of re-logging historic cuttings to better understand the distribution of oxidation and, in the process, divided the mixed oxide zone into highly oxidized (weathered and not part of the mixed zone) strong to medium oxide, weak oxide and fresh.

In addition to advancing the CdG mineral system, Primero evaluated several of the known epithermal vein systems in the district. Their work included detail geological mapping, stream sediments sampling, extensive soil sampling, rock chip sampling and analyses of available geophysical data. They completed 25 drill holes in vein targets totaling 4,218 meters that were distributed as follows: 19 drill holes in the Carmen-Providencia vein system, two drill holes in the Ave de Gracia vein, two drill holes in the Espiritu Santo vein and one hole in the La Paz vein.

In 2015, Primero elected not to put CdG into production and expenditures on the project were significantly reduced. The project became a low priority for the Company with limited activity during 2016 and 2017.

In December of 2017, Argonaut made a cash purchase of the CdG Project. Argonaut recognized the complexities associated with the mixed oxide mineralization and completed a quantitative re-classification (in percentage) of oxide and sulfide material by examining RC drill cuttings contained in historical “chip trays.” This work utilized four oxidation domains with this data transferred to 50 meter spaced vertical cross-sections through the CdG deposit. This led to development of a three-dimensional model, generated by Leapfrog software. In mid-2018, utilizing its updated geologic and oxide distribution model, Argonaut designed and implemented a core drilling program to obtain additional material for metallurgical testing and to better refine its understanding of the mixed oxide mineralization. The program consisted of 18 PQ sized core holes totaling 1,484.5 meters. The core was combined into 11 composites representing both intrusive hosted mineralization and sediment hosted mineralization and made up of varied oxidation states and grade ranges. The material was sent to KCA in Reno, NV for column heap leach testing.

A Technical Report prepared by Argonaut dated 31 January 2020 titled “Pre-Feasibility Study NI43-101 Technical Report Cerro del Gallo Heap Leach Project Guanajuato, Mexico” superseded previous studies and was utilized as a design basis for this study.

In November, 2024, Heliostar acquired the Cerro del Gallo project, as part of a larger portfolio of Mexican assets, from Florida Canyon Gold Inc. (A spin-out of Argonaut Gold).

6.2 Historical Heap Leach Facility and Waste Rock Dump Geotechnical

The following geotechnical aspects of the HLF and WRD sites at the Project site have been characterized in accordance with NOM-155 (Norma Oficial Mexicana).

6.2.1 Site Soil and Bedrock Geotechnical Characterization

6.2.1.1 Geotechnical Investigations

Starting in 2011, three geotechnical subsurface investigations were performed at the CdG site to support various technical studies. Refer to Golder 2019 for test pit and boring logs from the various field explorations. The investigations include:

- 2011: Knight Piésold (KP, 2012a):
  - Excavated 34 test pits (TP-01 through TP-39)
  - Drilled 4 borings (BH-01, BH-02, BH-03, and BH-05)
  - 8 Ripping trials
- 2013: Knight Piésold (KP, 2013):
  - Excavated 40 test pits (KPTP-01 through KPTP-40)

  - Drilled 11 borings (KPBH-01 through KPBH-08, KPBH-08A, KPBH-09, and KPBH-10)
- 2014: Golder Associates (Golder 2014):
  - Excavated 15 test pits (TP-55 through TP-70)

6.2.1.1.1 2011 Test Pits

Test pits were excavated within areas of the proposed leach pad, the solution ponds, and crusher and to achieve general site coverage. Test pits were excavated to depths ranging from approximately 1½ to 7 meters. Subsurface conditions encountered in each test pit were logged and photographed, and representative samples were collected for laboratory testing.

6.2.1.1.2 2011 Geotechnical Borings

Four borings were drilled to depths ranging between 12 meters and 26 meters. Each boring was drilled from the ground surface using HQ-size wireline diamond coring. Core was placed into boxes and geotechnically logged. Rock core samples were taken for laboratory testing to assist in the estimation of rock strength design parameters.

6.2.1.1.3 2011 Ripping Trials

Ripping trials were performed at eight locations to ascertain the rip-ability of the near-surface bedrock. Each trial ran approximately 100 meters in length. In most cases, the ripping went smoothly with full tyne penetration and the bulldozer progressed without appreciable slowing. Ripping slowed somewhat in one location at the western boundary of the HLF (near Test Pit KP-CDG-TP-29) due to the presence of large granitic boulders up to one meter in diameter embedded in a clayey ash matrix.

6.2.1.1.4 2013 Geotechnical Borings

Eleven geotechnical borings were drilled in 2013. Each boring was drilled from the ground surface using HQ size wireline diamond coring, with the exception of KPBH-08A. Boring KPBH-08A was drilled from the ground surface to a depth of 6 meters using a tri-cone rock roller and from 6 meters to 23 meters using diamond coring. Core was placed into core boxes and transported for logging, photographing, sampling and storage.

Standard Penetration Tests (SPTs) were undertaken in four of the borings from between 1½ and 13½ meters depth at typical intervals of 1½ meters.

Upon completion of the drilling program, KP requested that Primero grout the geotechnical borings. No groundwater monitoring installations were fitted to the geotechnical borings.

6.2.1.1.5 2013 Test Pits

In 2013, a total of 40 test pits (KPTP-01 through KPTP-40) were excavated across the plant site areas, heavy vehicle workshop, dam sites and potential borrow areas. The test pits were excavated to depths ranging from ½ meters to about 6 meters with the final depth dependent on the available space, ground hardness and test pit target depth. The subsurface conditions encountered in each test pit were logged and photographed. Representative samples of the subsurface materials were collected for laboratory testing. All test pits were backfilled with excavated spoil on completion of excavation.

6.2.1.1.6 2014 Test Pits

Golder excavated 15 test pits in 2014 within the footprint of the proposed waste dump area and along the perimeter of the WRD. Samples were collected and soil and rock descriptions were prepared. A backhoe was used at 11 of the sites. At four sites, exposures of soil and rock in natural slopes and cut slopes enabled direct observation of soil and rock conditions. Samples were sent to Golder's soil testing laboratory in Houston, Texas for testing.

6.2.1.2 Geotechnical Laboratory Testing

Representative samples of the subsurface materials were obtained from the test pits, borings, and from near-surface bulk sampling. Selected samples were tested between 2011 and 2014 to determine the critical engineering properties of the materials encountered. Samples were shipped to two laboratories; an independent testing laboratory, Segeomex in San Luis, Potosi, or to the Golder laboratory in Houston, Texas. Geotechnical Laboratory testing included the following:

- Natural water content determination (ASTM D2216)
- Atterberg limits determination (ASTM D4318)
- Particle size distribution analysis (ASTM D422/C136)
- Moisture-density relationship [standard Proctor compaction] (ASTM D698)
- Moisture-density relationship [modified Proctor compaction] (ASTM D1557)
- Unconfined compressive strength (UCS) testing of rock (ASTM D2938/7012)
- Unconfined compressive strength testing with measurement of Young's modulus (ASTM D3148)
- Emerson dispersion analysis (ASTM D4229)
- Flexible wall triaxial permeability on remolded soils (ASTM D5084 Method F)
- Point load index testing (ASTM D5731)
- Los Angeles Abrasion testing (ASTM C131)
- Stage-loading permeability testing

Summaries of the above laboratory test results are presented in a December 2019 Golder design report.

6.2.1.3 Borrow Source Laboratory Testing

6.2.1.3.1 Liner Bedding

Remolded permeability tests were performed on samples obtained by Knight Piésold (KP, 2012a) from within the upper colluvium and residual soils within the footprints of the HLF and WRD, as well as from a potential borrow area located approximately 1,800 meters northwest of the WRD. These soils were tested to evaluate their potential for use as a soil liner bedding beneath the leach pad geomembrane liner. Testing of soils sampled from potential off-site borrow areas indicate permeability values of 6.9 x 10^{-5} cm/sec (Silty Sand) to 1.8 x 10^{-5} cm/sec (Clayey Gravel). Laboratory tests performed on native on-site materials indicate permeability values of 2.17 x 10^{-4} cm/sec (Silty Gravel) to 4.64 x 10^{-6} cm/sec (Lean Clay).

Golder performed permeability tests on native materials obtained from the footprint of the WRD at various confining pressures. Two tests were performed at a confining pressure of 69 kPa with results of 5.90 x 10^{-6} and 3.27 x 10^{-6} cm/sec. Three tests were performed at a confining pressure of 138 kPa and ranged from 1.5 x 10^{-6} to 9.38 x 10^{-7} cm/sec. Two tests were performed at confining pressures of 155 kPa and 172 kPa with results of 1.32 x 10^{-6} cm/sec and 1.18 x 10^{-6} cm/sec, respectively. These materials were classified as Clayey Sand or Lean Clay and meets the maximum required project permeability of 1 x 10^{-5} cm/sec. Additional investigations will be performed in 2020 to determine the volume of clay available beneath the WRD.

6.2.1.3.2 Overliner Material

Point load strength index (Is) tests were performed on select rock core samples of low-grade ore to determine the materials suitability for use as overliner material. Point load tests performed by Golder's Mississauga, Ontario laboratory in accordance with ASTM D5731 and the International Society of Rock Mechanics (ISRM) indicate that the intact strength of the rock material can be classified as strong rock (R4) with an estimated unconfined compressive strength ranging between 38 megapascals (MPa) and greater than 230 MPa.

Two samples were selected from the point load test and crushed to a maximum particle size of 50 mm per the specification for overliner material. Crushed samples were shipped to Golder's Denver, Colorado laboratory for durability and loaded–permeability testing. Durability of the overliner material was tested in accordance with ASTM C131. Test results indicated a loss between 22 and 23 percent using a grading designation of "A". A mass loss less than 45 percent after 500 revolutions satisfies the minimum durability requirements for use as overliner, concrete aggregate, and riprap materials.

Staged loading, constant head, rigid-wall permeability tests were performed on two samples of proposed overliner material. The samples were loaded to a pressure of 1520 kilopascals (kPa) to simulate overburden pressures from up to 80 meters of heap ore. The constant head permeability tests were performed using 380 kPa, 760 kPa, 1140 kPa, and 1520 kPa vertical pressures. Test results indicate that permeabilities ranged from 2.5 cm/sec to 4.4 cm/sec, which exceeds the minimum required project permeability of 0.1 cm/sec.

6.2.1.4 Subsurface Geotechnical Conditions

The geotechnical profile of the foundation materials below the HLF and WRD sites is based on the findings of the investigations listed above and the geotechnical laboratory testing. Subsurface descriptions are based on the conditions encountered at the time of exploration. Soil and subsurface water conditions outside of the exploration locations may vary from those encountered during the field explorations.

6.2.1.4.1 Subsurface Soil and Bedrock Profile

Geotechnical conditions at the HLF and WRD sites are relatively uniform, consisting of a sequence of shallow soils blanketing bedrock. Where exposed, bedrock conforms uniformly to the ground surface and there are few outcrops that extend above the ground surface. The ground is stable with no signs of instability or landslides, and Golder observed no signs of recent faulting. The risk of liquefaction in soil deposits is negligible due to the dense and dry nature of soil deposits. These conditions are well-suited to provide safe and stable support of the proposed HLF and WRD. From top to bottom, the geotechnical sequence with depth consists of:

- Topsoil
- Colluvium and Alluvium Soils
- Residual Soils
- Highly- to Moderately-Weathered Bedrock

The soil cover above the Highly- to Moderately- to Highly-Weathered Bedrock was typically between 0.2 meters and 0.5 meters in depth, but extended as deep as 5 meters in valley slopes. Bedrock was exposed in the base of most drainages where soils had been eroded.

6.2.1.4.1.1 Topsoil

A surficial topsoil layer blankets the WRD and HLF sites averaging about 0.1 to 0.2 meters in thickness. In general, the topsoil consisted of dry Silty to Clayey Sand, with minor fractions of Silt and Lean Clay and varying amounts of fine to coarse, angular gravels. The soil was generally non-plastic to low plasticity, though some soils exhibited moderate plasticity, and topsoil was classified as loose to medium dense and dry. Root growth and

organic content was variable across the site due to the variability of vegetation. Where more densely vegetated, topsoil had a high root content, and plant and tree roots extended below the topsoil layer to depths of a meter or more.

6.2.1.4.1.2 Colluvium and Alluvium Soils

Most explorations encountered Colluvium beneath the topsoil predominantly on the slopes of hills and drainages, and Alluvium was encountered near the base of drainages. Colluvium was documented in thickness ranging from 0.2 to 0.9 meters, and Alluvium was observed in deposits as deep as 5 meters. The Colluvium classified similar to the Topsoil as a Silty to Clayey Sand or Silty Gravel, with plasticity varying from non-plastic to moderate plasticity. The fraction of angular to subangular gravel and cobbles increased with depth. Because of the relatively steep and hilly topography, Alluvium consisted of flood deposits and slope wash deposits, and classified as Silty to Clayey Sand and Gravel, with varying fractions of angular cobbles and small boulders. Colluvium and Alluvium were classified as medium dense to dense, and dry to moist.

6.2.1.4.1.3 Residual Soils and Highly-Weathered Bedrock

Residual soils were observed in most explorations either directly below the topsoil or below the Alluvium and Colluvium deposits. Their mineralogy and soil descriptions were variable, reflecting the underlying volcani-clastic sedimentary rock and welded ash flow tuff bedrock, as well as degree of weathering. The Residual soil layer was thin, generally 0.2 meters to 0.4 meters in thickness. These soils were characterized as blends of angular gravels, sands, silts, and clays, with occasional angular cobble-sized fragments. Gravel and cobble-sized rock were typically weathered shale and siltstone fragments similar to the underlying bedrock. On ridges, these soils were more extensively weathered to contain a higher fraction of plastic fines (clays), with classifications of Clayey Sand with Gravel and Clayey Sand, and were dark brown or red brown in color. The Silty Sand, Silty Gravel, and Silty Sand with Gravel encountered on valley slopes were either non-plastic or had low plasticity, and were medium brown in color. Observed moisture was variably dry to moist.

6.2.1.4.1.4 Moderately- to Highly-Weathered Bedrock

Three rock types associated with the Esperanza Formation were observed in the foundations of the HLF and WRD.

Sedimentary deposits of siltstone and shale, with minor deposits of sandstone, dominate the HLF and WRD sites. At shallow depths, these deposits are highly fractured and highly-weathered, and become lease fractured and weathered with depth. This rock was platy when ripped or excavated, with rock dimension typically ranging from about 1 to 4 cm, but occasionally as large as 30 cm. While excavating test pits, backhoe refusal was typically encountered at depths of 2 to 5 meters. Rock strength was classified as

medium to high strength, and color was described as variably orange, yellow, cream, gray, and green.

A gneiss deposit underlying the sedimentary sequence has been exposed in the base of drainages of the HLF as observed in Test Pits KP-CDG-05, -20, and -22. The cream-colored rock is less weathered, less fractured, and of higher strength than the sedimentary sequence, and forms a relatively non-erosive channel bed where exposed. Where lightly weathered, test pit excavation was difficult.

An exposure of hydrothermally altered tuff was mapped and identified by test pits in the footprint of the Stage 2 leach pad. Tan in color and soft, this deposit had indistinct thick bedding and appeared as a lightly- to moderately-cemented soil deposit predominated by silty to clayey sand. Zones contained gravel and cobbles, and the rock was blocky when excavated. The deposit was similar to the massive tuff deposits exposed in the proposed pit area and was observed to weather readily when exposed and to be highly erosive, resulting in deeply-incised outcrops.

Laboratory testing indicated that the elastic modulus (Es) of the bedrock ranges from 38 to 7,465 megapascals (MPa) within the upper 10 meters of the subsurface profile with unconfined compressive strengths (UCS) ranging from 3.68 to 27.05 MPa. From depths of 10 to 20 meters, the UCS ranges from 2.71 to 78.65 MPa, and Es ranges from 908 to 14,798 MPa. From 20 to 40 meters, the UCS ranges from 1.74 to 20.62 MPa with Es ranging from 732 to 44,323 MPa.

6.2.1.4.2 Subsurface Water

Based on a review of the referenced documents, subsurface water was generally not observed during drilling and test pit excavation activity, with the exception of Boring BH-01 (KPC 2012b). This boring was drilled within the limits of the HLF area with subsurface water encountered at an approximate depth of 8.9 meters below the ground surface. The project area is generally a rock site in hilly terrain and minor localized springs are present in the base of the drainages in the HLF and WRD footprint; however, the regional ground water table is deep with some perched flow through fractured rock and geologic contacts. It is anticipated that the ground water will not significantly affect the strengths of the bedrock below the WRD and HLF. However, groundwater conditions can fluctuate depending on seasonal precipitation, irrigation, and other factors.

6.2.2 Site Seismicity

The Project site is located in the Zone B, Peneseismic Region, as shown on NOM-155, Figure 3 Regiones Sísmicas en la República Mexicana. The Peneseismic Region of Mexico has low to moderate seismicity. Knight Piésold prepared a site-specific seismic hazard assessment of the project site in 2011 to more closely define seismic risks and to

develop seismic parameters for use in facilities design (KP, 2013a). Their study developed seismic ground motion parameters for the Project site using both probabilistic seismic hazard assessment (PSHA) and deterministic seismic hazard assessment (DSHA). Their report includes a detailed discussion of potential earthquake sources, as well as their analyses and findings. Golder has reviewed the seismic hazard assessment performed by KP and feels that their conclusions and recommendations are acceptable, as discussed in Golder's 2014 technical memorandum "*Criterios de Diseño Sismorresistentes en México y Parámetros Sísmicos a ser Empleados para el Diseño de la Pila de Lixiviación del Proyecto Cerro del Gallo*." Their findings are summarized in the following sections.

6.2.2.1 Seismic Parameters for HLF and WRD Design

The PSHA developed peak ground accelerations (PGA) for various return periods and annual risk as presented in Table 6.1.

Table 6.1: Summary of Probabilistic Seismic Hazard Assessment for Project Site

Return Period (years)	Annual Frequency of Exceedance (%)	Peak Ground Acceleration (g) (KP, 2013a)
100	1	0.05
475	0.2	0.08
1,000	0.1	0.10
2,500	0.04	0.12
5,000	0.02	0.14

Source: Golder (2014)

The standard of practice in North America is to design HLFs and WRDs that have relatively low consequence of movement (typical of those for the CdG Project) using a PGA determined for a 475 year return period. This acceleration is applicable during both operation and after these facilities have been closed, assuming that the ore on the leach pad is rinsed and the leach pad and WRD remain drained. Under these conditions, potential movements of the rinsed ore in the heap and the rock in the WRD would provide negligible environmental impact and negligible risk to human safety. The PGA of 0.08g from the 475-year return period was adopted for stability analyses of the HLF and WRD for the Project site.

6.2.2.2 Seismic Parameters for Structural Design

In accordance with the International Building Code (IBC), the maximum considered earthquake ground motion has been defined as the ground motion with a 2 percent probability of exceedance in 50 years (return period of about 2,500 years). Specifically, seismic parameters for use with the IBC are provided below for the site (Knight Piésold (2013):

- Seismic coefficient, SS = 0.26g
- Seismic coefficient, S1 = 0.32g
- Peak ground acceleration = 0.12g

These acceleration values correspond to a reference ground condition of Site Class B (defined by the IBC as “Rock”). For geotechnical foundation design of mine site structures, a design earthquake of magnitude 7.0 is recommended for seismic design analyses.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The CdG deposit is located in central Mexico within the Mesa Central physiographic province that includes the Guerrero Composite Terrane (Campa and Coney, 1983). The Guerrero Composite Terrane is characterized by submarine and subaerial volcanic and sedimentary successions that range in age from Jurassic to Middle–Late Cretaceous. Similar to other terranes of the North America Cordillera, the Guerrero Composite Terrane has been interpreted as an exotic crustal block accreted to what is now Mexico in Late Cretaceous time via a westward-dipping subduction zone that closed a major ocean basin (Lapierre et. al., 1992; Tardy et al., 1994; Dickinson and Lawton, 2001, etc.).

The oldest rocks in the CdG region are a deformed and regionally metamorphosed volcano-sedimentary sequence of Triassic to Cretaceous age (Randall et al 1994; Consejo de Recursos Minerales 1992). Consejo de Recursos Minerales referred to these rocks as the Esperanza Formation, described as carbonaceous and calcareous shale interbedded with arenite, limestone and andesite to basaltic flows, all weakly metamorphosed to phyllite, slate and marble.

In the CdG Project area, the Esperanza Formation consists of layered sedimentary rocks of argillaceous and silty argillaceous composition, and fragmental volcanic rocks of broadly intermediate composition, including ash tuffs, lithic to crystal tuffs and some volcanic breccias and agglomerates. In the Project area the Esperanza Formation is locally surrounded by Tertiary age rhyolitic flows, rhyolitic tuffs, trachyte-andesite and andesite.

A swarm of basaltic dykes intrude rocks of the Esperanza Formation predominantly along northwest trending fracture zones. The basaltic dykes contain olivine and xenoliths of altered felsic volcanogenic rock and some dioritic composition fragments, (Mason 2005a). The lack of overprinting metamorphic and hydrothermal alteration minerals indicates that these rocks were emplaced post peak regional deformation and felsic magmatism. The dykes are evident in airborne magnetic imagery as high amplitude discontinuous linear and circular magnetic anomalies often emplaced along the same structural zones as the larger epithermal silver-gold vein systems in the district.

In addition to numerous gold, silver and some mercury occurrences in the district there are a number of tin occurrences concentrated in a northwest-southeast trending belt that includes CdG. These tin showings and mapped granitic intrusive bodies suggest the

possibility of a magmatic province that may be prospective for additional intrusion-related copper-gold deposits.

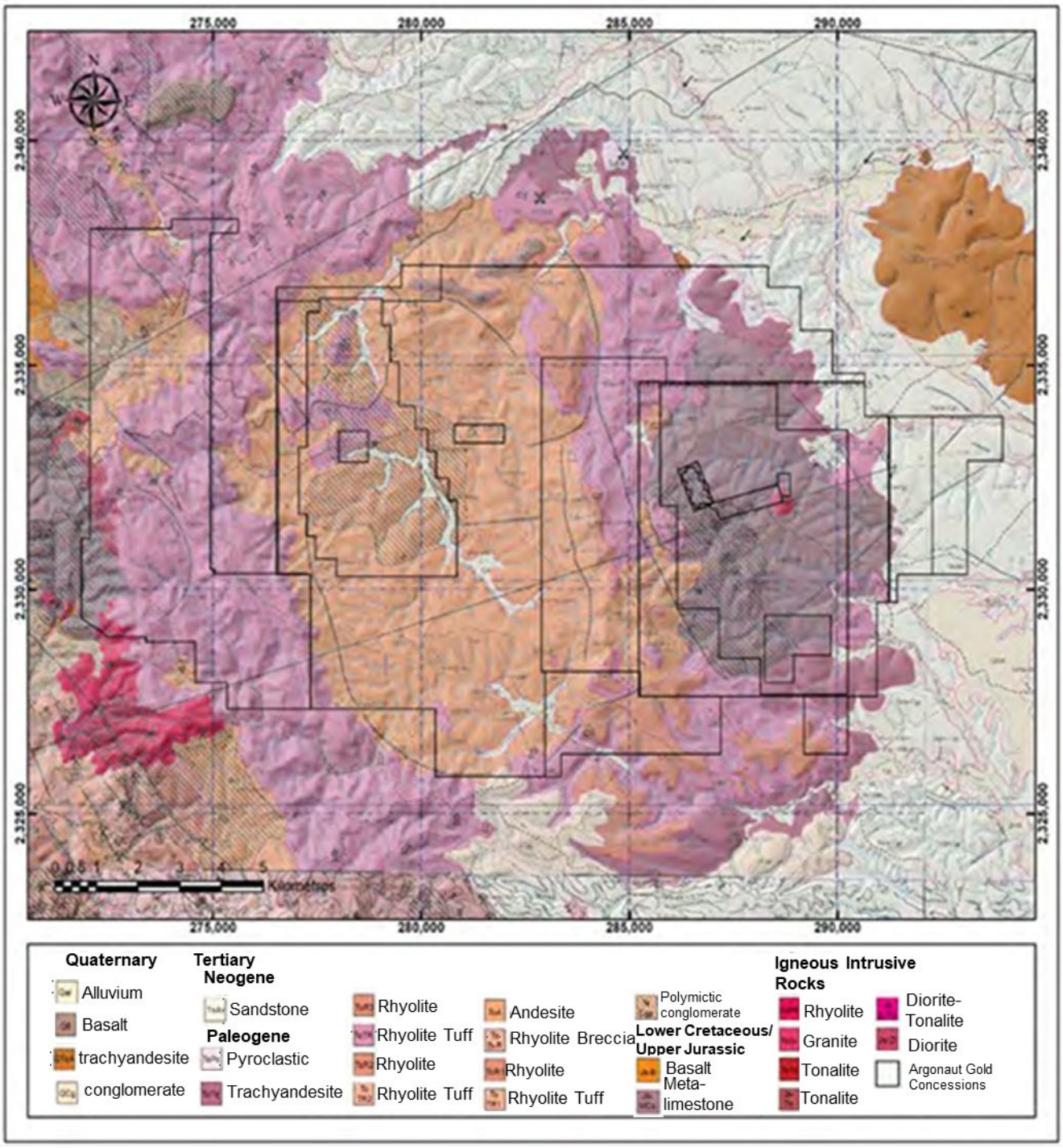
7.2 Project Geology

7.2.1 Clastic Sedimentary and Volcanoclastic Sedimentary Host Rocks

The Esperanza Formation in the CdG Project area consists of a conformable sequence of siliciclastic sedimentary and fragmental volcanogenic rocks (Figure 7.1**Error! Reference source not found.**). The stratigraphy strikes broadly east-west and dips northerly at 30-40°, except in the vicinity of CdG where dips steepen to greater than 60° (Groves, 2008). The stratigraphic thickness of the Esperanza Formation has not been determined. The formation is thought to be of Cretaceous age due to similar rocks within the Guanajuato district having been assigned to this age. Esperanza Formation basement rocks around CdG can be broadly subdivided into three litho-stratigraphic packages. To the south of the CdG intrusive complex a siliciclastic sedimentary sequence is recognized and consists dominantly of shale and interbedded arenite with minor lenses of conglomerate (Groves, 2008). This package is conformably overlain by a sequence of fragmental volcanogenic rocks of intermediate composition that are the host rocks for the CdG copper-gold-silver deposit (Mason, 2005a, Mason, 2006a).

The stratigraphy to the north and east of CdG is predominantly sedimentary with minor volcaniclastic horizons (Groves, 2008). To the west, volcanogenic sedimentary rocks with consistent shallow to moderate northerly dips. The southern flank of CdG is dominated by siliciclastic sedimentary rocks.

Figure 7.1: Regional Geology Map



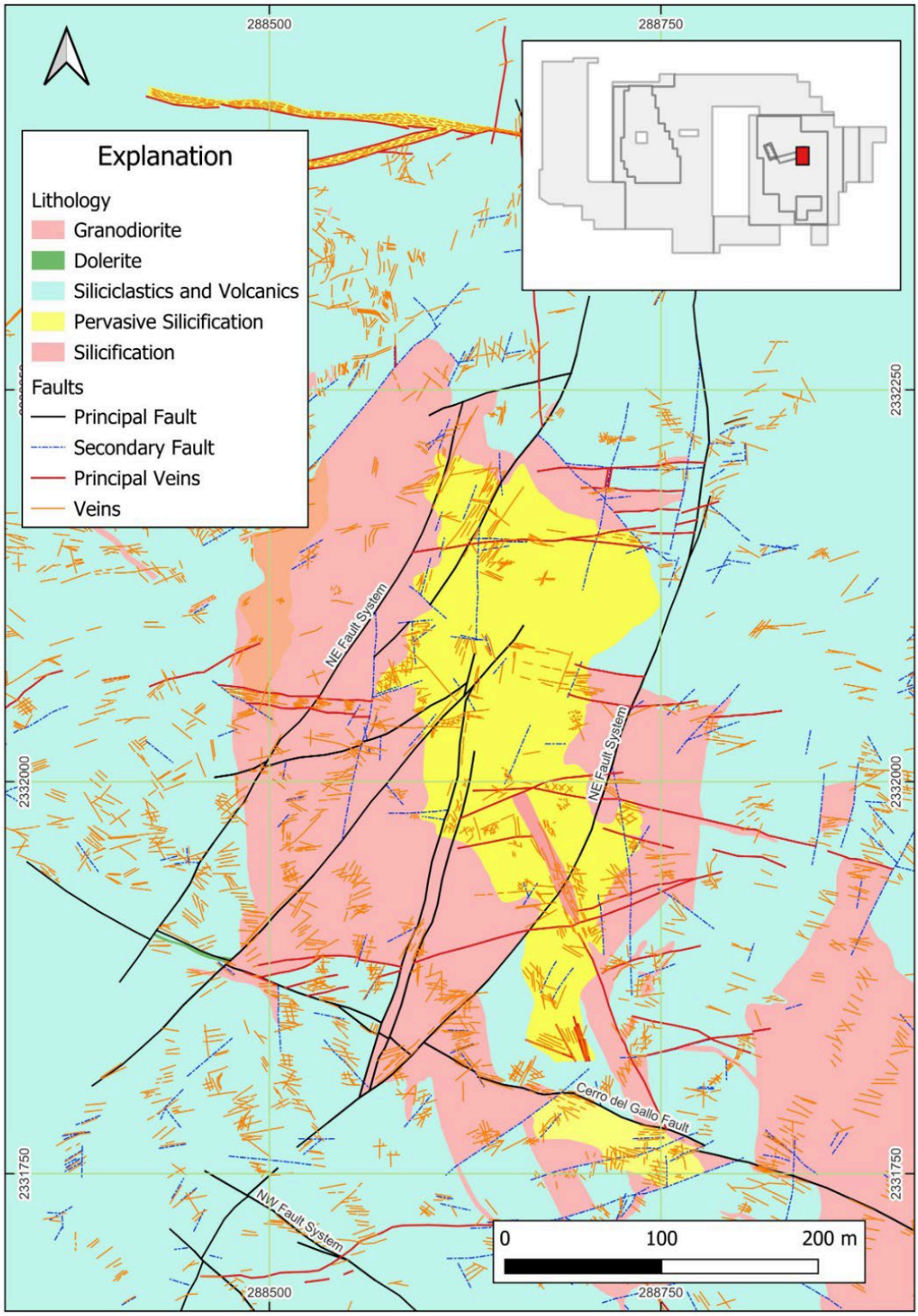
Source: PEA (2021)

7.2.2 Felsic Intrusive Host Rocks

CdG (the actual hill) is cored by a steep sided stock of granodiorite to monzogranitic composition with a surface exposure measuring approximately 600 meters in a north-

south direction and 300 meters wide (Figure 7.2). A subsidiary intrusive of more granodioritic composition was emplaced in the southeast side of this main intrusive and is likely a more intermediate phase of the same parent magma. Gold and copper are spatially and genetically associated with this intrusive complex and occur largely as an up-right quartz vein stockwork/breccia system formed peripheral to the intrusive (**Error! Reference source not found.**; **Error! Reference source not found.**). This gold-rich zone varies in width but is locally as wide as 50 meters and is largely silicified intrusive. Outward from this mineralized annulus, veins and silicification continue into the country rock with decreasing intensity (Figure 7-4).

Figure 7.2: Map of Local Cerro del Gallo Geology



Source: Heliostar (2026)

Figure 7.3: Oxidized Outcrop along Intrusive Margin



Source: PEA (2021)

Figure 7.4: Mineralized Quartz Veins in Weathered Country Rock



Source: PEA (2021)

CdG fits well into a copper-gold porphyry mineralization model in that mineralization is related to volatile phases of a felsic intrusive complex. This mineralization is genetically and spatially associated with this intrusive and is proximally zoned concentrically around it. Copper locally overlaps the gold zone and extends outward from the intrusive farther

than the gold system. Silver appears to be structurally related and occurs as overlapping epithermal veins and is clearly younger than the gold-copper system.

The CdG intrusive complex comprises multiple phases of calc-alkaline felsic intrusive rocks of granodioritic to rhyolitic composition emplaced at high crustal levels into a relatively flat lying volcanogenic tuff sequence. This suite of felsic intrusives indicates progressive fractionation of a magma chamber at depth. A sub-volcanic environment is postulated. An Oligocene age for intrusion emplacement is inferred based on the spatial and temporal association of similar felsic intrusive rocks with epithermal silver- gold mineralization at Guanajuato. The intrusive related mineral system is centered on CdG and forms an upright elongate stock of granodioritic to monzogranitic composition (Mason 2005b, Mason 2006b), which was previously described as a quartz monzonite (Consejo de Recursos Minerales, 1992), and a quartz monzonite porphyry (Rowins, 2000a). These observations are in accord with work of Perez (2018) who describes the rocks in thin sections as a quartz-monzonitic-hypabyssal intrusive.

A siliceous cap of bladed and lattice textured cherty quartz extends over the top of CdG giving the hill the appearance of a “rooster’s comb,” hence the name Cerro del Gallo or “Hill of the Rooster.” The siliceous cap extends along the axis of elongation of the outcropping main intrusive and is as much as 100 meters wide and 400 meters long and centered on the peak of CdG. This siliceous cap represents the top of the intrusive system and includes minor amounts of strongly silicified volcano-sedimentary wall rocks.

7.3 Deposit Descriptions

7.3.1 Deposit Dimensions

The CdG deposit is approximately 600 m by 300 m in size. Gold mineralization generally follows the intrusive contact and forms a concentric halo up to 75 m in width. Copper mineralization commonly follows the gold mineralization but extends into the intrusive and hornfels (HF) units. Silver mineralization is less controlled by the intrusive contact and a northwesterly striking trend.

7.3.2 Structure

The CdG Project has complex structural history dating back to what are believed to be Jurassic aged volcaniclastic dominated basement rocks that have been intensely folded and regionally metamorphosed to more recent lithologic units that display relatively young fracturing and faulting.

Several post-mineral faults shown in **Error! Reference source not found.** apparently locally offset mineralization and act as hard boundaries to mineralization. The main post-mineral faults are described as follows:

7.3.2.1 Cerro Del Gallo fault

The Cerro del Gallo fault cross cuts the CdG deposit and locally displaces the mineralized system. This structure has a general trend of 310º and is inclined around 80° to the NE and locally forms a hard boundary to mineralization.

7.3.2.2 NE fault system

The NE fault system does locally offset mineralization. It consists of at least five separate structures and trends between 030º-050º with dips between 75º-80º to the SE. Portions of this fault system were utilized in the geologic model as hard boundaries to mineralization.

7.3.2.3 NW fault system

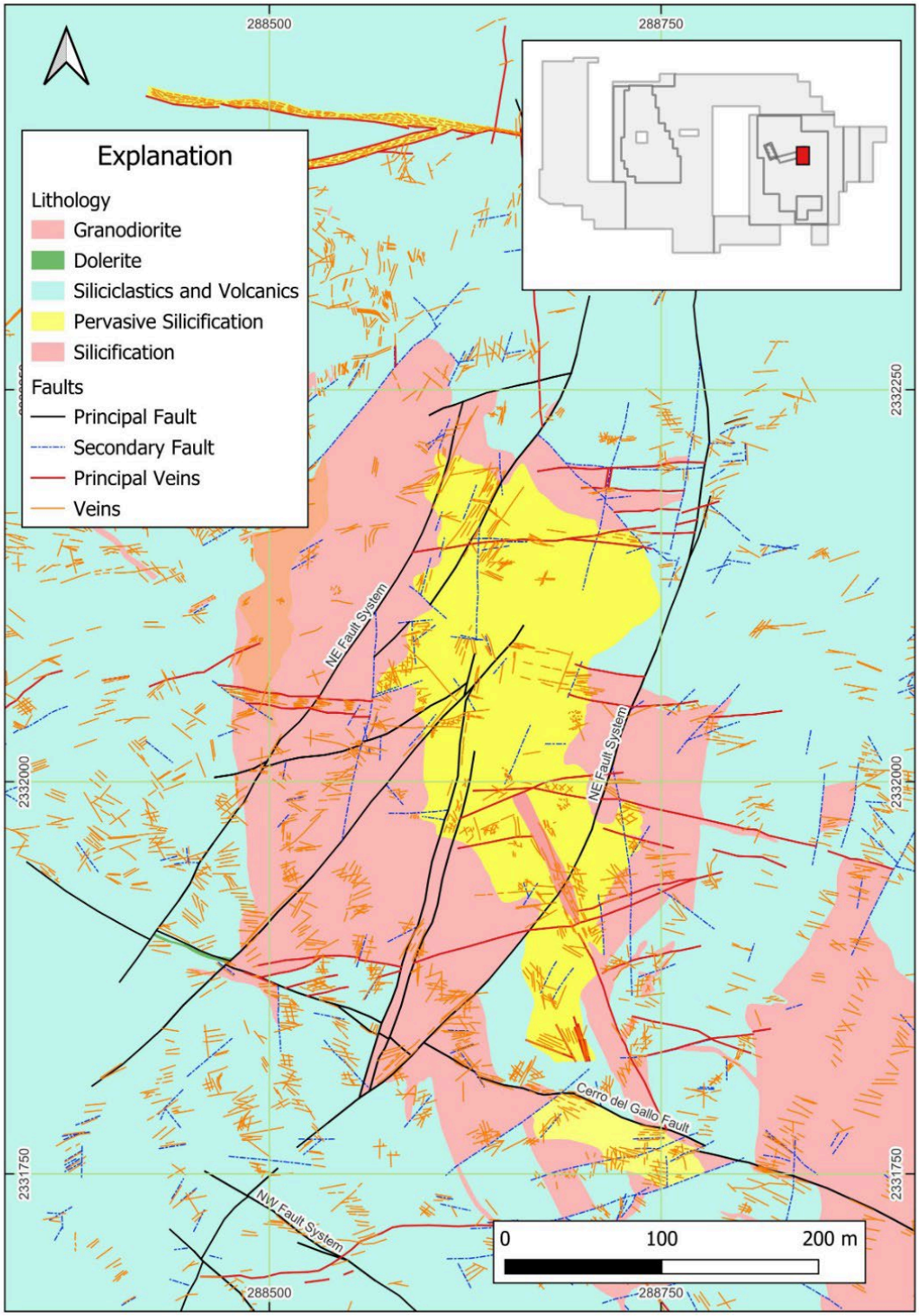
The NW fault system is located on the southern flanks of CdG and consists of a series of structures that trend to the NW and dip 50 to 60º to the NE.

7.3.2.4 La Paz fault

The La Paz fault trends northwesterly and dips steeply to the north. It acts as a local barrier to mineralization and also separates sedimentary rocks to the south from volcaniclastic rocks to the north. This fault also hosts a one-meter-wide epithermal vein with anomalous gold and silver.

Mineralized veins are domained by dominant structural direction (**Error! Reference source not found.**).

Figure 7.5: Mineralization Trends by Domains



Source: Heliostar (2026)

7.3.3 Mineralization

This section is summarized from earlier Technical Reports completed by San Antón and Cerro Resources.

The main copper-gold mineralizing event is considered to be intrusive related and referred to as a copper-gold porphyry system. Within the district there are also younger epithermal vein systems high in silver and lead. These are similar to the style of mineralization that makes up the Guanajuato Mining District.

Skarns with associated occurrences of gold, copper, zinc, and lead occur around the periphery of the intrusive. Skarn mineralization is scattered and has only relatively small tonnage potential.

At CdG, mineralization is hosted in both felsic intrusive and volcanic tuff wall-rock. Mineralization is disseminated and vein or fracture controlled and extends from 200 m to 400 m outward from the mineralizing intrusive complex.

The strongest gold mineralization at CdG is associated with intense quartz stockwork development and silicification within a wall-rock annulus forming the outer limits of a felsic stock with the system losing intensity outward with a decrease in stockwork and quartz vein density.

Sulfide comprises less than 2% by volume of the mineralized rocks. Gold-copper mineralization is zoned concentrically around the felsic intrusive with higher grade gold mineralization proximal to and within an outer annulus of the intrusion. The highest copper grades are found outward from the gold zone. Zinc mineralization is locally anomalous outside the copper zone; metal zonation boundaries are gradational and there is an overlap in the gold-copper zone and the copper-zinc zone.

Pyrite is the dominant iron sulfide mineral with accessory pyrrhotite and marcasite dispersed throughout the mineralized system. Pyrite occurs in two forms; as euhedral to subhedral cubic crystals and crystal aggregates in veins and disseminations of primary origin, and also as secondary fine-grained patches after pyrrhotite (Mason, 2005a). Pyrrhotite is most common in the outer copper zone. It is magnetic and in high enough concentration to form a donut shaped magnetic anomaly around the CdG mineral system. Native gold occurs in quartz veins and as inclusions in pyrite, chalcopyrite and bismuthinite. Gold grains range in size from generally 0.5-4 micrometers in diameter to rarely 10-20 micrometers (Mason, 2005a; Mason, 2006a; Townend, 2006). Electron-probe microanalysis on a limited number of native gold grains indicates gold has a fineness of 860-880 with silver comprising the rest (Mason, 2006b, Mason, 2007).

In terms of geochemistry and all elements assayed, gold has the strongest correlation with bismuth.

The majority of silver at CdG is related to late structurally controlled epithermal veins that overprint the intrusive related copper-gold system. Within the veins, Perez (2018) identified tetrahedrite and electrum as the main silver minerals associated with galena and sphalerite. Silver locally occurs as small grains of native silver or ruby silver as either pyrargyrite or proustite.

Within the CdG mineral system, chalcopyrite is the most common base metal sulfide phase and occurs in fracture veinlets often associated with intense silicification and as disseminations. Chalcopyrite is commonly closely associated with pyrite and marcasite, and rarely as inclusions in coarse arsenopyrite (Townend, 2006). Traces of bornite were reported (Mason, 2006b; Townend, 2006); however, bornite is not volumetrically significant.

Minor secondary copper minerals including malachite and azurite have formed by weathering of Cu-bearing sulfide minerals and are locally present in surface outcrops. Native copper, covellite and chalcocite are found deeper in the regolith profile, and their formation is attributed to supergene weathering processes.

Arsenopyrite is relatively common and occurs as coarse discrete grains often associated with chalcopyrite.

7.3.4 Alteration

Hydrothermal alteration is zoned concentrically around the main intrusive complex and intensity decreases outwards. Overprinting hydrothermal events result in complex alteration patterns that are difficult to map as discrete areas. Within and outward from the main intrusive there is a general absence of primary textures due to high percentages of silica. Potassic alteration is the dominant style of alteration in the core of the hydrothermal system which is centered on the Main Intrusive.

The tuffaceous wall rocks in contact with the intrusive system have experienced pervasive hydrothermal alteration to form fine grained replacement assemblages of albite, K-feldspar, biotite, sericite, quartz, sphene, rutile and pyrite (Mason, 2005b; Mason, 2007). Much of the mineralization is fracture controlled and related to K-feldspar, biotite, pyrrhotite, and chalcopyrite, but locally forms thicker granular textured veins dominated by quartz with minor biotite, chlorite, calcite, pyrite, pyrrhotite, chalcopyrite, molybdenite and bismuthinite (Mason 2005b).

Hydrothermal propylitic alteration overprints potassic alteration within both the felsic intrusives and surrounding wall-rocks and extends laterally beyond the zone of potassic alteration surrounding CdG.

Propylitic alteration extends over a radius of up to one km from the peak of CdG (**Error! Reference source not found.**). This alteration is more extensive to the west and consists mainly of fracture-controlled replacement assemblages of albite, chlorite, calcite and pyrite.

7.4 QP Comments on Section 7

The geology, structure, and mineralization are sufficiently well understood to support a PFS. The extent of the alteration is not well documented in the geology logs which precludes developing a 3-D alteration model. It is recommended a drill core be relogged for alteration.

8 DEPOSIT TYPES

8.1 Overview

The CdG copper-gold-silver deposit is considered to be a member of a distinct subclass of “reduced” porphyry-style copper-gold mineralization as first proposed by Rowins (2000b). These reduced porphyry copper-gold deposits lack primary hematite, magnetite and sulfate minerals, but contain abundant hypogene pyrrhotite, commonly have carbonic ore fluids, and are associated with ilmenite-bearing, reduced I-type granitoids (Rowins, 2000b). CdG displays all these features. In addition, there is a consistent pattern of higher temperature potassic alteration overprinted by lower temperature propylitic-style mineral alteration. Propylitic alteration boundaries are gradational and irregular, and more widespread than potassic alteration. This alteration pattern is consistent with many other porphyry copper-gold deposits throughout the world. Tellurium-bearing minerals are also common in porphyry copper-gold deposits, as they are at CdG.

The CdG copper-gold-silver deposit also has characteristics supporting an intrusion-related gold system (IRGS) model as proposed by Thompson et al. (1999), Rowins (2000b), Champion (2005), and Hart (2005). IRGS deposits are typically found in continental tectonic settings inboard of convergent plate boundaries, often where the regional metallogeny is characterized by tungsten-tin magmatic provinces. Felsic intrusive rocks have an intermediate oxidation state between ilmenite and magnetite series with a gold-enriched metal assemblage derived from igneous fractionation, and a distinctive metallogenic signature of gold, bismuth, tin and tungsten. Hydrothermal fluids are carbonic, and pyrrhotite is common.

The CdG Project has several epithermal veining systems, one of which, the Ave de Gracia, transects CdG. None of the vein systems has had chemical determination for classification of sulfidation type, although there are several geological characteristics similar to low sulfidation epithermal deposits, all of them determined by geological mapping and geological description. Some of these characteristics are: a) sericite or illite ± adularia, chlorite alteration minerals; b) filling cavities/porosities vein type, banding and hydrothermal breccias; c) carbonate replacement textures; d) low content of bulk sulfur, low presence base metals (Pb, Zn).

Massive sulfide mineralization occurs in and around the CdG deposit. These rocks are composed of more than 60% sulfides, with variable quantities of pyrite, pyrrhotite, chalcopyrite, sphalerite, arsenopyrite and galena and are normally strata bound unless

remobilized. In the earlier logging programs these occurrences of massive sulfides were described as skarns. Similar occurrences are reported in the Mesozoic rocks of the Guanajuato districts (Randall, et al., 1994).

8.2 QP Comments on Section 8

Understanding of the geological setting and deposit type is adequate to support mineral exploration at CdG.

9 EXPLORATION

9.1 Summary

Heliostar has done no exploration on the property since it was acquired and is relying on historical exploration which is briefly discussed below.

The majority of modern exploration on the CdG copper-gold system and the surrounding epithermal veins was accomplished by Cerro Resources and its predecessor companies between 2006 and 2008 and then by Primero from 2013 to 2018.

During 2006-2008, multiple surface geochemical sampling programs were completed including stream sediment, BLEG, Niton® soil, and conventional soil sampling. These geochemical programs are described in Groves (2008) and Groves and Carman (2007).

From 2013 to 2018, under Primero's ownership, exploration within the property included additional epithermal vein exploration, programs of detail geological mapping, stream sediments sampling, extensive soil sampling, rock chip sampling and analyses of available geophysical data. During 2014 Primero hired a consulting firm (Western Mining Services) based in Denver, CO to conduct a district target generation from the geological information available (Historical + Primero). A total of eighteen district-scale targets of different deposit types (epithermal, skarn and intrusive-related) were identified, of which, six were considered high priority. To date, these targets remain untested.

Work by Argonaut, starting in December of 2017, included a detailed re-logging and re-classification (in percentage) of oxide and sulfide material in rock chips trays from historical RC drilling. This study focused on defining four separate oxide domains. This data, in addition to geologic information, was transferred to 50-meter spaced vertical cross sections through the deposit. This combined data was utilized to construct a three-dimensional model using Leapfrog software. The objective of the program was to determine the tonnage and grade of gold and silver mineralization within each geo-metallurgical zone (or oxidation zone).

Argonaut conducted detailed mapping of the deposit and limited surface exploration in the surrounding district. The deposit was mapped at 1:500 scale with a focus specifically on collecting structural data and detailed information with respect to the mineralization. During this time, Argonaut collected numerous surface samples of the veins and mineralized areas. In the district surrounding the main CdG deposit, Argonaut conducted

reconnaissance exploration consisting primarily of selective sampling and 1:10,000 scale mapping of targets previously identified by Cerro Resources. To date, the district has yet to see detailed, dedicated exploration.

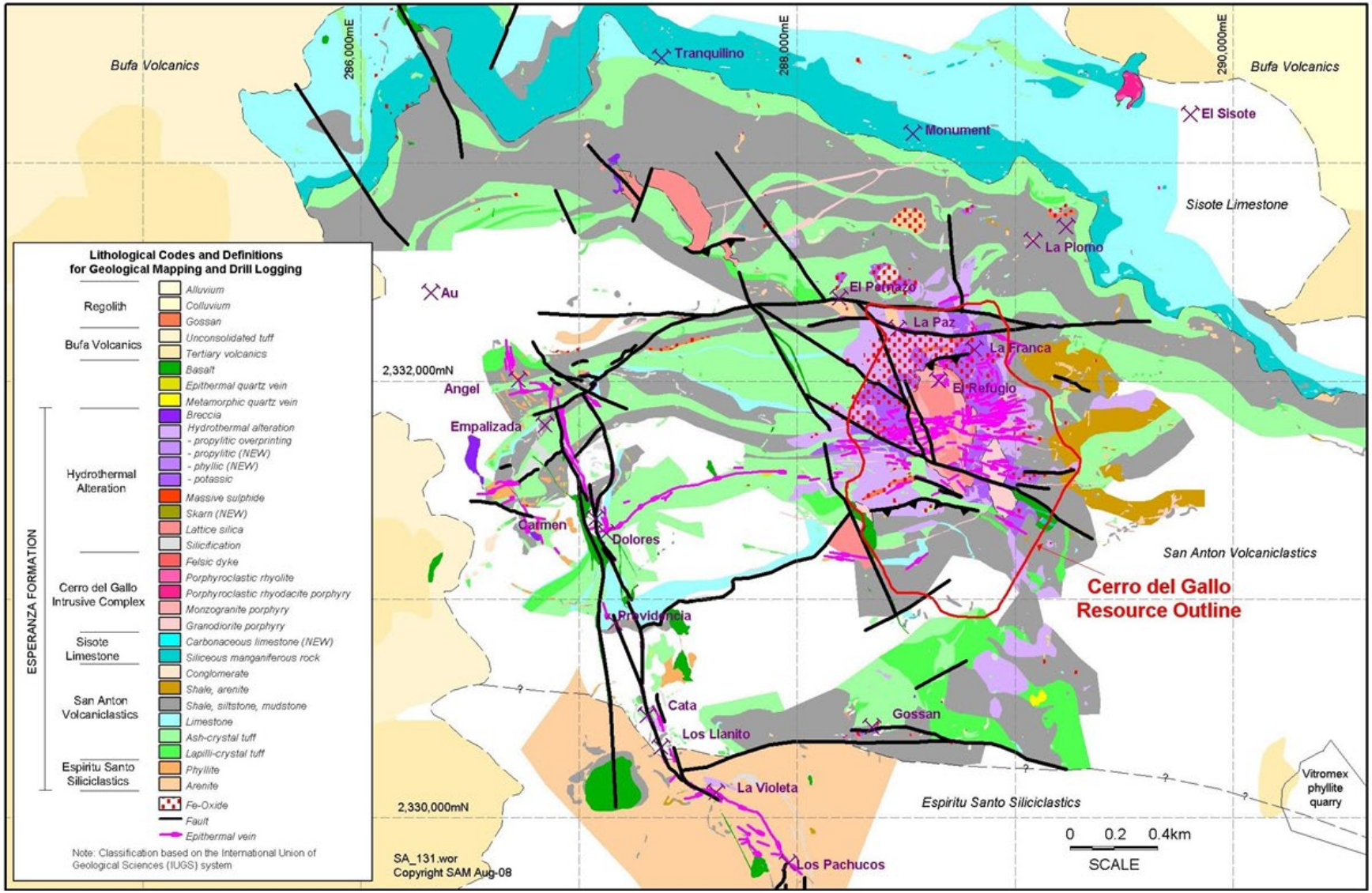
9.2 Grids and Surveys

The CdG area lies within Universal Transverse Mercator (UTM) Zone 14 Q. All coordinates were based on North American Datum 1927 (NAD27) for Mexico through 2008. Starting in 2013 with Primero, coordinates were based on World Geodetic System 1984 (WGS84). Heliostar used WGS84. NAD27 coordinates were converted to WGS84 coordinates which are used now.

9.3 Geological Mapping

Geological mapping is discussed in a report by Groves (2007; 2008). Most of the area was mapped at 1:2500 with some mapping at 1:1000 across the deposit and surrounding area. Groves (2007) focused on epithermal vein systems adjacent to or in the area of CdG. Groves (2008) focused on CdG proper. Figure 9.1 is based on that mapping.

Figure 9.1:Geological Map of the CdG Area

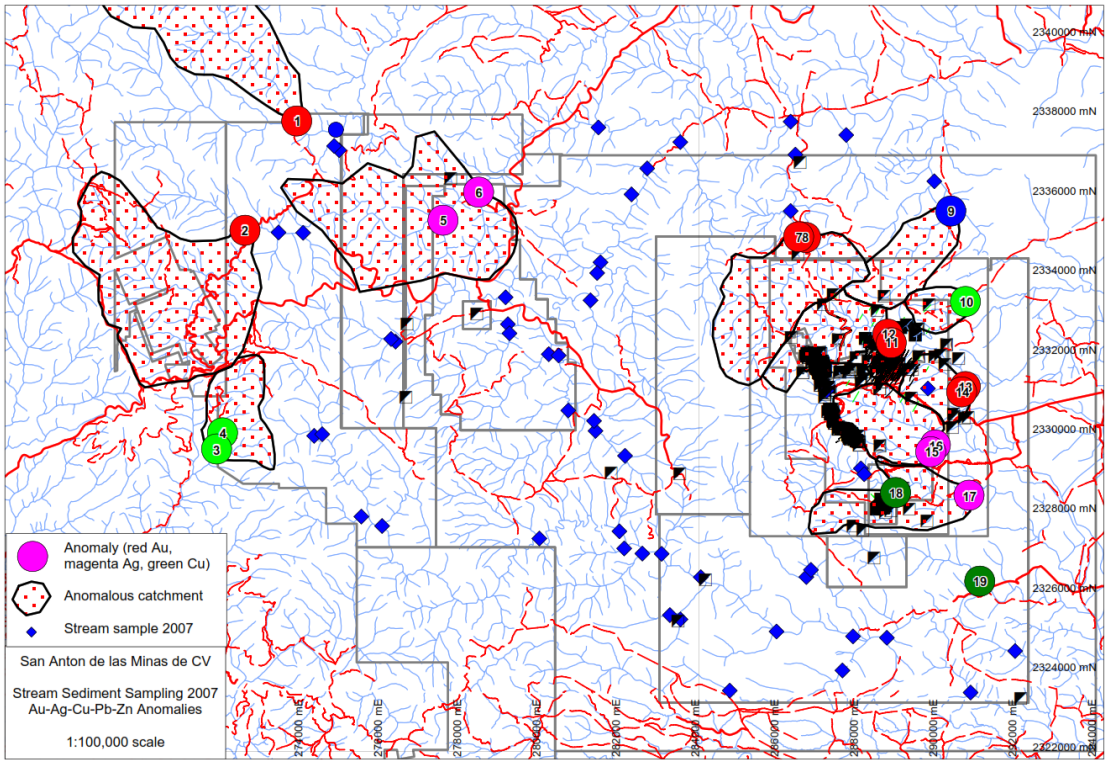


Source: Primero Mining (2016).

9.4 Geochemical Sampling

Groves and Carman (2007) discuss results of stream and soil geochemistry in the CdG area. Figure 9.2 shows 2007 stream sediment sample locations and anomalous catchment areas defined on the basis of these samples. Prior to 2014, Silver Standard and Geologix collected over 2,700 soil samples at the San Agustin property. Those samples were analyzed for gold and other elements.

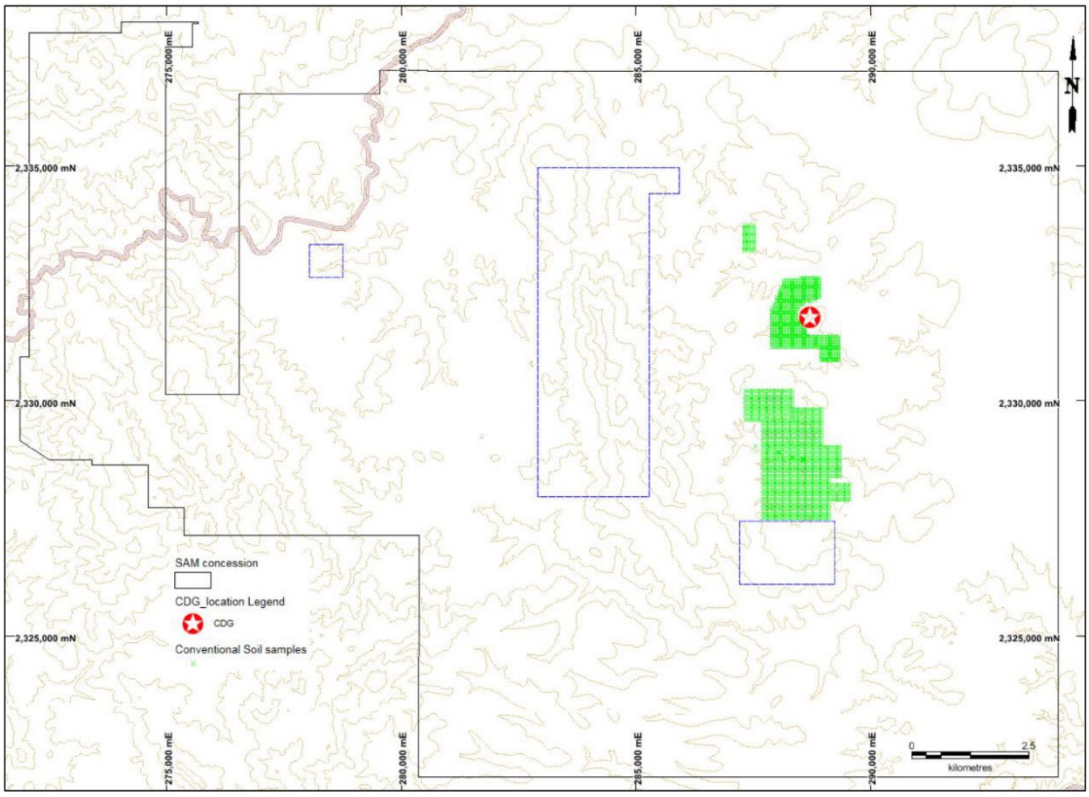
Figure 9.2: 2007 Stream Sediment Sampling Showing Sample Locations and Anomalous Catchments



Source: Primero Mining (2016).

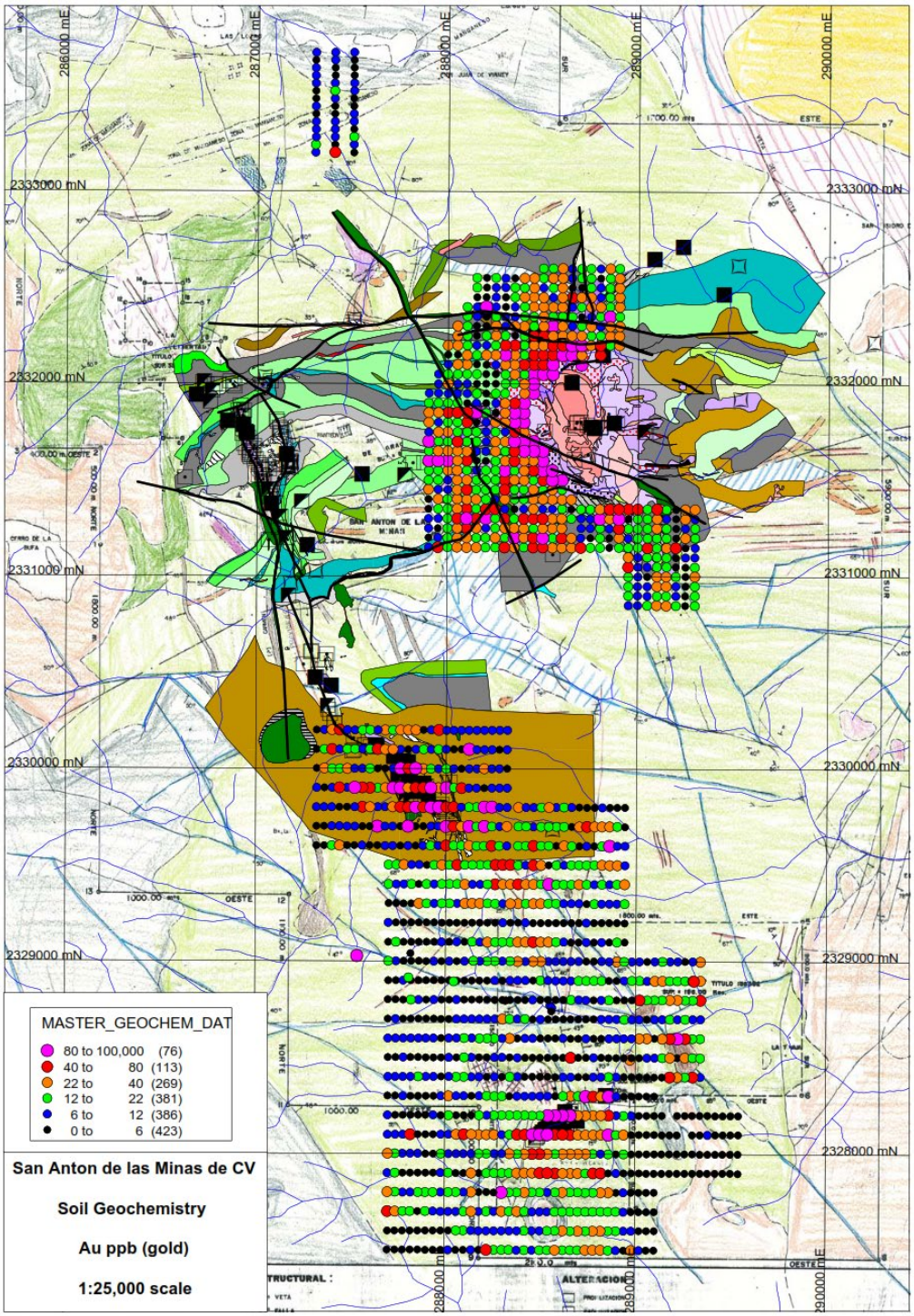
Figure 9.3 shows the 2007 soil sample locations relative to the property.
Source: Primero Mining (2016).
shows the location of the 2007 soil samples and summarizes the Au results of those samples. Approximately 1,645 soil samples were collected for this program and anlayzed using multi-element procedures.

Figure 9.3: 2007 Soil Sample Locations



Source: Primero Mining (2016).

Figure 9.4: 2007 Soil Sample Locations and Au Results.

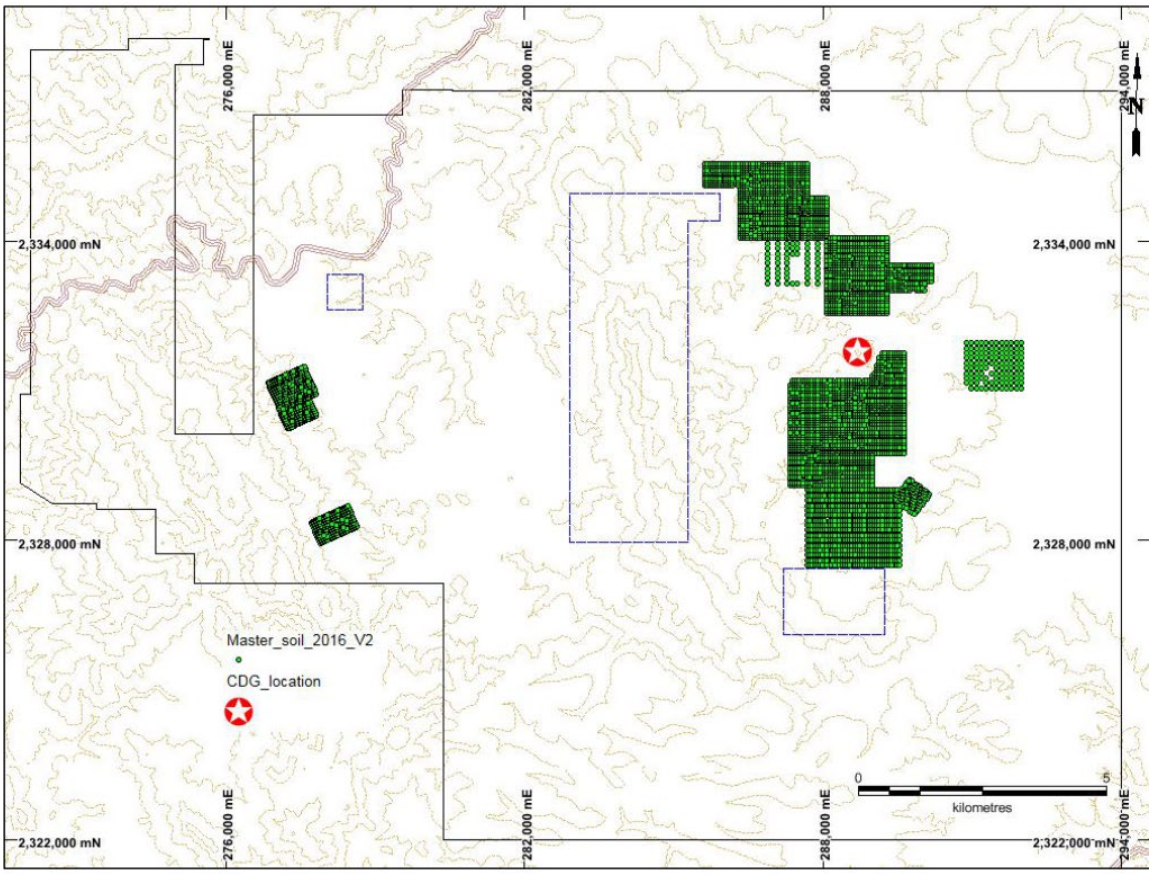


Source: Primero Mining (2016).

In 2014, approximately 5,889 soil samples were collected in the area (Figure 9.5) to followup on existing geochemistry in the CdG region and anlayzed using multi-element procedures.

Many anomalous samples were identified. Most have not been explored more.

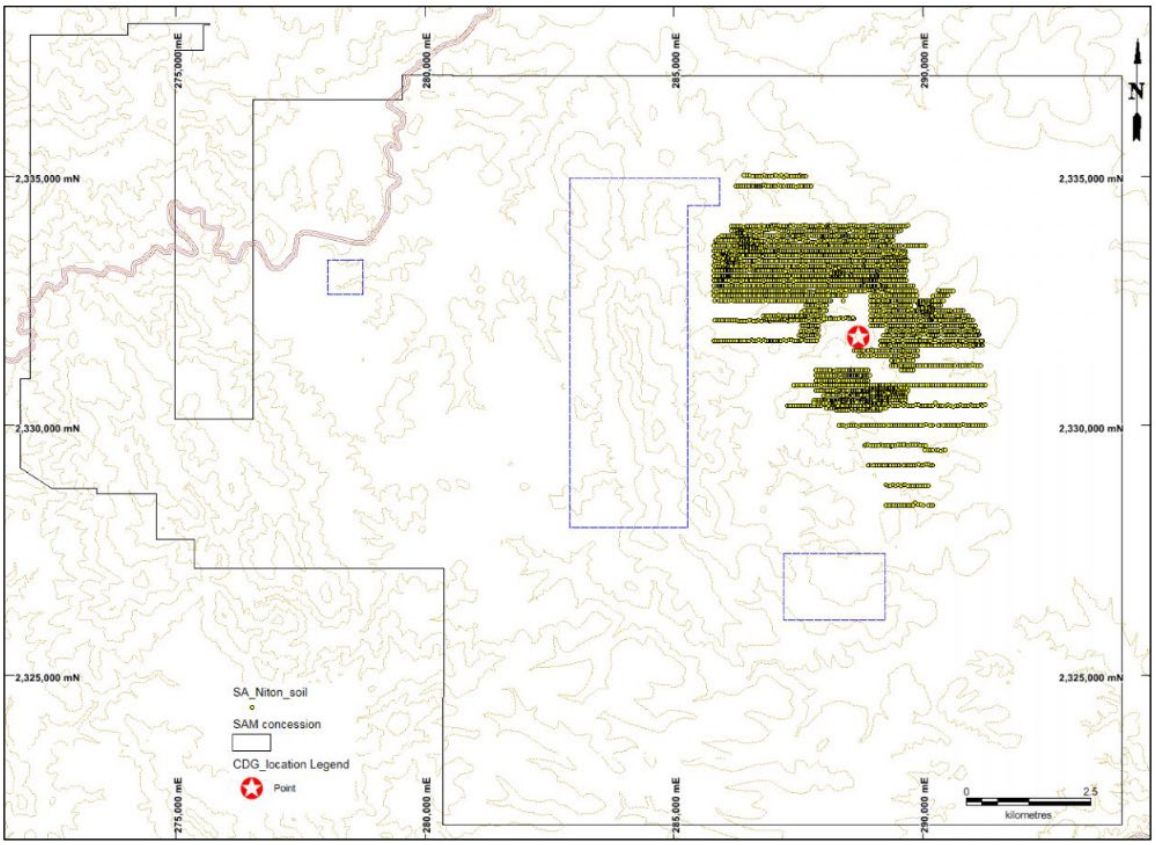
Figure 9.5: Primero Soil Sampling



Source: Primero Mining (2016).

In early 2008 a Niton hand-held XRF was employed to follow-up on anomalous soil samples at Tranquillino, Monumento, and the Southern Pb-Zn Anomaly. Approximately 3,111 samples were analyzed in this program. Results are summarized in volume 2 of Groves and Carman (2007). Locations for those samples are shown in **Error! Reference source not found.**. This work is reasonable, but hand-held XRF instruments are limited in their capabilities and should be viewed with caution.

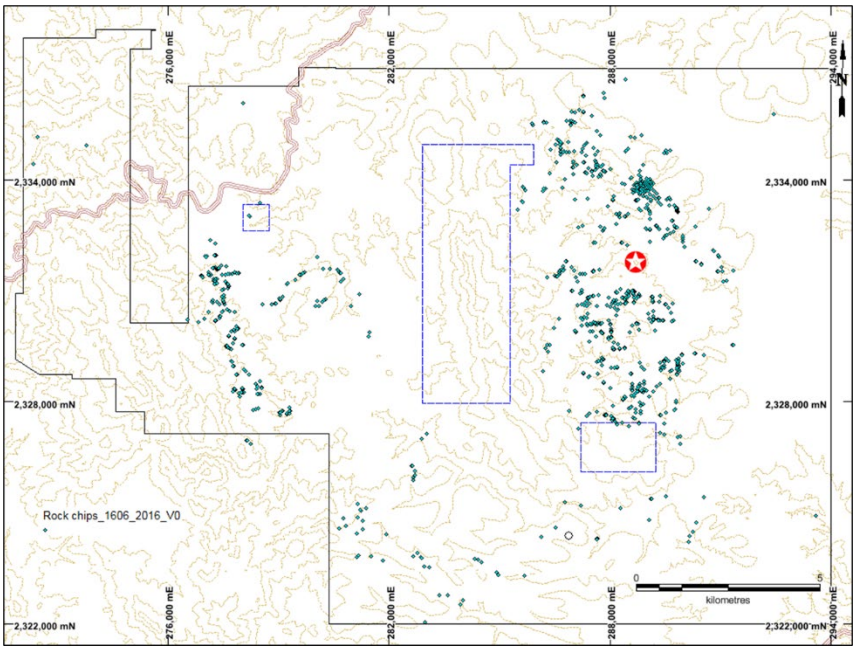
Figure 9.6: Samples Analyzed by NITON



Source: Primero Mining (2016).

From 2014 to mid-2016, 1,671 rock chip samples were collected from the property. Figure 9.7 shows the locations of those samples. Some very anomalous materials were collected and analyzed, but none, to the QP's knowledge have been followed up with additional exploration.

Figure 9.7: Rock Chip Sample Locations



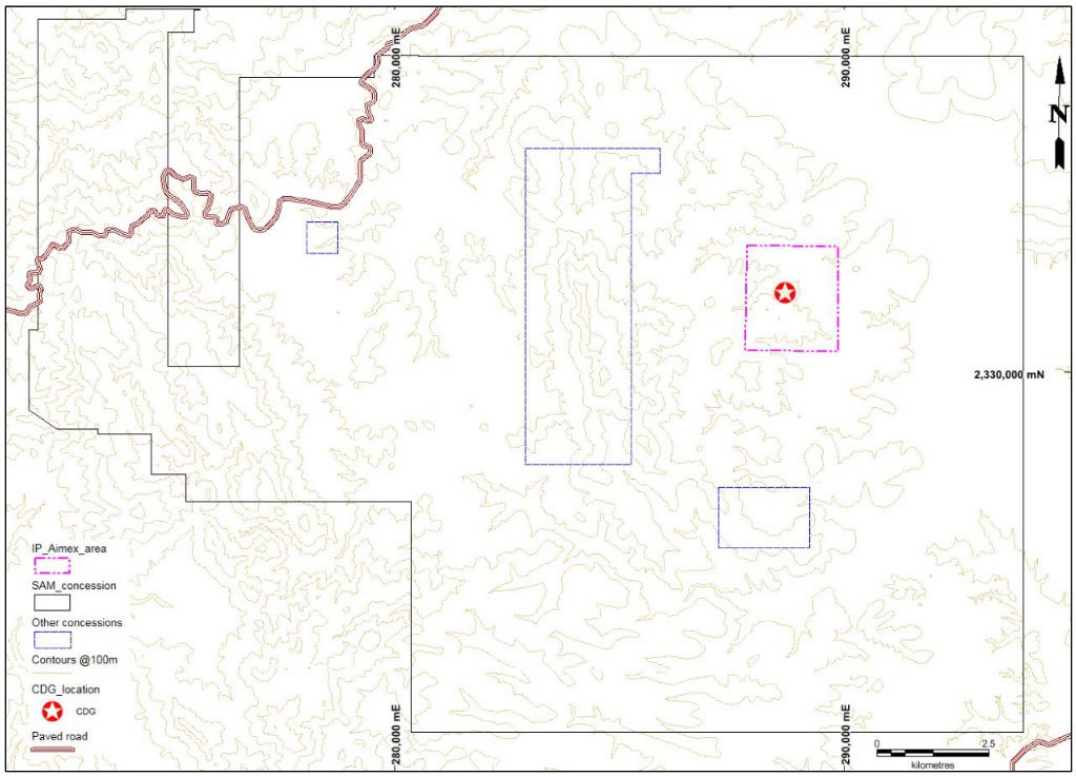
Source: Primero Mining (2016)

9.5 Geophysical Surveys

McPhar Geosurveys Ltd (McPhar Geosurveys, 2005) was contracted to conduct an airborne geophysical survey program on the San Anton Property beginning in late January 2005. The program consisted of a high-resolution helicopter magnetic and radiometric survey. The final survey flight was completed on 17 February 2005. A total of 3077 line-kilometers of data were acquired, covering an area of approximately 270 square kilometers. The survey area was flown in one block with a nominal mean terrain clearance of 30 meters on flight lines oriented east-west at a spacing of 100 meters and tie lines oriented north-south at a spacing of 1000 meters. Instruments included a Scintrex CS-2 High Resolution Cesium Magnetometer, a Pico Envirotec GRS410 Gamma Spectrometer with NaI(Tl) detector, a Terra TRA-3000/TRI-30 Radar Altimeter, and a DGPS Max positioning system.

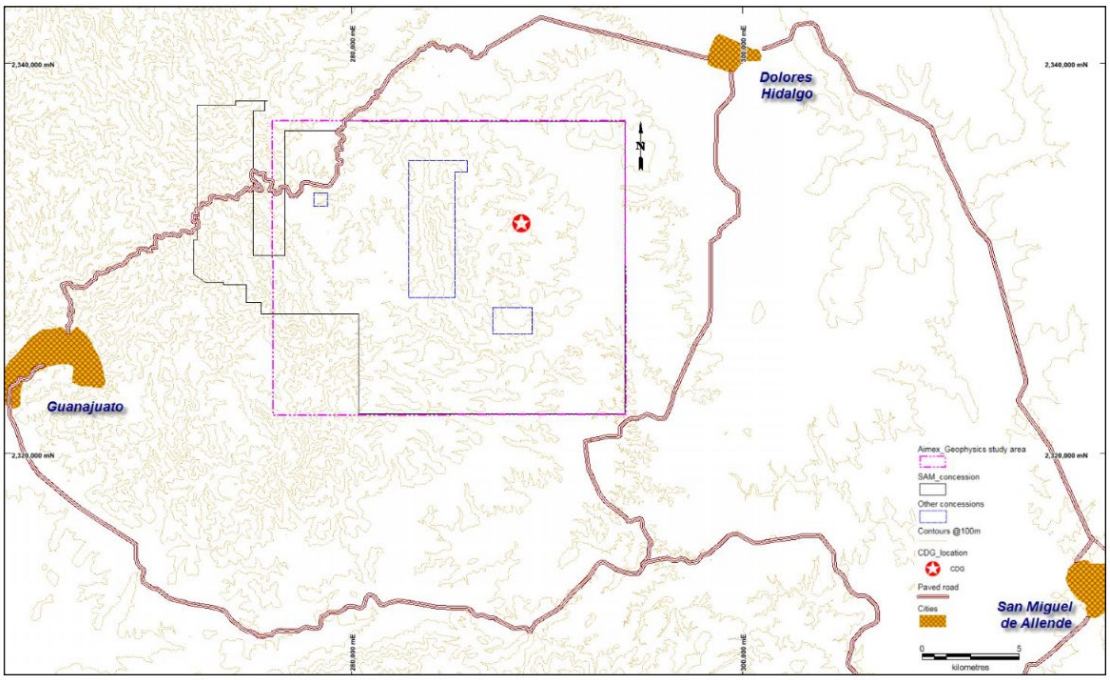
In June and July, 2005, SJ Geophysics (Vancouver) was retained to complete field data collection for a three-dimensional Induced Polarization survey over the Cerro del Gallo porphyry system (Swiridiuk, 2005a) (Figure 9.8) with a much broader airborne magnetics program (Figure 9.9). AIMEX Geophysics completed the modeling based on the field data. This work was aimed at identifying disseminated sulfides associated with skarns and other areas of potential gold and copper mineralization. AIMEX Geophysics went on to refine the models (Swiridiuk, 2005b).

Figure 9.8: Location of 3D Induced Polarization Study



Source: Primero Mining (2016).

Figure 9.9: Location of AIMEX Airborne Magnetics Study



Source: Primero Mining (2016).

Zonge International Inc. (Zonge) was contracted by previous operators in 2010 to conduct geophysical work. Their report was titled dipole-dipole Complex Resistivity IP (CRIP), Natural Source AMT, Gravity, Ground Magnetic and Radiometric geophysical surveys (Zonge, 2010). Argonaut obtained the report and raw data from this survey and included it in its geographical information system (GIS) database. Several chargeability anomalies were observed and will be used as part of the information for targeting drill holes at the Cerro de Gallo Project.

In 2013, Smailbegovic (2013) reprocessed the existing magnetic and radiometric data to suppress vegetation, outline structural and geomorphological features, correlate data with locations of observed anomalies, and matched geological models with these observations.

In 2015, Wave Geophysics (2015) used existing data to produce a MAG3D of the area. Information for this work is sparse with a single poorly documented slide deck. This work used ASTER data prepared by Western Mining Services (2014).

9.6 Petrology, Mineralogy, and Research Studies

Mason (2005a, 2005b. 2006a, 2006b, 2007) discuss results of various petrographic studies performed on CdG samples for Kings Minerals NL. Petrographic descriptions of five CdG samples are discussed in a report by Centro de estudios Mineralógicos (2015). Primero Mining had two samples examined at the University of Guanajuato. Results of that examination are discussed in Martínez Reyes et al. (2014).

9.7 Exploration Potential

In 2016, Primero, with the help of Western Mining Services) identified a number of areas with exploration potential in the region (Table 9.1).

Table 9.1: Cerro del Gallo Areas for Additional Exploration (east sector)

Target	*Target type*	*Stream sediments*	*Soil samples*	*Rock chips*	*Geologic/Geophysics Highlights*
WMS-06	Skarn	± Au, ± Ag, Cu, As, Pb, Zn, ± Bi, ± Co, Sb, ± Te, ± Se, ± Cd	±Ag, Al, ±As, ±Ba, Ca, Cu, Cr, Co, Cr, Fe, ±La, Li, Mg, Mn, ±Na, Ni, ±Sb, Sc, ±Ti, V, ±Y.	Cu, Co, Cr, Ni, V, Sc, Y	• Limestone package (Esperanza formation) of ≈ 300m thickness and 2.5 km long. • Evidences of skarn alteration (petrography).
San Isidro	Skarn	± Au, ± Ag, Cu, As, Pb, Zn, ± Bi, ± Co, Sb, ± Te, ± Se, ± Cd	±Au, ±Ag, Ba, Be, Ca, ±Cd, Cu, Co, ±Cr, Fe, ±La, ±Li, ±Mg, Mn, ±Na, ±Ni, ±Sb, ±Ti, ±V, W, Y, Zr.	Cu, Co, Cr, Ni, V	• Skarn mineralization/alteration on drillholes around CDG. • Felsic intrusive (CDG) and Tonalite-Granodiorite outcropping in target areas.
Silleros	Skarn	± Au, ± Ag, Cu, As, Pb, Zn, ± Bi, ± Co, Sb, ± Te, ± Se, ± Cd	±Ag, Ba, ±La, ±Na, Zr.	Ba, Cu, Co	• Weak Geophysical anomalies (RTM, AS)in the WMS-06 and San Isidro area.
San Juan Vianey	Skarn	± Au, ± Ag, Cu, As, Pb, Zn, ± Bi, ± Co, Sb, ± Te, ± Se, ± Cd	±Au, Ag, As, Ca, Cd, ±Hg, Sb, W	Ag, Cu, Co, Cr, Ni, V, W	
Espíritu Santo	Epithermal	±Au	Au,	Au, Ag, ±Pb,	• Structural flexure on sandstone domain and silicification alteration halo emplaced in Esperanza formation. • Several prospects and old workings. • Some rock chips values over 0.5 g/t Au and ag Values.
Santa Bárbara (La Tinaja)	Epithermal	±Au, ±As, ±Hg, ±Ni, Sb, ±W	Au, ±Ag, Ca, K, Sb, W,	Au, ±Hg, W	• Located along regional structural corridor of Esperanza formation. • Some rock chips results > 0.5 g/t Au, Ag rare.
San Luis Rey	Epithermal	±Au, ±As, ±Ni, Sb, ±W	Au, Ca, ±Cd, ±Mn, Sb, Y,	Au, ±As, ±Hg, Sb, V	
WMS-20	Epithermal	±Ba, Ga, La, Th, Tl, Te, ±Zn, U	±As, ±Sb, ±Ba	±Ba, ±La, ±Th	• Felsic dikes trend sub-parallel to Sierra and Veta Madre system (Guanajuato Mining district). • Tectonic window in Cedro andesite exposing Guanajuato conglomerate. • Ba-Th anomalies could represent top of an epitermal system. • 3D magnetic inversion shows an underground link with WMS-03.
WMS-03	Skarn	±As, ±Co, Bi, ±Sc, ±Se, Te, V	±Ag, As, Bi, Mo, ±Cu	Bi, ±Se, ±Ga, Te	
WMS-04	PCD/Intrusive	±As, ±Co, Bi, ±Sc, ±Se, Cu, Te, V	--	Bi, ±Se, ±Ga, Te	• Bi-Te geochemical anomalies. • Moderate (locally high) pyrite-chlorite alteration.
WMS-05	PCD/Intrusive	±As, ±Co, Bi, ±Sc, ±Se, Te, Zn, V	--	Bi, ±Se, ±Ga, Te	

Source: Primero, 2016.

East of CdG, areas for additional exploration include the following epithermal vein systems:

- Carmen-Providencia vein system;
- Ave de Gracia vein system.

These targets remain largely untested. Cursory review of the data by the QP suggests that these are reasonable targets that should be tested.

9.8 QP Comments on Section 9

Exploration methods and procedures at CdG are consistent with common industry practices. There are a number of geochemical and geophysical anomalies that warrant additional exploration. The QP is of the opinion that significant exploration potential exists on the property.

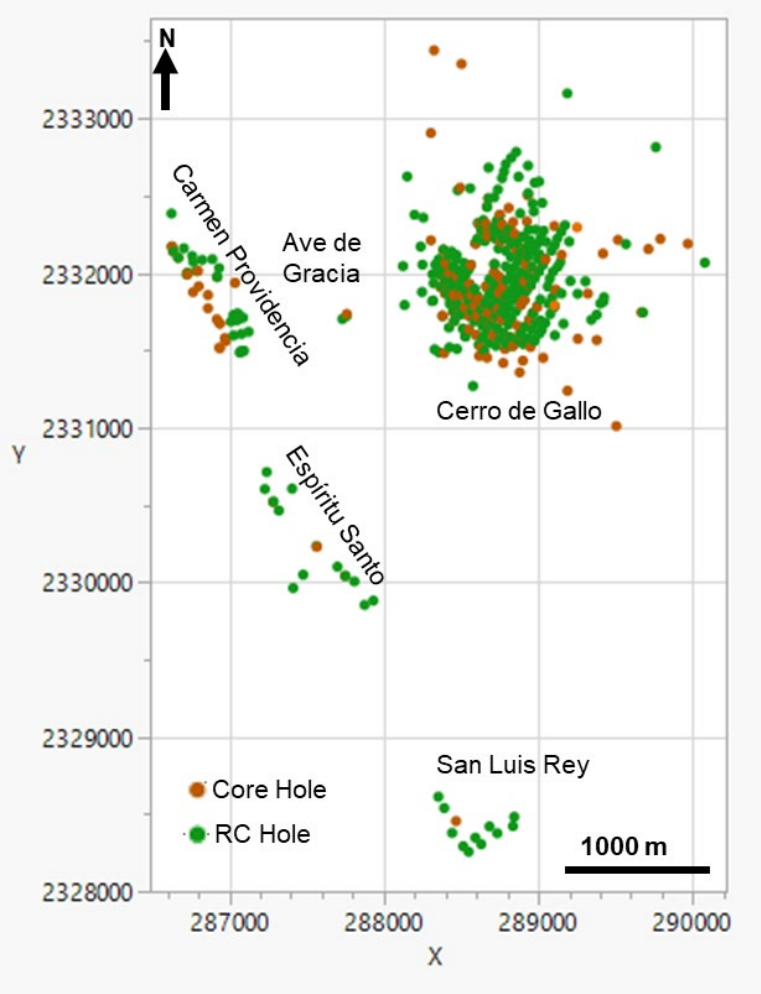
10 Drilling

10.1 Introduction

Drilling was completed mostly in 2004 through 2008 by Cerro Resources (359 holes; 97,460.77 m). In 2013, Primero drilled 51 holes (12,829.16 m). In 2018, Argonaut drilled 18 holes (1,484.50 m) primarily to collect material for metallurgical sampling. Table 10.1 summarizes the known drilling.

Figure 10.1 shows the locations of core and RC drill holes at CdG. No other drilling is known on the property.

Figure 10.1: Drill Hole Location Map



Source: MTS (2025)

Table 10.1: Drill Summary by Company, Year, and Drill Type

Company	Year	RC Holes	RC Meters	Core Holes	Core Meters	Total Holes	Total Meters
Cerro	2004	3	335.28			3	335.28
	2005	80	14,884.92	9	5,402.95	89	20,287.87
	2006	109	24,005.97	31	12,935.80	140	36,941.77
	2007	63	13,838.33	30	15,452.17	93	29,290.50
	2008	27	7,036.30	7	3,569.05	34	10,605.35
Primero	2013			51	12,829.16	51	12,829.16
Argonaut	2018			18	1,484.50	18	1,484.50
Total		282	60,100.80	146	51,673.63	428	111,774.43

Source: MTS (2025)

10.2 Drilling Used in Mineral Resource Estimation

All of the data in the project database were used in various aspects of the mineral resource estimate.

The drilling used for the CdG resource model is limited to the CdG hill. Table 10.2 summarizes the area extents.

Table 10.2: Limits for the Area Extents of CdG

	Minimum	Maximum
Easting	287500	290000
Northing	2330500	2333000

Source: MTS (2025)

10.3 Drill Methods

Drilling is a combination of reverse circulation (RC) and core (using diamond tools). Table 10.1 summarizes the methods and amounts of drilling.

10.3.1 Reverse Circulation (RC) Drilling

Cerro Resources drilled 282 holes, a total of 60,100.8 m, using reverse circulation (RC) drills and methods. Cerro utilized an all-terrain articulated “buggy” drill rig (large rubber-tired), a Prospector W-750. Hole diameters ranged from 4¾ inch to 5½ inch using face return drill bits and double-walled pipe. Almost all of this RC drilling was angled. Most of the RC drilling was less than 200 meters deep. The maximum depth drilled with RC was around 350 m. At greater depths, and where higher air pressure was required, Cerro implemented a secondary booster. It is not documented but Cerro apparently drilled dry both above and below the water table. Below the water table, Cerro increased air

pressure and volume to maintain high sample quality. In general, groundwater inflow rates were low and easily manageable; however, Cerro reported high groundwater flow rates in a few areas, generally narrow fracture zones, that resulted in abandonment of some drill holes.

Sampling was conducted using a cyclone followed by hand-splitting through a free-standing three tier Jones-type riffle splitter.

Based on review of the records, examination of RC cuttings in sample trays, it is the opinion of this QP that Cerro conducted drilling and sampling to acceptable standards, using industry accepted practices, and that the sample quality was in general very good.

10.3.2 Core Drilling

All core drilled from 2004 through 2008 by Cerro was oriented core. Drilling used an Atlas Copco CS-1500 truck-mounted core drill. Normally the holes were started with HQ (63.5 mm) sized core and, if necessary, reduced to NQ sized core (47.6 mm).

From 2013 through 2014, Primero's core drilling program was performed by Major Drilling with a Major 50 truck mounted drill using diamond tools. These holes were positioned in areas requiring closer spaced drilling, often because the original RC holes were not completed due to high water content or where the RC holes were lost in mineralization. Primero's drilling also included PQ (85 mm) sized core holes to twin mineralized RC drill holes. Core handling and logging procedures for Primero were identical to those of Cerro.

In 2018 Argonaut drilled 18 PQ-size core holes. This program, totaling 1,484.5 meters, was designed to obtain mineralized material for metallurgical studies. Argonaut utilized a UDR 200 drill rig supplied by Major Drilling. This core drilling program was intended entirely to produce metallurgical test samples. As such, Argonaut twinned existing drill holes, and sent whole PQ core to the lab.

Standard core handling procedures initiated by Cerro Resources were utilized throughout all three core drilling campaigns at CdG. Core cutting was done by saw followed by industry standard sampling procedures. Argonaut; however, did not sample the core for standard assays as noted above.

The procedures were as follows:

- The core was removed from the tube and re-configured to its original in situ state to form a continuous cylinder;
- Core was marked up with a longitudinal line on the "south side" for sawing;
- Core was photographed in wet and dry mode in natural light;

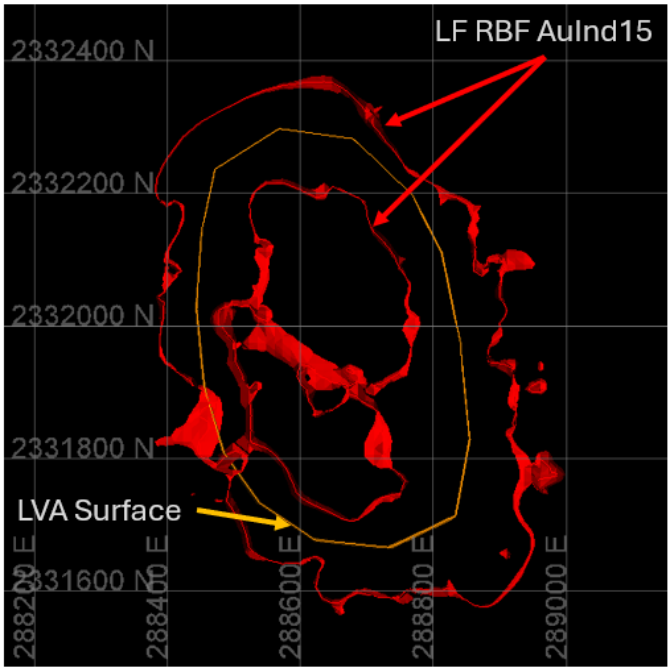
- Geotechnical logging, including core recovery and RQD, was completed;
- Geological logging was completed including lithological, alteration, structural, and mineralization logging;
- Core was then sawed in half;
- One half was sent for assay and the other half retained in the original core box.

10.3.3 Drilling Configuration

Drilling was predominantly performed on nominal 50-meter fences, oriented NNE- SSW to intersect a known north-westerly trending structural corridor. Holes were normally angled between 55 and 65º, either to the northeast or southwest, depending on the position of the drill fence (or drill section). Specifically, drill fences were oriented to alternate every other section between NNE bearing and SSW bearing. These holes can locally crisscross at depth. Drill spacing within the resource area averages approximately 50 meters.

Mineralization generally displays a concentric distribution around the carapace of the intrusive as can be seen in Figure 10.2. To a lesser extent, mineralization displays a secondary NNW orientation, following the dominant structural trend through the region. The drilling pattern utilized to define the CdG resource, the NNE-SSW fences, is well-oriented to the NNW mineralization trend and to a lesser extent, the concentric mineralization. During initial exploration activities the presence of a concentric mineralization is not only difficult to detect but also the intrusive centroid is near impossible to define. The systematic drilling to date has proven advantageous in understanding the mineral distribution and geology. Now that the concentric orientation is confirmed, additional definition drilling will consider a radial type drilling orientation (from the intrusive limbs towards the centroid) to more optimally intersect the direction of greatest grade variance.

Figure 10.2: Plan (2,100 m level) Showing Leapfrog RBF Indicator Shell using a 0.15 g/t Grade Threshold



Source: MTS (2025)

10.4 Logging

RC drill samples obtained by CdG were logged at five-foot (1.52-meter) intervals on-site at the time of drilling. A split from each interval was sieved, washed and logged visually, using a hand lens. Logging criteria included: lithology, alteration, degree of oxidation, and mineralization. Summary log sheets were prepared on site at the conclusion of each hole. Representative chips from each five-foot (1.52-meter) interval were collected and retained in plastic sample chip trays stored at the San Antón field office. Drill hole data were recorded on handwritten logs onto a pre-printed log sheet template database and eventually merged with assay results. Drill data included hole identification, coordinates (in NAD27 for Mexico), depth, and sample number, including duplicates, blanks and standards.

Core handling and logging procedures for Primero were identical to those described above for Cerro.

Argonaut continued using the same core handling and logging procedures implemented by Cerro. Drill core went through standard logging procedures to obtain geotechnical and geologic data such as lithology, alteration, and mineralization. Special emphasis was given to recording the degree of oxidation of the core material.

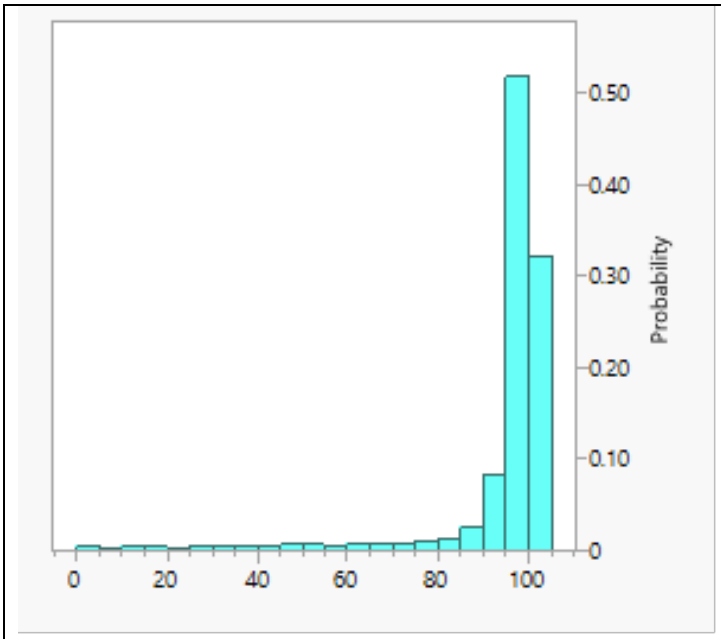
Density samples (whole core) were collected at approximately 30-meter intervals in select holes, through the entire hole.

10.5 Recovery

Data for core recoveries were not recorded for all drill programs. Data for the 2013 core drilling program are summarized in the histogram in Table 10.3. The data indicate that 92% of the intervals had better than 90% recovery which is considered by the QP to be quite acceptable. The QP reviewed some core from all drill programs and is of the opinion that the 2013 recovery is representative of all recovery, but because all the core has been split, it is not possible to recover reliable recovery data.

The QP is not aware of any recovery data for RC samples.

Table 10.3: Histogram of Core Recovery for 2013



Summary Statistics		
Mean		95.43
Std Dev		11.61
Std Err Mean		0.165
Upper 95% Mean		95.75
Lower 95% Mean		95.11
N		4924
N Missing		0
Minimum		0
Maximum		101
Median		98.36
10% Trimmed Mean		98.01
Quantiles		
100.0%	maximum	101
99.5%		100.33
97.5%		100
90.0%		100
75.0%	quartile	100
50.0%	median	98.36
25.0%	quartile	96.38
10.0%		92
2.5%		60
0.5%		15.41
0.0%	minimum	0

Source: MTS (2025)

10.6 Collar Surveys

Drill collar positions were established by hand held GPS units. Drill directions and dip angles were set up using a Brunton compass.

After completion of the hole, final hole locations were determined by a contract surveyor using a total station instrument. Each drill hole was then marked with a square concrete cap (approximately 40 by 40 cm) inscribed with the drill hole number.

Regional topography is modelled from a LiDAR aerial survey and ground control points. The original, pre-mining topography was provided in DXF format from which a percentage block not in air is calculated and used to estimate current Mineral Resources.

10.7 Downhole Surveys

Down-hole surveys were completed for all core and RC holes on the project. Initially, down-hole surveying was done using an Eastman single shot camera. After 2004, a Reflex EZ-Shot electronic solid-state single shot survey tool was implemented. Surveys were normally completed just below casing and at various depths (depending on the rate of change in azimuth and/or inclination), and at the bottom of the hole. Core holes were surveyed as drilling proceeds at nominal 30-meter intervals.

10.8 Oriented Core

All core drilled by Cerro was oriented core using the Ballmark® orientation system. Core was collected from the drill site and taken to a logging facility adjacent to the Dolores Shaft. The core was washed, photographed and “marked up” as described above. The core was then geotechnically and structurally logged to identify hardness, weathering, fractures, core dip angle and core dip direction. Ballmark® imprints were used to assist in determining vein and fracture orientations (dip and strike) with the assistance of a goniometer. After geotechnical data collection, the core was geologically logged, cut and sampled as described elsewhere in this section.

10.9 Drilling Since Database Close-Out Date

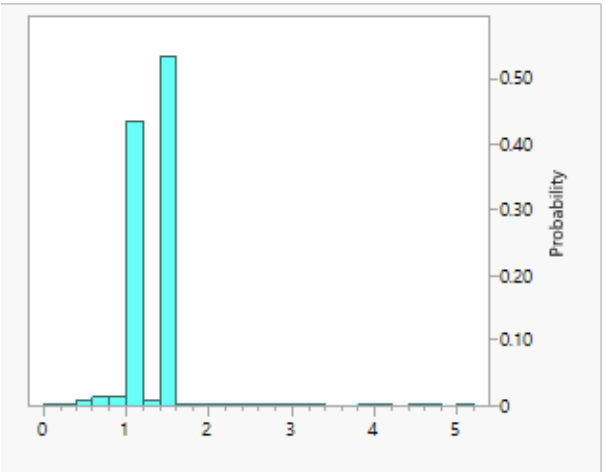
No drilling was completed after the database was closed.

10.10 Sample Length/True Thickness

Figure 10.3 summarizes sample lengths which are dominantly 1 m (43%) or 1.52 m (5 ft; 53%). These are appropriate for the style of mineralization and the envisioned mining method.

Because this is essentially a porphyry-type deposit with disseminated mineralization and large areas of stockwork mineralization to be mined by open pit methods, the concept of true thickness is largely not relevant. Some veins have been encountered, but those will be mined as part of the open pit.

Figure 10.3: Histogram of Sample Lengths



Source: MTS (2025)

10.11 QP Comments on Section 10

Drilling at CdG was completed using methods and procedures that are commonly used in the industry. Collar surveys, downhole surveys, and core orientation were completed using instruments that are widely used in the industry.

Geological logging was performed by two operators using two different nomenclatures that are not consistent. The QPs and Heliostar personnel used the data for lithology and oxide state to generate new descriptors. The QPs consider this to be an acceptable “fix” for this stage of the project, but recommend that all core and RC cuttings be relogged using a standard nomenclature.

Sample lengths are appropriate and consistent with industry procedures.

The QP recommends that all available core be relogged using consistent nomenclature. A second phase of relogging would include checking available RC chip trays to conform to the core nomenclature.

The QP also recommends that all available core drilling be relogged for oxidation type and subsequently all RC chip trays be checked to conform to the oxidation nomenclature.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling

Once geological logging was completed (including lithology, alteration, structural, and mineralization logging), the core was cut into halves with a specially modified tile saw. One half of the core was sent for assay and the other half was retained in the original core box and stored on site.

11.2 Density Determinations

In a 2012 Technical Report completed by Cerro, it was reported that 895 HQ core samples were selected for density determination. Samples were selected to represent the main rock types that occur within the deposit. Samples (whole core) were collected at approximately 30 m intervals through select core holes with core sample lengths averaging 13 cm. The caliper method was used as follows:

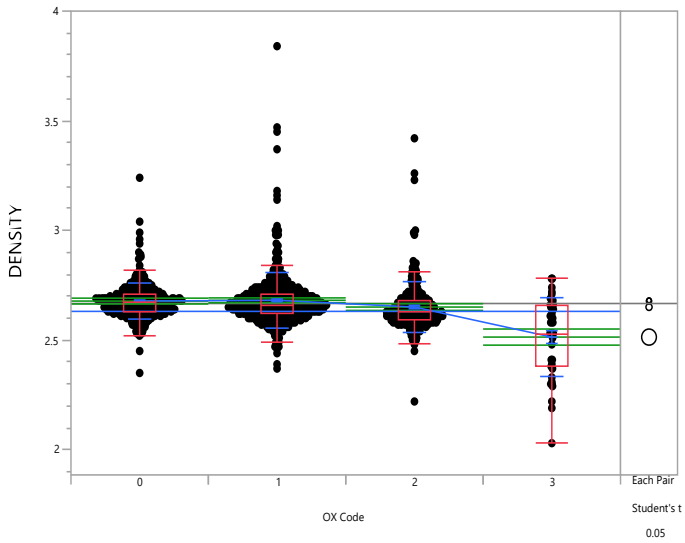
1. The ends of the core were cut with a saw perpendicular to the longitudinal axis of the core;
2. The core was then dried;
3. The diameter of the core was measured with a digital caliper at three points and averaged;
4. The length of the core was measured with a steel tape measure;
5. The mass of the dry core was determined using scientific scales with a quoted accuracy of ±0.01 gram.

In 2018, Argonaut measured density in the upper more oxidized profiles of the resource. This work consisted of collecting an additional 47 samples with the aim of increasing the number of density measurements in the near surface oxide profiles. Prior to this the oxide profile contained only two density measurements but potentially represented 5 – 10% of the resource ounces.

Error! Reference source not found. summarizes density by oxide state. Note that the means are not exactly equivalent to means in

Table 11.1 because the data were, in part, recoded by the QP.

Figure 11.1: Boxplot of Density by Oxide State



	Sulfide	Mixed Sulfide	Mixed Oxide	Oxide
Code	**0**	**1**	**2**	**3**
Number	290	421	217	38
Mean	2.68	2.68	2.65	2.51
Std Dev	0.082	0.127	0.116	0.180
Std Err Mean	0.005	0.006	0.008	0.029
Lower 95%	2.67	2.67	2.64	2.45
Upper 95%	2.69	2.69	2.67	2.57

Source: MTS (2025)

Table 11.1 shows the number of data by oxidation profile and average densities. This data shows that the 2.65 g/cm^3 applied value is approximately 1% lower than the average density for the deepest sulfide profile. The previous mineral resource estimate used density values adjusted (discounted) to account for fracture porosity and sample selection bias. The QP considers the concept of discounting density values to be of value, but that the discount values require mining experience where true bulk densities can be determined by mined volumes and tonnages. The method used in the previous mineral resource estimates is not considered to be reliable so raw density data were used for this estimate.

Table 11.1: Density Measurements within Block Model Project Limits

Oxidation Domain	Number	Density (g/cm^3)	Standard Deviation	Coefficient of Variation	Porosity Discount
Oxide	16	2.41	0.24	0.1	3.8%
Oxide-mix	32	2.62	0.11	0.04	1.5%
Sulfide-mix	120	2.64	0.12	0.05	1.2%
Sulfide	612	2.68	0.13	0.05	1.0%
Total	**780**	**2.66**	**0.13**	**0.05**	**1.1%**

Source: PEA (2021)

11.3 Sample Preparation and Analytical Laboratories

Sample preparation was completed at SGS in Durango, Mexico. Analyses were completed at either SGS Durango or SGS Toronto (Table 11.2).

The most common, and important accreditation for an assay laboratory is the ISO/IEC 17025:2017 accreditation which requires that the laboratory complete competence of testing and calibration for specific methods used by the laboratory.

The SGS sample preparation facilities are not ISO 17025 accredited but accreditation documents note that "The physical sample preparation involving accredited test method for Minerals Analysis as listed on the Scope of Accreditation may be performed at the SGS Mexico, SGS Minerales – Durango, SCC File 15831 location or at off-site sample preparation laboratories that are monitored regularly for quality control and quality assurance practices:" SGS Hermosillo is listed as an off-site sample preparation facility. SGS Durango was first ISO17025 accredited by the Standards Council of Canada (SCC) in late 2009 (SCC File Number 15831). The ISO 17025:2017 accreditation is valid to late 2029. Specific tests covered by the accreditation include:

- AAS33E50 – Determination of Silver and Copper in Exploration Grade Samples by Three Acid Digest and Atomic Absorption Spectroscopy;
- FAA30V5 – Determination of Gold in Exploration Samples by Lead Fusion Fire Assay and Atomic Absorption Spectrometry [30g];
- GE_ICP21B20 – Multi-element Determination in Exploration Grade Samples by Aqua Regia Digestion and Inductively Coupled Plasma Optical Emission Spectrometry [ICP-OES; Ag, Al, As, Ba, Be, Bi, Ca, Cd, Cr, Co, Cu, Fe, Hg, K, La, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sn, Sr, Ti, V, W, Y, Zn, Zr];

- GE_ICP40Q12 – Multi-element Determination in Exploration Grade Samples by Four Acid Digestion and Inductively Coupled Plasma Optical Emission Spectrometry [ICP-OES; Ag, Al, As, Ba, Be, Bi, Ca, Cd, Cr, Co, Cu, Fe, K, La, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sn, Sr, Ti, V, W, Y, Zn, Zr];
- GO_FAG30V – Determination of Ore Grade Gold by Lead Fusion Fire Assay and Gravimetric Finish [30g];
- GO_FAG50V – Determination of Ore Grade Gold by Lead Fusion Fire Assay and Gravimetric Finish [50g];
- GO_FAG37V - Determination of Ore Grade Silver by Lead Fusion Fire Assay, Gravimetric measurement and Gold by Lead Fusion Fire Assay and Atomic Absorption Spectrometry [30g];
- GO_ICP90Q100 – Determination of Various Elements in Ore Grade Samples using Sodium Peroxide Fusion and Inductively Coupled Plasma Optical Emission Spectrometry [ICP-OES; As, Fe, Cu, Ni, Pb, Sb, Zn];
- GO_AAS21C50 – Determination of Metals in Ore Grade samples using Aqua Regia digestion and Atomic Absorption Spectrometry [Cu, Ag].

The Toronto SGS analytical facility is ISO 9002 registered and ISO 17025:2017 accredited for specific tests (SCC No. 456). The first ISO 17025 accreditation was in March 1995 (SCC File Number 15254). Specific tests at SGS Toronto include all of the tests noted for SGS Durango above and numerous additional methods not relevant to this project.

Table 11.2: Locations of Sample Preparation and Assays by Year

Year	Sample Preparation	Assay
2005	Durango	SGS Toronto – all assays
2006	Durango	SGS Toronto – all assays
2007	Durango	SGS Toronto – all assays (before about 15 Oct) SGS – Durango – FAS313 (Au), FAG303 (Au) (after about 15 Oct)
2008	Durango	SGS – Durango – FAS313 (Au), FAG303 (Au) SGS – Toronto – ICP40B (multielement), AAS21E (Ag), AAS21E (Cu), CSA06V (S)
2013	Durango	SGS – Durango – FAA313 (Au), AAS21E (Ag), ICP14B - multielement
2014	Durango	SGS – Durango – FAA313 (Au), AAS21E (Ag), ICP14B - multielement

2021	None (pulps only)	SGS – Durango – GC_CSC67C (CN soluble copper)

Source : MTS, 2025

11.4 Sample Preparation

Sample preparation for all drill and surface samples generated by Cerro and Primero was completed at the SGS sample preparation facility in Durango, Mexico. Sample preparation included:

- Samples were oven dried at 105ºC;
- RC and core samples were crushed to 90% passing 2 mm;
- A 1,000 g sub-sample was split for pulverizing;
- The 1,000 g sub-sample was then pulverized to 90% passing 75 µm (200 mesh);
- From the pulverized 1,000 g subsample, a pulp of approximately 200-250 g was taken for analysis.

11.5 Analysis

Table 11.3 summarizes the analytical history of the project by company, year, and method.

11.5.1 December 2004 to October 2013

Drill samples assayed from 2004 until October 2005 were analyzed for gold, silver, copper, lead, zinc, molybdenum and bismuth. Gold was determined by fire assay using a 30 g sample pulp with instrumental Atomic Absorption Spectrometry (AAS) finish (SGS Method FAA313). The lower detection limit of this method was 5 ppb with an upper limit of 10,000 ppb. All samples assaying greater than 1,000 ppb gold were reassayed by fire assay using a 30 g sample pulp with a gravimetric finish (SGS Method FAG303). This is an “ore grade” procedure for higher gold values. The lower detection limit for this method is 0.03 ppm.

Starting in November 2007, fire assays for gold were done at the same SGS Durango facility as was sample preparation.

Prior to November 2005, silver and copper values were determined using SGS Method AAS40E which incorporates multi-acid digestion and an AAS finish on a 2.0 g nominal weight sample. After November 2005, silver and copper were determined using SGS method AAS21E consisting of a three-acid digestion with AAS finish on 2 g sample and is applicable for material containing less than 300 ppm silver and less than 10,000 ppm copper. This method has a lower detection limit for silver of 0.5 ppm. Silver values over 300 ppm were determined by one of two methods. One method, AAS21E consists of 3-acid digestion with AAS finish on a 2 g sample. The other method, AAA50, mainly applies to a silver range between 300-1,000 ppm. With this method, a 2.0 g sample undergoes four-acid digestion and is then finished using AAS. If the sample has more than 1,000 ppm silver, a fire assay overlimit method is used (Method FAG303). Prior to November 2005, all silver analyses greater than 300 ppm were re-analyzed by lead collection fire assay using a 30 g sample and gravimetric finish. The lower detection limit for silver using this method is 3 ppm. During this same period samples containing silver values between 300 ppm and 1,000 ppm were assayed by SGS Method AAA50. Copper values were also determined by the same method with a lower detection limit of 0.5 ppm and an upper range limit of 10,000 ppm. Copper values over 10,000 ppm (“ore grade”) were determined by sodium peroxide fusion/Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES) finish on the solution (SGS Method ICA50).

Prior to October 2005 drill samples were routinely assayed for lead, zinc, molybdenum and bismuth with values determined by ICP-OES on a 0.2 g sample using SGS Method ICP40B which uses a four-acid digestion. This procedure has a lower detection limit for lead of 2 ppm, zinc 0.5 ppm, molybdenum 1 ppm, and bismuth 5 ppm. The upper limit for

lead and zinc values using this method is 10,000 ppm. Over-range base metal elements, including copper, were reassayed by ore grade analysis Method ICA50. This method has a lower detection limit of 0.01%, and no upper limit.

Table 11.3: Summary of Analytical Methods by Year and Company

Company	Cerro Resources			Primero Mining		
Lab	SGS			SGS		
	Method	From	To	Method	From	To
Routine Assays						
Au	FAA313	Dec-2004	Dec-2008	FAA313	2013	2014
Ag	AAS40E	Dec-2004	Oct-2005	ICP14B	2013	2014
	AAS21E	Nov-2005	Dec-2008			
Cu	AAS40E	Dec-2004	Oct-2005	ICP14B	2013	2014
	AAS21E	Dec-2004	Dec-2008			
Overlimit Assays						
	Method	**From**	**To**	**Method**	**From**	**To**
Au	FAG303	Dec-2004	Dec-2008	FAG303	2013	2014
Ag	AAA50	Nov-2005	Dec-2008	AAS21E	2013	2014
	FAG303	Dec-2004	Dec-2008	FAA313	2013	2014
Cu	ICA50		Nov-2005	ICP90Q	2013	2014

Notes:
FAA313 - Gold - fire assay using a 30 g sample pulp with AAS finish. Lower detection limit 5 ppb, upper limit of 10,000 ppb.
AAS40E – Silver and Copper - multi-acid digestion with AAS finish on a 2.0 g sample.

AAS21E – Silver – over 300 ppm, 3-acid digestion with AAS finish on a 2 g sample.

ICP14B - standard suite of 32 elements using ICP analysis.
FAG303- Gold – plus 1,000 ppb fire assay using a 30 g sample pulp with a gravimetric finish. Lower detection limit of 0.03 ppm.

AAA50 – Silver - 300-1,000 ppm, 2.0 g sample undergoes four acid digestion and AAS analysis.
ICA50 - Copper – plus 10,000 ppm sodium peroxide fusion/Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES) finish.

ICP90Q – Base metals greater than 10,000 ppm using ICP analysis, lower detection of 0.01%.

Source: Defilippi et al., 2020

11.5.2 January 2013 to Present

After October 2005, drill samples were assayed for a standard suite of 32 elements using SGS Method ICP40B. More recently, over range base metals greater than 10,000 ppm were analyzed using the SGS ore grade analysis method ICP90Q, which has a lower detection of 0.01%.

All 2013 drill core samples generated by Primero were assayed for gold using SGS fire assay method FAA313 which utilizes a 30 g sample pulp with an AAS finish. The lower detection limit for this method is 0.01 ppm and the upper limit is 10 ppm which covers the majority of gold values encountered at CdG. All samples with assays greater than 10 ppm gold were reassayed by fire assay on a 30 g sample pulp using a gravimetric finish (SGS method FAG303). This is an “ore grade” method for high gold values. The lower detection limit for this method is 0.5 ppm and the upper detection limit is 10,000 ppm.

Primero also assayed their 2013 core samples for silver and copper (35 element package) using an aqua regia (HNO_3-HCl) digestion with an ICP finish, (SGS method ICP14B). This method utilizes a 0.5-gram sample and is appropriate for material with organic, oxidized, or high sulfide mineral content but it is the weakest of the SGS digestions and does not attack silicate minerals. It is not appropriate for ore-grade determinations. For silver, the lower detection limit is 2 ppm and the upper detection limit is 100 ppm. All samples with assays greater than 100 ppm silver were reassayed by SGS method AAS21E (2 g sample, 3-acid digestion with AAS finish). The upper detection limit for this method is 300 ppm silver. All samples with assays greater than 300 ppm silver were reassayed by fire assay on a 30 g sample pulp using a gravimetric finish (SGS method FAG313). This is an “ore-grade” method for high silver values. The lower detection limit for this method is 10 ppm and the upper detection limit is 10,000 ppm. For copper, the lower detection limit for SGS method ICP14B is 1 ppm and the upper detection limit is 10,000 ppm. All samples with greater than 10,000 ppm copper were reassayed by SGS method ICP90Q. This “ore-grade” method utilizes a 0.5 g sample, sodium peroxide fusion digestion, and an ICP finish. The lower detection limit for this method is 0.1% and the upper detection limit is 30%.

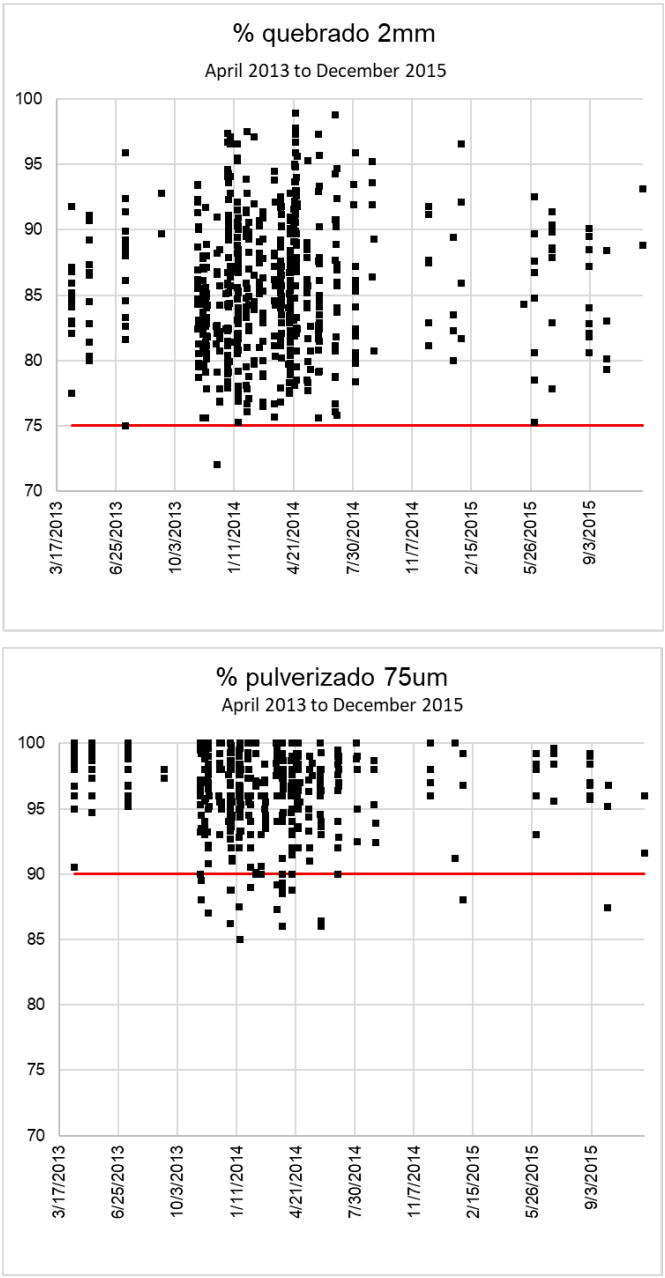
Check samples submitted to ALS Minerals in 2014 by Primero utilized comparable gold, silver, and copper analytical techniques as described above. Gold was analyzed by ALS method AA23 (30 g sample, fire assay digestion, AAS finish). All samples with greater than 10 ppm gold were reassayed by ALS method GRA21 (30 g sample, fire assay digestion, gravimetric finish). Silver and copper were analyzed by ALS method ICP41 (0.5 g sample, aqua regia digestion, ICP finish). All silver and copper samples with values at the upper detection limit of method ICP41 were analyzed by ALS method OG46. This “ore-grade” method utilizes a 0.4 g sample, aqua regia digestion, and ICP finish.

11.6 Quality Assurance and Quality Control

During 2013-2015 drilling programs, samples were sieved were periodically sieved to ensure that the required size distribution was being attained.

summarizes the results of sieve testing 2 mm and 75 µm product. Both show acceptable results.

Figure 11.2: Summary of Sample Preparation QC for 2013-2014



Source: MTS (2025)

11.6.1 Duplicate Samples

Results for field duplicates for 2005-2009 are summarized in Figure 11.3. The QP uses the 90th percentile of the absolute relative difference (ARD) as an estimator of precision. Typically, a cutoff of 30 times lower detection limit (LDL) is chosen so that only data above 30x LDL are used. In these cases:

- FAA313 used 20 x LDL in order to capture more data;
- FAG303 utilized all of the data;
- Ag_ppm_AAS21E used 20 x LDL in order to capture more data;
- Cu_ppm_AAS21E used 30 x LDL.

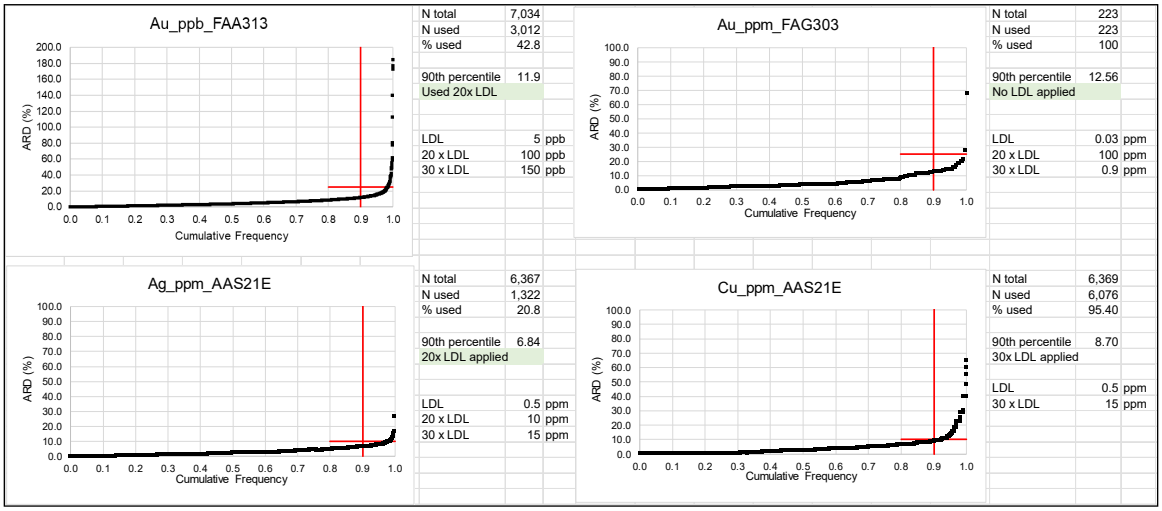
The results all show that precision is better than expected:

- FAA313 used about 43% of the available data to attain 20 x LDL. Precision is about ±12% which is much better than the anticipated ±25%;
- FAG303 used 100% of the available data. Precision is about ±13% which is much better than the anticipated ±25%;
- Ag_ppm_AAS21E used about 21% of the data to attain 20 x LDL. Precision is estimated to be about ±6.8% which is well within the anticipated bounds of ±10%;
- Cu_ppm_AAS21E used 95% of the data to attain 30 x LDL. Precision is estimated to be ±8.7% which is within the generally accepted bounds of ±10%, but the QP anticipated that precision would be better than ±5% so there may be a problem with Cu analyses or sample preparation that should be addressed in future exploration programs.

Overall, these results show that precision for the 2005-2009 analytical period were within reasonable limits and adequate to support mineral resource estimation and mine planning.

There are no duplicate data for the 2013-2014 analytical work.

Figure 11.3: Field Duplicate Summary



Source: MTS (2025)

11.6.2 Standards

Standards were routinely inserted into the sample stream every 40 samples.

The QP has results from 2013-2014 SGS work (Table 11.4). Those results are generally acceptable. CVs, a measure of precision, are reasonable. Bias, a measure of accuracy, comparing the SGS results to the best value (BV) for the standards are all within ±5% which is a generally accepted limit in the industry.

Table 11.4: Standards Summary For 2013-2014

Standard	N	Mean	Std Dev	CV (%)	BV	Bias
CDN-CM-30_Au	178	1.319	0.0818	6.20	1.3	1.43
CDN-CM-30_Ag	178	16.4	0.94	5.74	15.9	3.24
CDN-CM-30_Cu	178	7214	233	3.24	7300	-1.18
CDN-CM-27_Au	43	0.640	0.0541	8.46	0.636	0.56
CDN-CM-27_Ag	43	2.0	0	-	-	-
CDN-CM-27_Cu	43	5831	140	2.41	5920	-1.50
CDN-CGS-30_Au	42	0.322	0.0338	10.5	0.338	-4.68
CDN-CGS-30_Ag	44	8.7	0.63	7.26	-	-
CDN-CGS-30_Cu	44	1547	47	3.03	1540	0.44
CDN-GS-2K_Au	46	1.975	0.1322	6.70	1.97	0.24
CDN-GS-2K_Ag	46	2.0	0	0.00	-	-
CDN-GS-2K_Cu	45	33.4	2.8	8.3	-	-

Source: MTS, 2025

Table 11.5 summarizes standard results for the ALS check assay work. CVs are generally acceptable; however, three of the standards have only three or four results and for those standards, the CV is not a reliable measure of precision. Bias is generally better than ±5% and in all cases is better than ±10%. Bias for samples with only three or four results is not a reliable measure of accuracy although it does provide a general idea of accuracy.

Table 11.5: Standards Summary For ALS Check Assays

	N	Mean	Std Dev	CV	BV	Bias
CDN-CM-27_Au	3	0.624	0.0420	6.74	0.636	-1.89
CDN-CM-27_Ag	3	1.1	0.26	24.05	-	-
CDN-CM-27_Cu	3	5870	255	4.35	5920	-0.84
CDN-CM-30_Au	15	1.300	0.0672	5.17	1.3	-0.03
CDN-CM-30_Ag	15	16.8	0.78	4.63	15.9	5.37
CDN-CM-30_Cu	15	7469	211	2.82	7300	2.31
CDN-CGS-30_Au	4	0.367	0.0282	7.68	0.338	8.65
CDN-CGS-30_Ag	4	8.8	0.350	3.97	-	-
CDN-CGS-30_Cu	4	1593	34.0	2.14	1540	3.41
CDN-GS-2K_Au	4	2.024	0.1390	6.87	1.97	2.73
CDN-GS-2K_Ag	4	2.2	0.10	4.40	-	-
CDN-GS-2K_Cu	4	60	2.9	4.85	-	-

Source:

These data indicate that, in 2013-2014, SGS data were sufficiently accurate to support mineral resource estimation and mine planning. The ALS data suggest that ALS was sufficiently accurate to provide a reliable estimate of accuracy of the SGS data.

The QP did not discover any 2005-2008 standards data. Summaries of those data in previous reports (

Table 11.6) indicate that the 2005-2008 analytical data are sufficiently accurate to support mineral resource estimation and mine planning, but the QP has not verified those data.

Table 11.6: Summary of 2005-2008 Standards Data

Standard	Element	Units	N	Best Value	Mean value	Bias
OREAS 15Pa	Au	ppm	268	1.02	1	-1.96
OREAS 15Pb	Au	ppm	213	1.06	1.11	4.72
OREAS 15Pc	Au	ppm	248	1.61	1.66	3.11
OREAS 50P	Au	ppb	143	727	712	-2.06
	Cu	%	-	0.691	0.69	-0.14
OREAS 50Pb	Au	ppb	186	841	844	0.36
	Cu	%	-	0.744	0.697	-6.32
OREAS 51P	Au	ppb	312	430	426	-0.93
	Cu	%	-	0.728	0.704	-3.30
OREAS 52Pb	Au	ppb	45	307	314	2.28
	Cu	%	-	0.334	0.3	-10.18
OREAS 53P	Au	ppb	366	380	381	0.26
	Cu	%	-	0.413	0.384	-7.02
OREAS 53Pb	Au	ppb	31	623	633	1.61
	Cu	%	-	0.546	0.496	-9.16
OREAS 60P	Au	ppm	80	2.6	2.66	2.31
	Ag	ppm	-	4.9	4.7	-4.08
OREAS 62Pa	Au	ppm	78	9.64	9.22	-4.36
	Ag	ppm	-	18.4	17.6	-4.35
OREAS 22P	Au	ppb	737	<5	-	-
Quartz Blank	Au	ppb	93	-	-	-

Source: PEA (2021)

11.6.3 Blank Samples

The QP did not receive any 2005-2008 blank data. A summary of those data in previous reports (

Table 11.6) indicate that the 2005-2008 analytical data are sufficiently contamination free to support mineral resource estimation and mine planning, but the QP has not verified those data.

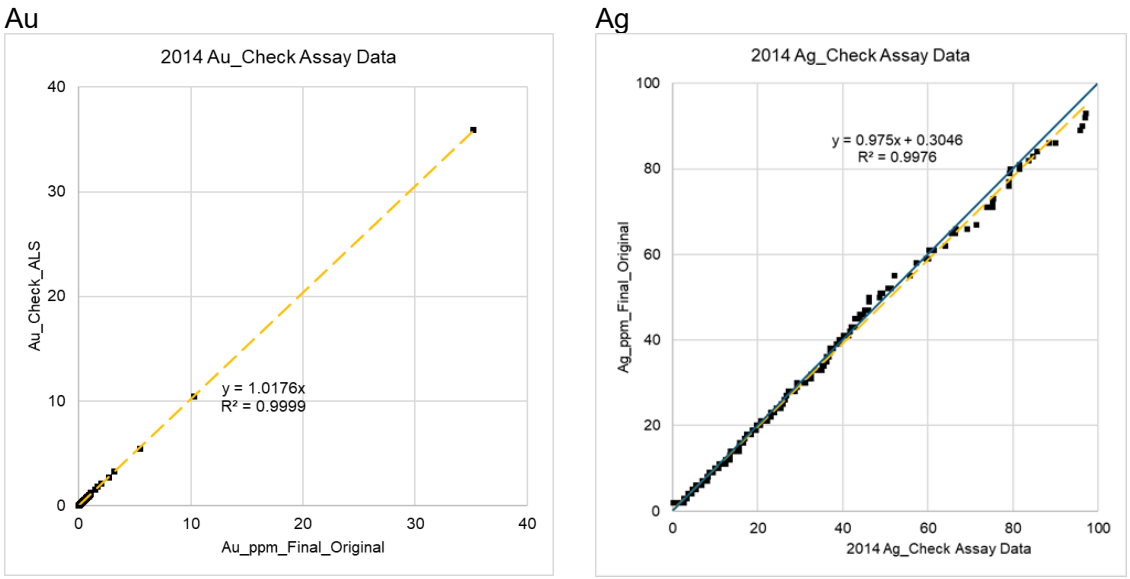
The QP received 14 blank data from ALS and 38 blank data from SGS generated in 2013-2014. For Au and Ag samples, only one of the results was greater than the detection limit and it was 2x LDL. For Cu, one result at ALS was >5 x LDL with a number of results in the LDL to 5x LDL range. Cu at SGS shows five results in the 11 to 19 ppm range which is considered by the QP to be acceptable. No evidence of any systematic contamination exists in these data.

11.6.4 Check Assays

In 2014, approximately 305 samples were submitted to ALS for check assays. The QP received no check assay data for other analytical programs or time periods.
The QP reviewed data for Au, Ag, and Cu. Figure 11.4 shows Q-Q plots for Au and Ag. Data for Au are positively biased at ALS by about 1.8% which is considered by the QP to be acceptable. Ag data are negatively biased at ALS by about 2.5% which is acceptable. For the 2013 – 2014 analytical program, these data suggest that the primary laboratory, SGS, produced acceptably accurate data.

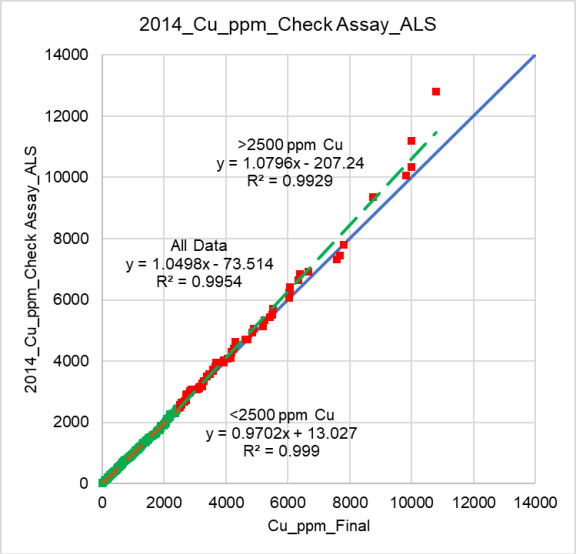
Cu is shown in Figure 11.5. Overall, ALS is positively biased by about 5% which is just within the ±5% limits generally accepted by the industry. However, data below 2,500 ppm are biased by only about 3% which is quite acceptable. Data above 2,500 ppm are biased by about 8% which is not generally acceptable. The source of this conditional bias is not known and should be considered to be a potential problem if Cu is to be estimated and incorporated into the mineral resource as a pay metal.

Figure 11.4: Au and Ag Check Assay Summary



Source: MTS (2025)

Figure 11.5: Cu Check Assay Summary



Source: MTS (2025)

11.7 Databases

The database is managed using Access software.

11.8 Sample Security

All RC and DDH samples collected were bagged, and then sealed prior to transport.

Samples were then securely stored until collected by SGS personnel. Samples collected during drilling were under constant surveillance until collected by SGS personnel. Samples collected from core were stored at a secure core yard facility under constant surveillance until collected by SGS personnel. When drilling was in progress, samples were collected and transported by SGS personnel to Durango approximately twice weekly.

SGS has a sample preparation facility in Durango, Mexico that is within a one-day drive of the property. Sample pulps were prepared in Durango prior to transporting via air-freight to the Toronto laboratory in Canada for analysis.

11.9 Sample Storage

Core, RC cuttings, bulk rejects, and pulps are stored near the site in a security-fenced core logging facility and within the project office area. The QP observed these items at the core logging facility during the site visit.

11.10 QP Comments on Section 11

The QP concludes:

- Sampling procedures are consistent with industry practices;
- Density determinations were performed with a variation of a commonly used method. The QP considers the density data to be reasonable;
- Adjustment of density because of selection bias, while warranted, is premature. These types of adjustments should be done based on mining experience;
- Sample preparation and QC was adequate;
- Analytical methods are widely used in the industry and appropriate for the type of mineralization. QC was adequate;
- Data are stored in Access software;
- Sample security was adequate;
- Samples and unused core are stored on site in a secure facility.

12 DATA VERIFICATION

12.1 Internal Data Verification

Heliostar staff surveyed the drill hole coordinates for eleven drill holes and various field outcrops at CdG in June and July 2025 to confirm drill hole coordinates in the database. Mineral tenure status was ascertained by obtaining a title legal opinion. Hardcopies of assay certificates, drill logs, survey certificates and downhole surveys were reviewed. Heliostar staff also undertook verification of geological logging by reviewing available diamond drill core and reverse circulation (RC) drill chips. Logged lithology, alteration, mineralization, and oxidation state were compared with the physical core and RC chips to confirm consistency between recorded geological descriptions and observed geological characteristics.

Assay results were visually and statistically compared against geological observations from core and RC chips to ensure that reported gold, silver, and copper grades were consistent with observed mineralization styles, sulfide content, and alteration intensity. No material discrepancies were identified during this review.

12.2 External Analytical Data Verification

12.2.1 2019 Verification

Defilippi, et al. (2020), discussed data verification completed by Kappes, Cassiday & Associates on the 2019 database. That work included:

- An audit of the data;
- Investigation and verification of:
    - Overlapping intervals;
    - Duplicate entries;
    - Missing data;
    - Assay values at nonsensical levels (e.g. beyond mineralogical possibility);
    - Calculated values at nonsensical levels (e.g. sample recovery beyond 100%);
    - Collar locations.

12.2.2 Pre-2013 Drill Hole Data

Golder (2006) verified the assay data integrity as used for the CdG resource estimate by completing a random comparison of approximately 5% of the database records against the original hard copy assay certificates. Only the Au (ppb), Au (g/t), Ag (g/t), and Cu

(ppm) data were compared. No discrepancies were found. A check was also done comparing the lithological logging to the core photos. No significant errors were found.

As part of the implementation of the acQuire data management system, all original assay results as received from SGS were used to re-establish the assay database and a cross-check against the original MS Access database highlighted no significant errors.

An extract from the acQuire drill hole database was supplied to Golder Associates who loaded and partially validated the data in MS Access. This partial validation primarily checked the structure of the database, such as gaps in the data and mismatches between collar, survey, assay and geology data. It also highlights outliers for each individual field. Only minor corrections to the drill hole database were required, which were completed prior to the data being used for resource estimation.

During the re-log program assay values were plotted on new log sheets and cross checked with the original drill logs. At this same time, drill collars and surveys were also checked against in the data base before the data was imported into GEMS.

12.2.3 2019 Data Verification

Argonaut commissioned Sanford Information Systems ("SIS") to complete a database audit during July of 2019 (Sanford Information Systems, 2019). This section summarizes that work.

12.2.3.1 Assays

Sanford Information Systems verified the database against signed assay certificates. Unit of measure errors were detected in Au in the database which appeared to be due to SGS changing reporting units from ppb to ppm. The database administrator did not account for the unit change and published Au ppm and Au ppb values in the same field. Additionally, a significant number of Au, Ag, and Cu values were missing from the database which were primarily determined to be lower detection limit values. Finally, Au ppm values had all been rounded to two decimals. Silver and copper assays were also subject to rounding issues.

Due to the quality issues with the results in the assay table and the need for accuracy, all original assay csv files were recompiled by element, unit of measure, and analytical method into the database. During the recompilation, the csv files were verified against the signed assay certificates. No issues with assay integrity were identified in the csv files.

QC samples provided by Rafael Puente for the 2013 campaign were imported into the assay table and cataloged by sample type and CRM. Analysis of the QC sample data confirmed that the guidelines are being followed.

Most RC assay intervals contained small gaps which originated by dropping decimals after feet were converted to meters.

12.2.3.2 Drill Hole Collar Locations

Drill holes were checked for mis-alignment using high-resolution air photo imagery and a high-resolution topographic surface. All hole locations matched very well with disturbance observed on the imagery. No holes were more than 3 meters below the surface topography. Three holes were 2-3 meters below the surface topography. One hole was 3-4 meters above the surface topography. Two holes were 2-3 meters above the surface topography. A resurvey of all holes more than 3 meters from the surface topography was recommended. Only one hole within the resource area required a resurvey.

12.2.3.3 Downhole Surveys

Original downhole survey records were checked against the database. A number of errors were detected and 50% of the dataset was recompiled in feet from the original datasheets. The primary issues were missing records and rounding errors in depth due to the conversion of feet to meters and dropping decimals. Only a few drill holes contained major deviations in azimuth or dip from the datasheet values which were corrected in the recompilation. From the new compilation, true north was recalculated using declination values published by NOAA. Annual variations and project location were used to determine declination values.

12.2.3.4 2019 QP Opinion

In the opinion of the QP responsible for this section of this technical report, the CdG drill hole data are accurate and suitable for mineral resource estimation.

12.3 Data Verification Performed by the 2025 Geology and Mineral Resource QPs

QPs for geology and mineral resource estimation visited site from 18-20 June 2025. During that site visit, the project site, site office, and core shed were visited.

12.3.1 Collar Location Verification

While on the project site, the QPs collected collar location data with a hand-held GPS unit (Garmin Tactix). Heliostar personnel had a TSC3 Data Controller in handset mode on site and collected additional location data. The QP found that some of the collars were

marked with concrete blocks (Figure 12.1) but that many were either not present, covered, or damaged. Locations for holes identified and located in the field are shown in Table 12.1. The differences are generally less than 8 m. Considering the precision of a hand-held GPS (±3 m x-y), the QP is of the opinion these data to confirm the locations of the holes in the field.

The QP was provided with original collar location data for 123 holes. Those data were compiled into a new database and compared to the locations in the project database. The QP discovered that data were originally surveyed in different coordinate systems than what is in the database. One of the systems was 34.4 m east, 199.1 m north, and 32.91 m lower than the WGS84 Zone 4 Q that is in use in the database. The other is 25.87 m east, 198.75 m north, and 1.03 m lower than the data in the database. The third system is 34.28 m east, 199.07 m north, and 1.03 m lower than the data in the database. The consistency of the differences indicates to the QP that the conversions were done properly and that the locations in the database can be relied upon for mineral resource estimation.

The locations in question were provided to Heliostar and verified as correct by their personnel.

Figure 12.1: Example of Concrete Block Marking Drill Holes (arrow points in direction of hole)



Source: MTS 2025

Table 12.1: Collar Locations Picked Up by QP (MTS) and Heliostar (Trimble)

HoleID	X	Y	Z	MTS X	MTS Y	MTS dx	MTS dy	Trimble x	Trimble y	Trimble Z	dx	dy	dz
DS-017	287085.88	2331492.76	2267.47	287086	2331494	-0.12	-1.24	287083.1	2331499.2	2284.0	2.8	-6.4	-16.5
DS-018	287083.78	2331491.98	2267.11	287085	2331490	-1.22	1.98	287082.1	2331498.8	2283.4	1.7	-6.8	-16.3
SA-017	288822.18	2332190.52	2199.70	288826	2332197	-3.82	-6.48						
SA-144	288567.35	2331755.23	2250.92					288567.4	2331755.0	2250.7	-0.1	0.2	0.3
SA-149	288791.72	2331847.71	2260.91	288795	2331851	-3.28	-3.29						
SA-152	288823.70	2332189.84	2199.73	288824	2332197	-0.30	-7.16						
SA-154	288821.15	2331995.63	2262.44	288821	2331997	0.15	-1.37						
SA-170	288672.11	2332239.94	2189.42	288675	2332244	-2.89	-4.06						
SA-173	288790.36	2332143.75	2214.63	288794	2332147	-3.64	-3.25						
SA-174	288879.72	2331896.56	2237.95	288883	2331902	-3.28	-5.44						
SA-175	288856.35	2331859.28	2240.57	288858	2331865	-1.65	-5.72						
SA18-MET-002	288824.14	2332187.50	2199.82	288829	2332191	-4.86	-3.50						
SA18-MET-003	288768.11	2331908.98	2281.46	288766	2331916	2.11	-7.02	288768.3	2331909.3	2282.1	-0.2	-0.3	-0.6
SA18-MET-004	288719.80	2332225.19	2191.36	288722	2332232	-2.20	-6.81	288721.1	2332226.3	2190.1	-1.3	-1.1	1.3
SA18-MET-006	288747.1	2331975.0	2293.2					288747.1	2331974.6	2292.2	0.0	0.5	1.0
SA18-MET-007	288673.31	2332238.87	2189.35	288678	2332244	-4.69	-5.13						
SA18-MET-011	288665.11	2331914.40	2300.60					288665.7	2331918.0	2305.5	-0.6	-3.6	-4.9
SA18-MET-014	288604.20	2331720.62	2249.45	288609	2331724	-4.80	-3.38						
SA18-MET-016	288677.41	2331746.82	2254.90	288682	2331749	-4.59	-2.18						
SA-229	288856.06	2331916.22	2245.89	288858	2331923	-1.94	-6.78						
SA-246	288846.11	2332255.52	2182.07	288849	2332261	-2.89	-5.48	288846.0	2332257.6	2180.9	0.1	-2.1	1.2
SA-257	288924.21	2332169.12	2206.54	288925	2332175	-0.79	-5.88						
SA-258	288806.64	2331833.61	2252.68	288807	2331839	-0.36	-5.39						
SA-246	288846.11	2332255.52	2182.07	288849	2332261	-2.89	-5.48	288846.0	2332257.6	2180.9	0.1	-2.1	1.2

12.3.2 Downhole Surveys

The QP compiled 1,160 original downhole surveys (31% of data in database) into a new database and compared those data to data in the database. A small number of discrepancies were discovered and corrected by Heliostar.

The QP checked the data for surveys that deviated more than is reasonable. Many in the industry use 2°/10 m as a deviation limit for H-size core. Smaller core (N and B) can bend somewhat more. Larger core (P) can bend less. RC rods typically deviate less than H-size core. A small number of extreme deviations (>5°/10 m) were discovered. Most of those were at the collar of the hole and were isolated. They will have little or no impact on the location of samples but are being checked and adjusted as time allows.

12.3.3 Core Logging

The QP visited the core shed and compared core to logs for a number of holes and intersections. The intent was primarily to verify that the lithologies were dominantly felsic intrusive rocks, volcaniclastic sedimentary rocks and fine-grained marine sedimentary rocks. These rock types were identified. Over the years, two lithology logging nomenclatures evolved because recent relogging of core used a different nomenclature than the legacy logging. Much of the core was not relogged. However, the detailed review of core by the QP allowed the QP to confidently make adjustments to the historical nomenclature to bring that work into line with more recent logging. That provided a basis for a reasonable geological model that was produced from the adjusted data with minor changed due to correlation problems within the geological model. However, these adjusted lithology codes are not as reliable as properly logged codes.

The QP recommended that all core be relogged using a standardized nomenclature for lithology, but is confident that the geological model based on the adjusted data is adequate to support PFS-level mineral resource estimation and preliminary mine planning. This model may not be adequate to support FS-level mineral resource estimation and mine planning.

Oxide state logging used four categories – oxide, oxide mix, sulfide mix, and sulfide. Oxide material is essentially entirely oxidized and sulfide material is entirely fresh with no discernable oxidation. The QP found that oxide state logging is somewhat subjective which is normally the case, but that it was reasonable. Those data were used to support an oxide state model that was then superimposed on the lithology model.

The QP performed a cursory review of mineralization and structure logs by comparing the database to the core. No concerns were discovered.

Geotechnical logs include recovery and rock quality designation (RQD). The QP recalculated the values from the raw data and found no errors. While in the core shed, the QP reviewed rock quality and recovery and found that the values in the database are reasonable. Note that it is not possible to precisely re-measure these values because the core has been split and sampled which tends to degrade the rock quality, but overall quality is discernable and well correlated with the values in the database. The QP considers the geotechnical measurements to be reasonable.

12.3.4 Sampling

The QP reviewed sample interval lengths in the database and found them to be reasonable. Beyond that, no verification is possible as no drilling was in progress at the time of the site visit and the QP could not observe sampling procedures while on site.

12.3.5 Sample Preparation

No sample preparation was in progress during the site visit so sample preparation was not observed by the QP. During compilation of original data for the database audit, the QP recorded sample preparation procedures and QA-QC data for sample preparation found in some original assay certificates. Those recorded procedures verified that sample preparation was as described in various company reports and that it was consistent with industry practices.

12.3.6 Analysis

No sample analysis was in progress during the site visit; however, the procedures used were indicated on the original assay certificates which verified the methods used (described in Section 11.5). Those methods are consistent with industry practices.

12.3.7 Density

The QP reviewed the density data and the procedures used to determine density. No density data were being collected at the time of the site visit so actual procedures were not observed. The density data are reasonable, but the QP did not have access to original data so calculations could not be verified. The methodology, as described in company documents, is consistent with industry practices.

12.3.8 QA-QC

The QP reviewed QA-QC data in the database and, to the extent possible, company reports. The QP found that the QA-QC data were collected properly. The data were analyzed using methods commonly used in the industry.

The QP investigated the possibility of downhole contamination in the RC holes. No evidence of any systematic downhole contamination was found. This does not preclude the possibility of isolated contamination.

12.4 Heap Leach Facility and Waste Rock Dump Geotechnical Information

For the geotechnical information for the HLF and WRD in Section 6, AECOM utilized historical data from Golder well as historical data from Knight Piesold and Primero, the previous operator. In the opinion of the QP responsible for the geotechnical information contained in Section 6, the data is sufficient to use for the design of the WRD and HLF contained in this report.

13 Mineral Processing and Metallurgical Testing

13.1 Mineral Processing Summary

The CdG Project will be designed as an open-pit mine with a heap leach operation utilizing a multiple-lift, single-use leach pad. Test work evaluated heap leach recoveries with crushed material.

Crushing will be accomplished using two stages of conventional crushing and one stage of high-pressure grinding roll (HPGR) crushing that will produce a 4 to 6 mm product (80% passing size or p80). The final product from the crushing circuit will be agglomerated with cement and conveyed to the heap leach pad where a conveyor/stacking system will place the material in discrete lifts.

The stacked ore will be leached using a dilute cyanide solution to recover gold, silver and copper. Leaching will be conducted in two stages, or phases, that vary in time, solution flow, and cyanide concentration. The gold, silver, and copper bearing solution will be collected in the pregnant solution pond and pumped to a Sulfidization, Acidification, Recycle, Thickening plant (SART). Pregnant solution will be acidified with sulfuric acid in the SART plant. Copper and silver will be precipitated as sulfides by the addition of sodium sulfide. The precipitate will be thickened and filtered to produce a wet filter cake for shipment. The copper thickener overflow solution will be neutralized with slaked lime. Gypsum will precipitate during the neutralization process and will be removed in a thickener. The gypsum thickener underflow will be pumped to a collection pond during the early stages of the project, ultimately the gypsum byproduct is expected to be stored long-term in an unused area of the heap leach pad. Gypsum thickener overflow will gravity-flow to the ADR plant for gold recovery.

The resulting barren solution will flow over a carbon safety screen, then to a barren solution storage tank. Make-up sodium cyanide will be added, and then the barren solution will be pumped back up to the heap.

Gold will be produced as metal doré from a conventional activated carbon adsorption process (ADR) and shipped to a precious metal refiner. The copper and silver will be processed via SART and resulting products sold and shipped to smelters.

Additional details on processing are presented in Section 17.

13.2 Metallurgical Test Work Summary

The laboratory test work has included characterization, comminution, flotation, and cyanidation through column, bottle roll and agitated (shake) leach tests. Data was further differentiated by crush size and type (conventional and HPGR). Historical test work included laboratory work prior to 2015 performed by others. The most recent test work includes results from 2018 and 2019 performed by KCA.

Historical test work included three main categories for the ore types (Weathered/Oxide, Mixed, Fresh/Sulfide) while the current test work further differentiates the Mixed category into two separate material types (Mixed Oxide, Mixed Sulfide). In total there are four separate material ore type categories that were considered in the recent test work and in this study: Oxide, Mixed Oxide, Mixed Sulfide, and Sulfide. The projected field recoveries and reagent consumptions for the four separate material ore types were derived from the current test work since the historic test work that matched the process criteria (column leached HPGR crushed material) were not able to be included in this analysis due to lack of information that connect the old samples to the mineralization breakdown selected for this project. However, the historical test results are generally in line with the most recent results.

The projected field gold and silver recoveries based on the available test work results are summarized in Table 13.1 and the estimated reagent consumptions are presented in Table 13.2. CdG is estimated to use a 120-day leach cycle on the heap leach with a HPGR crush size from 4 to 6 mm (80% passing size or p80).

Table 13.1 Cerro del Gallo Recoveries by Material Type

Ore Type (HPGR)	Feed Distribution LOM Average[1]	Column Test Recoveries		Projected Field Recoveries[2]		
		Au, %	Ag, %	Au, %	Ag, %	Cu%
Weathered (Oxide)	9.2%	76	63	74	60	17
Mixed Oxide	5.8%	70	76	68	73	62
Mixed Sulfide	38.0%	63	61	61	58	73
Fresh (Sulfide)	47.0%	55	37	53	35	59
Life of Mine (LOM) Average				59	49	62
[1] Based on HRC mine plan Oct 2025						
[2] Gold Discount, %	2					
[2] Silver Discount, %	3					
[2] Copper Discount, %	5					

Source: KCA (2025)

Table 13.2 Cerro del Gallo Reagent Consumptions by Material Type

Ore Type (HPGR)	Feed Distribution LOM Average[1]	Column Test Reagents, kg/t			Projected Field Reagents, kg/t		
		NaCN	Cement[2]	Lime	NaCN[3]	Cement	Lime
Weathered (Oxide)	9.2%	2.3	20.0	–	0.87	10.0	–
Mixed Oxide	5.8%	2.7	10.0	–	0.34	10.0	–
Mixed Sulfide	38.0%	3.3	10.0	–	0.87	10.0	–
Fresh (Sulfide)	47.0%	2.4	7.5	–	0.60	10.0	–
Life of Mine (LOM) Average					0.71	10	–
[1] Based on HRC mine plan Oct 2025							
[2] Based on compacted permeability test work, 80 m							
[3] Based on SART Column Tests and METSIM simulation							

Source: KCA (2025)

The material samples analyzed by KCA have demonstrated amenability to cyanide heap leaching. Elevated levels of copper in the ore material may have a significant effect on potential economic extraction if not managed properly.

13.3 Historical Testing

Historical test work included laboratory work prior to 2015 performed by others.

Test work prior to 2015 was summarized from *Technical Report First Stage Heap Leach Feasibility Study Cerro del Gallo Gold Silver Project* issued by Sedgman Limited, Reserva International, and Mine Development Associates in June 2012. The historical test work includes comminution, HPGR, flotation, gravity separation, agitated leach, intermittent bottle roll cyanide leach, percolation, column leach, Merrill Crowe, carbon adsorption, and SART. Historical test work included three main categories of ore types Weathered/Oxide, Mixed, and Fresh/Sulfide.

13.4 Metallurgical Testing Pre-2015

Test work commenced in 2006 and was conducted by various metallurgical laboratories, including: Independent Metallurgical Laboratories (IML) in Welshpool, Australia, SGS Lakefield Oretest Pty Ltd (Oretest) in Malaga, Australia, ALS AMMTEC in Balcatta, Australia, JKTech of Brisbane Queensland, Australia, and Polysius Australia Pty Ltd (Polysius) of Henderson, Australia.

The test work was performed on drill core samples and RC chip samples and included three main categories for the ore types (Weathered, Mixed, Fresh). Flotation test work was explored but not pursued. Subsequent test work was focused on cyanide leaching due to the potential economic viability.

13.4.1 Comminution Test Work

The results of the comminution test work indicate that some of the ore types are somewhat abrasive and may require moderately high energy for grinding. Results summarized in Table 13.3.

Table 13.3 Cerro del Gallo Comminution Test Work Results

Abrasion Index	Bond Work Index (kWh/tonne)			
(g)	Rod Mill	Ball Mill	Rwi:Bwi	Comp. Ball Mill
0.3974	24.9	17.5	1.42	12.9

Source: KCA (2019)

13.4.2 High Pressure Grinding Roll (HPGR) Test Work

The HPGR test work included single pass and locked cycle crushing, a Polysius ATWAL wear test, and Bond ball mill work index tests. Results are summarized in Table 13.4 through Table 13.7.

Table 13.4 Cerro del Gallo Bond Ball Mill Work Index Results Summary

Sample ID	F80 (microns)	P80 (microns)	Work Index (kWh/t)
Master Feed	2,374	75	14.9
Test 8 center product (-4mm)	2,153	85	14.6

Source: KCA (2019)

Table 13.5 Cerro del Gallo Single Pass HPGR Test Work

Test No.	Moisture	Roll Speed	Specific Throughput	Specific Grinding Force	Specific Energy Input Net Feed (dry)
	(%)	(m/s)	(ts/hm3 (dry))	(N/mm2)	(kWh/t)
1	3.0	0.2	214.8	3.94	1.90
2	3.0	0.2	221.8	3.08	1.46
3	3.0	0.5	214.7	4.02	1.96
4	3.0	0.2	207.1	3.84	2.14

Source: KCA (2019)

Table 13.6 Cerro del Gallo Closed Circuit HPGR Test Work Results

Test No.	Moisture	Specific Throughput	Specific Grinding Force	Specific Energy Input Net Feed (dry)
	(%)	(ts/hm3 (dry))	(N/mm2)	(kWh/t)
5	3.0	222.3	3.74	1.78
6	3.0	222.7	3.80	1.65
7	3.0	218.3	3.80	1.64
8	3.0	222.2	3.71	1.60

Source: KCA (2019)

Table 13.7 Cerro del Gallo HPGR Wear Rate Test Work Results

Parameter	Test 1	Test 2
Throughput (kg/h)	212.5	187.8
Specific throughput (ts/hm3)	164.7	145.6
Total wear rate (g/t)	41.0	60.5

Source: KCA (2019)

13.4.3 Flotation Test Work

The flotation test work was conducted on Weathered/Oxide and Mixed Oxide sample material. Results indicated low recoveries with gold, silver, and copper. The test results are presented in Table 13.8.

Table 13.8 Cerro del Gallo Average Cleaner Flotation Test Results

Con. Wt	Concentrate Grades			Recoveries (%)		
(%)	% Cu	Au g/t	As g/t	Cu	Au	As
0.70	4.90	55.8	4450	38.9	33.2	10.8

Source: KCA (2019)

13.4.4 Gravity Separation Test Work

The gravity separation test work indicated low performance in recovering gold, silver, and copper. Results are presented in Table 13.9.

Table 13.9 Cerro del Gallo Average Gravity Separation Results

Mass	Assays (g/t)			Distributions (%)		
(%)	Au	Ag	Cu	Au	Ag	Cu
0.34	28.2	436	3355	11.6	8.8	1.6

Source: KCA (2019)

13.4.5 Agitated Cyanide Leach Test Work

Agitated cyanide leach tests were conducted. The campaign results were overall positive and indicated moderately high gold, silver dissolutions with moderate copper dissolution. The average agitated cyanide leach results are shown in Table 13.10.

Table 13.10 Cerro del Gallo Average Agitated Cyanide Leaches

Camp.	Reagents (kg/t)			Assays (g/t)						Dissolutions (%)		
	NaCN Added	NaCN Cons.	Lime Added	Calc. Head (Au)	Res (Au)	Calc. Head (Ag)	Res. (Ag)	Calc. Head (Cu)	Res. (Cu)	Au	Ag	Cu
1	2.91	2.14	1.30	1.20	0.22	18	12	-	-	80	33.0	-
2	1.93	1.41	1.99	1.17	0.21	13	6	774	383	83	63	37.4
3	2.25	2.25	1.55	1.31	0.14	12	4	775	324	89	72	57.1

Source: KCA (2019)

13.4.6 Intermittent Bottle Roll Leach Test Work

The leach test work included intermittent bottle rolls, and generally observed increased recoveries of gold, silver, and copper with finer crush sizes. Results from the test campaign are presented in Table 13.11

Table 13.11 Cerro del Gallo Intermittent Bottle Roll Cyanide Leach Results

Camp.	Reagents (kg/t)			Assays (g/t)						Dissolutions (%)		
	NaCN Added	NaCN Cons.	Lime Added	Calc. Head (Au)	Res (Au)	Calc. Head (Ag)	Res. (Ag)	Calc. Head (Cu)	Res. (Cu)	Au	Ag	Cu
1-6.3mm	1.84	0.62	0.81	0.91	0.59	18	16	-	-	35.8	12.4	-
1-3.35mm	2.75	1.56	2.03	1.49	0.58	20	12	-	-	57.6	37.3	-
2-19mm	2.78	1.31	2.89	1.11	0.61	18	15	1038	723	43.1	29.3	29.4
2-6.3mm	3.16	1.57	3.28	1.15	0.53	19	12	985	655	52.1	40.0	33.9
3-8mm[C]	3.31	1.75	3.10	1.17	0.60	14	8	760	505	48.8	43.6	33.6
3-8mm[H]	2.72	1.19	2.78	1.31	0.54	13	6	638	313	59.5	54.7	49.7
4-8mm[C]	2.57	1.78	2.82	1.33	0.56	15	8	839	447	58.3	46.4	47.2
5-3.27mm[H]	1.25	0.96	2.34	1.69	0.63	11	6	596	358	62.8	47.6	39.9
6-3.27mm[H]	1.62	1.32	2.53	1.76	0.62	13	6	677	359	64.8	52.2	47.0

C = Conventional crushed
H = HPGR crushed
Source: KCA (2019)

13.4.7 Percolation Test Work

The percolation test work included percolation and slump tests with cement addition. To achieve an acceptable slump below 10% and percolation rate above 10,000 L/m^2/h, at a 6.3 mm crush size, 4 kg/t of cement addition was required.

13.4.8 Column Leach Test Work

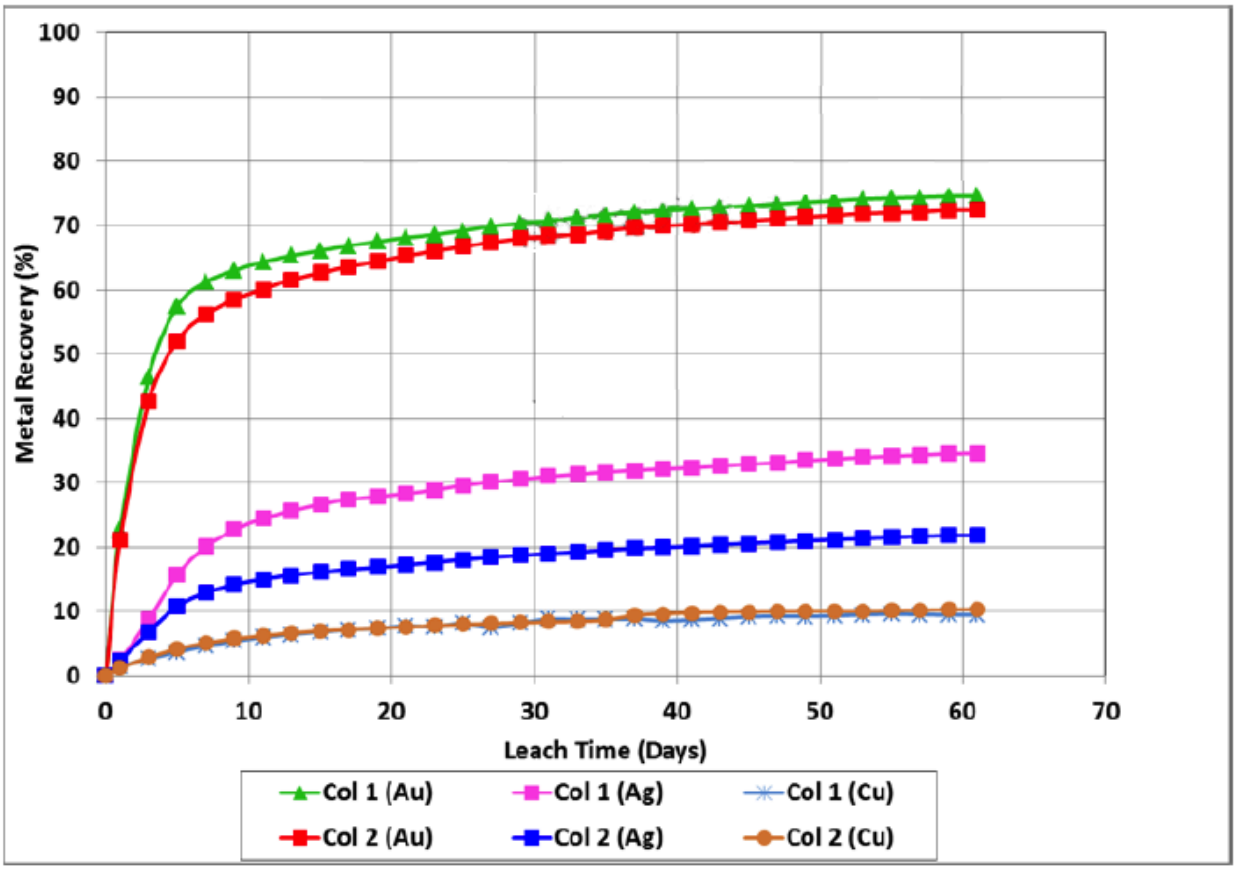
The initial column leach tests were conducted on Weathered/Oxide material at two separate crush sizes. In total the columns were leached for 61 days and increased recoveries were observed for the smaller crush size. Results are presented in Table 13.12 and illustrated in Figure 13.1.

Table 13.12 Cerro del Gallo Column Leach Results

Crush Size (mm)	Reagents (kg/t)			Assays (g/t)						Dissolutions (%)		
	NaCN Added	NaCN Cons.	Lime Added	Calc. Head (Au)	Res (Au)	Calc. Head (Ag)	Res. (Ag)	Calc. Head (Cu)	Res. (Cu)	Au	Ag	Cu
-6.3	0.83	0.20	0.07	0.70	0.15	7.8	4.5	204	178	78.4	42.1	12.6
-12.5	0.90	0.27	0.11	0.71	0.18	8.1	5.5	205	181	75.0	32.4	12.1

Source: KCA (2019)

Figure 13.1: Cerro del Gallo Column Leach Graphs



Source: KCA (2019)

Subsequent column leach tests were conducted on Weathered/Oxide and Mixed Oxide composite samples both conventionally and HPGR crushed. The columns leached for a total of 115 days each. Results are presented in Table 13.13 and show the HPGR crushed material with higher average metal dissolutions. Sample S2, conventionally crushed, was agglomerated with 6 kg/t of cement and resulted with higher cyanide consumption along with much higher copper recovery.

Table 13.13 Cerro del Gallo Column Leach – Conventional versus HPGR Crushing

Crush Size (mm)	Reagents (kg/t)			Assays (g/t)						Dissolutions (%)		
	NaCN Added	NaCN Cons.	Lime Added	Calc. Head (Au)	Res (Au)	Calc. Head (Ag)	Res. (Ag)	Calc. Head (Cu)	Res. (Cu)	Au	Ag	Cu
Conv.	1.36	1.08	1.28	1.20	0.58	13	8	786	553	52.2	38.0	31.5
HPGR	1.38	1.15	1.02	1.21	0.42	12	6	618	376	65.6	52.1	38.5
Conv. S2	4.39	4.39	1.03	1.24	0.53	12	6	1355	348	57.6	49.9	73.4

Source: KCA (2019)

Considering the column leach results with different ore types, no trend was observed between strongly, moderately, and weakly oxidized ore material types. Results are presented in Table 13.14.

Table 13.14 Cerro del Gallo Column Leach – Ore Types

Ore Type	Dissolutions (%) Conventional Crush			Dissolutions (%) HPGR Crush			Dissolutions (%) Conventional Crush S2		
	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu
Weathered	59.7	36.5	20.4	73.3	54.5	26.9	65.7	45.2	76.2
Strongly Oxidized	48.2	48.2	42.6	63.7	54.9	45.6	49.6	53.6	63.9
Moderately Oxidized	51.5	36.9	37.6	65.3	48.1	43.0	59.2	49.6	75.9
Weakly Oxidized	43.2	35.7	24.0	60.7	59.0	31.2	52.4	51.8	72.8

Source: KCA (2019)

13.4.9 Merrill Crowe Test Work

The Merrill Crowe tests were conducted on filtered pregnant solution and resulted in gold extractions of 85.8%, silver extractions 99.9%, and copper extractions of 13.1%.

13.4.10 Carbon Adsorption Test Work

The carbon adsorption test work was carried out to determine the Fleming and Nicol k and n values for gold, silver, and copper. Kinetics of adsorption were observed as moderate for gold, moderate to slow for silver, and slow to moderate for copper. On Series 2 there is a suspected assaying error with the copper carbon loading, maximum loading is approximated at 50,000 g/t. Results are summarized in Table 13.15.

Table 13.15 Cerro del Gallo Carbon Contact Results Summary

Series	Gold			Silver			Copper		
	k	n	Carbon Loading (g/t)	k	n	Carbon Loading (g/t)	k	n	Carbon Loading (g/t)
1	133.5	0.689	705	62.7	0.661	3,739	1.4	0.702	2,774
2	156.8	0.705	1,166	134.5	0.514	5,302	62.9	0.541	138,874

Source: KCA (2019)

13.4.11 SART Test Work

The SART test work was conducted on filtered pregnant leach solution. Results of this study produced a high-grade precipitate that may be further processed or sold, but also contained significant amounts of gold, zinc, mercury and iron that may pose issues with marketing. The results are presented in Table 13.16.

Table 13.16 Cerro del Gallo SART Results

Sample	Assays								
	CN Total (mg/L)	WADCN Calc. (mg/L)	SCN (mg/L)	CN Free (mg/L)	Au (mg/L)	Ag (mg/L)	Cu (mg/L)	Zn (mg/L)	Fe (mg/L)
Preliminary Tests									
Feed Solution	550	547	180	240	1.00	4.48	285	30.8	1.0
Barren Solution	500	499	190	159	0.62	0.00	0.1	5.0	0.5
Precipitate	-	-	-	-	-	0.29%	51.6%	4.0%	0.18%
Optimized Tests									
Feed Solution	560	560	210	150	1.12	5.56	317	21.9	<1
Barren Solution	440	440	225	285	0.62	0.02	11.6	7.0	<1
Precipitate	-	-	-	-	25.3 g/t	1.33%	58.3%	2.9%	0%

Source: KCA (2019)

The solution discharged from the SART test work was further processed with activated carbon resulting in 96.8% gold adsorption. Results are shown in Table 13.17.

Table 13.17 Cerro del Gallo SART Solution Carbon Adsorption

Product	Solution Assays (ppm)			Carbon Assay	Gold Adsorption
	Au	Ag	Cu	Au (g/t	(%)
Feed solution	0.62	0.002	0.10	0	0
Barren solution	0.02	0.002	0.25	59	96.8

Source: KCA (2019)

13.5 Metallurgical Testing 2018 and 2019

KCA's laboratory conducted test work on four main categories of ore types Weathered/Oxide, Mixed Oxide, Mixed Sulfide, and Fresh/Sulfide. These categories are based on observation of core content of sulfide material where Weathered/Oxide is 0%-15% sulfides, Mixed Oxide is 15%-55% sulfides, Mixed Sulfide is 55%-85% sulfide and Fresh/Sulfide is 85%-100% sulfides. The metallurgical test work included head analyses (assay and multi-element), agglomeration, percolation, compacted permeability, bottle roll and column leach on conventionally and HPGR crushed material. Comminution test work was performed by an external laboratory, Hazen Research Incorporated.

The head analyses test work included head fire assays for gold and silver, head screen analysis with assays by size fraction, assays by quantitative methods for carbon, sulfur and mercury and semi-quantitative assays by means of an ICAP-OES for an additional series of elements and for whole rock constituents. Results of the head analyses are presented in Table 13.18 through Table 13.25.

Table 13.18 Cerro del Gallo KCA Laboratory Gold and Silver Head Assays

KCA Sample No.	Description	Mineral Type	Average Assay, gms Au/MT	Average Assay, gms Ag/MT
82512 B	CO1	OX	0.270	14.09
82513 B	CO2	MIX OX	1.143	12.51
82514 A	CO3	MIX OX	0.732	20.35
82515 A	CO4	MIX SULF 1	0.739	14.50
82516 A	CO5	MIX SULF 1	0.375	29.71
82517 A	CO6	MIX SULF 2	0.495	22.41
82518 B	CO7	MIX SULF 2	0.835	6.51
82519 B	CO8	SULF HG	1.694	2.40
82520 A	CO9	SULF	1.903	5.79
82521 A	CO10	OX HG	1.365	9.70
82522 A	CO11	MIX SULF HG	3.064	3.39

Source: KCA (2019)

Table 13.19 Cerro del Gallo KCA Laboratory Copper, Lead and Zinc

KCA Sample No.	Description	Mineral Type	Copper, mg/kg	Lead, mg/kg	Zinc, mg/kg
82512 B	CO1	OX	480	263	672
82513 B	CO2	MIX OX	600	153	782
82514 A	CO3	MIX OX	461	340	260
82515 A	CO4	MIX SULF 1	691	188	911
82516 A	CO5	MIX SULF 1	2310	713	803
82517 A	CO6	MIX SULF 2	992	404	1300
82518 B	CO7	MIX SULF 2	1200	347	844
82519 B	CO8	SULF HG	711	17	419
82520 A	CO9	SULF	731	123	560
82521 A	CO10	OX HG	686	55	296
82522 A	CO11	MIX SULF HG	671	11	229

Source: KCA (2019)

Table 13.20 Cerro del Gallo KCA Laboratory Carbon and Sulfur

KCA Sample No.	Description	Mineral Type	Total Carbon, %	Organic Carbon, %	Inorganic Carbon, %	Total Sulfur, %	Sulfide Sulfur, %	Sulfate Sulfur, %
82512 B	CO1	OX	<0.01	<0.01	<0.01	0.05	<0.01	0.04
82513 B	CO2	MIX OX	0.02	0.02	<0.01	0.14	0.05	0.09
82514 A	CO3	MIX OX	0.05	0.04	0.02	0.30	0.20	0.10
82515 A	CO4	MIX SULF 1	0.02	0.02	<0.01	1.13	0.94	0.20
82516 A	CO5	MIX SULF 1	0.06	0.04	0.02	1.80	1.60	0.20
82517 A	CO6	MIX SULF 2	0.04	0.02	0.02	2.06	1.76	0.30
82518 B	CO7	MIX SULF 2	0.04	0.02	0.02	1.29	1.13	0.16
82519 B	CO8	SULF HG	0.16	0.07	0.09	1.47	1.23	0.25
82520 A	CO9	SULF	0.04	0.03	0.01	0.95	0.84	0.11
82521 A	CO10	OX HG	0.05	0.04	0.01	0.12	0.04	0.08
82522 A	CO11	MIX SULF HG	0.02	<0.01	0.02	0.53	0.43	0.09

Source: KCA (2019)

Table 13.21 Cerro del Gallo KCA Laboratory Mercury and Copper

KCA Sample No.	Description	Mineral Type	Total Mercury, mg/kg	Total Copper, mg/kg	Cyanide Soluble Copper, mg/kg	Cyanide Soluble Copper, %
82512 B	CO1	OX	<0.02	480	93.70	20%
82513 B	CO2	MIX OX	<0.02	600	81.60	14%
82514 A	CO3	MIX OX	0.10	461	78.45	17%
82515 A	CO4	MIX SULF 1	<0.02	691	78.40	11%
82516 A	CO5	MIX SULF 1	<0.02	2310	167.40	7%
82517 A	CO6	MIX SULF 2	<0.02	992	69.00	7%
82518 B	CO7	MIX SULF 2	<0.02	1200	73.05	6%
82519 B	CO8	SULF HG	<0.02	711	58.30	8%
82520 A	CO9	SULF	<0.02	731	60.35	8%
82521 A	CO10	OX HG	<0.02	686	68.65	10%
82522 A	CO11	MIX SULF HG	0.02	671	10.28	2%

Source: KCA (2019)

Table 13.22 Cerro del Gallo KCA Laboratory Multi-Element Analysis (1 of 2)

Constituent	Unit	KCA Sample No. 82512 B CO1	KCA Sample No. 82513 B CO2	KCA Sample No. 82514 A CO3	KCA Sample No. 82515 A CO4	KCA Sample No. 82516 A CO5	KCA Sample No. 82517 A CO6
Al	%	5.98	4.08	3.99	5.20	3.85	4.79
As	mg/kg	343	212	312	889	257	209
Ba	mg/kg	919	464	396	438	346	431
Bi	mg/kg	93	374	78	384	125	154
$C_{(total)}$	%	<0.01	0.02	0.05	0.02	0.06	0.04
$C_{(organic)}$	%	<0.01	0.02	0.04	0.02	0.04	0.02
$C_{(inorganic)}$	%	<0.01	<0.01	0.02	<0.01	0.02	0.02
Ca	%	0.03	0.25	0.03	0.38	0.04	0.17
Cd	mg/kg	9	12	8	32	22	44
Co	mg/kg	3	8	2	10	8	11
Cr	mg/kg	25	70	87	91	75	92
$Cu_{(total)}$	mg/kg	480	600	461	691	2310	992
$Cu_{(cyanide\ soluble)}$ [1]	mg/kg	93.7	81.6	78.5	78.4	167.4	69.0
Fe	%	3.32	3.25	2.6	4.18	4.11	5.16
Hg	mg/kg	<0.02	<0.02	0.10	<0.02	<0.02	<0.02
K	%	3.65	2.46	3.37	3.41	2.37	2.3
Mg	%	0.18	0.60	0.11	0.82	0.13	0.76
Mn	mg/kg	868	592	73	537	164	784
Mo	mg/kg	23	14	25	11	47	36
Na	%	0.19	0.37	0.16	0.55	0.18	0.58
Ni	mg/kg	4	17	15	41	11	27
Pb	mg/kg	263	153	340	188	713	404
$S_{(total)}$	%	0.05	0.14	0.3	1.13	1.8	2.06
$S_{(sulfide)}$	%	<0.01	0.05	0.2	0.94	1.6	1.76
$S_{(sulfate)}$	%	0.04	0.09	0.1	0.2	0.2	0.3
Sb	mg/kg	15	17	21	17	11	7
Se	mg/kg	9	14	17	50	16	17
Sr	mg/kg	133	57	61	99	53	92
Te	mg/kg	8	14	9	12	12	15
Ti	%	0.07	0.16	0.05	0.21	0.11	0.17
V	mg/kg	31	62	33	80	46	88
W	mg/kg	56	75	41	56	35	45
Zn	mg/kg	672	782	260	911	803	1300

Source: KCA (2019)

Table 13.23 Cerro del Gallo KCA Laboratory Multi-Element Analysis (2 of 2)

Constituent	Unit	KCA Sample No. 82518 B CO7	KCA Sample No. 82519 B CO8	KCA Sample No. 82520 A CO9	KCA Sample No. 82521 A C10	KCA Sample No. 82522 A C11
Al	%	3.52	5.33	4.92	3.76	4.27
As	mg/kg	249	124	508	155	140
Ba	mg/kg	323	495	526	446	387
Bi	mg/kg	139	298	216	208	170
$C_{(total)}$	%	0.04	0.16	0.04	0.05	0.02
$C_{(organic)}$	%	0.02	0.07	0.03	0.04	<0.01
$C_{(inorganic)}$	%	0.02	0.09	0.01	0.01	0.02
Ca	%	0.09	1.76	0.15	0.13	0.15
Cd	mg/kg	95	8	16	6	8
Co	mg/kg	9	14	7	4	6
Cr	mg/kg	100	88	69	94	75
$Cu_{(total)}$	mg/kg	1200	711	731	686	671
$Cu_{(cyanide\ soluble)}$[1]	mg/kg	73.05	58.3	60.35	68.65	10.28
Fe	%	3.44	4.2	2.29	3.38	1.79
Hg	mg/kg	<0.02	<0.02	<0.02	<0.02	0.02
K	%	2.55	2.87	3.71	1.69	3.6
Mg	%	0.35	0.78	0.40	0.18	0.26
Mn	mg/kg	342	822	388	192	171
Mo	mg/kg	46	19	25	35	24
Na	%	0.35	1.46	0.79	0.46	0.49
Ni	mg/kg	12	23	7	10	7
Pb	mg/kg	347	17	123	55	11
$S_{(total)}$	%	1.29	1.47	0.95	0.12	0.53
$S_{(sulfide)}$	%	1.13	1.23	0.84	0.04	0.43
$S_{(sulfate)}$	%	0.16	0.25	0.11	0.08	0.09
Sb	mg/kg	7	10	4	9	6
Se	mg/kg	6	12	<5	12	<5
Sr	mg/kg	67	151	105	74	89
Te	mg/kg	10	20	12	15	11
Ti	%	0.13	0.27	0.09	0.13	0.08
V	mg/kg	41	67	26	55	23
W	mg/kg	42	108	38	37	39
Zn	mg/kg	844	419	560	296	229

Source: KCA (2019)

Table 13.24 Cerro del Gallo KCA Laboratory Whole Rock Analyses (1 of 2)

Constituent	Unit	KCA Sample No. 82512 B CO1		KCA Sample No. 82513 B CO2		KCA Sample No. 82514 A CO3		KCA Sample No. 82515 A CO4		KCA Sample No. 82516 A CO5		KCA Sample No. 82517 A CO6	
SiO_2	%	74.5		77.7		80.6		70.4		78.4		73.4	
Si	%		34.83		36.33		37.68		32.91		36.65		34.32
Al_2O_3	%	11.4		8.56		8.64		11.4		8.24		9.86	
Al	%		6.03		4.53		4.57		6.03		4.36		5.22
Fe_2O_3	%	4.56		4.92		3.91		6.62		5.71		7.02	
Fe	%		3.19		3.44		2.73		4.63		3.99		4.91
CaO	%	0.05		0.74		<0.01		0.96		<0.01		0.23	
Ca	%		0.04		0.53		0.00		0.69		0.00		0.16
MgO	%	0.33		1.13		0.18		1.56		0.23		1.39	
Mg	%		0.20		0.68		0.11		0.94		0.14		0.84
Na_2O	%	0.22		0.51		0.14		0.73		0.20		0.75	
Na	%		0.16		0.38		0.10		0.54		0.15		0.56
K_2O	%	4.34		3.05		4.32		4.12		3.13		2.80	
K	%		3.60		2.53		3.59		3.42		2.60		2.32
TiO_2	%	0.36		0.43		0.22		0.52		0.3		0.53	
Ti	%		0.22		0.26		0.13		0.31		0.18		0.32
MnO	%	0.13		0.09		0.01		0.08		0.02		0.11	
Mn	%		0.10		0.07		0.01		0.06		0.02		0.09
SrO	%	0.01		<0.01		<0.01		<0.01		<0.01		<0.01	
Sr	%		0.01		0.00		0.00		0.00		0.00		0.00
BaO	%	0.11		0.06		0.05		0.05		0.04		0.05	
Ba	%		0.10		0.05		0.04		0.04		0.04		0.04
Cr_2O_3	%	<0.01		0.02		0.01		0.02		0.01		0.01	
Cr	%		0.00		0.01		0.01		0.01		0.01		0.01
P_2O_5	%	0.10		0.11		0.08		0.11		0.07		0.09	
P	%		0.04		0.05		0.03		0.05		0.03		0.04
$LOI_{1090°C}$	%	3.39		2.70		1.79		3.34		3.41		3.83	
SUM	%	99.5		100.0		100.0		99.9		99.8		100.1	

Note: The SUM is the total of the oxide constituents and the loss on ignition.

Source: KCA (2019)

Table 13.25 Cerro del Gallo KCA Laboratory Whole Rock Analyses (2 of 2)

Constituent	Unit	KCA Sample No. 82518 B		KCA Sample No. 82519 B		KCA Sample No. 82520 A		KCA Sample No. 82521 A		KCA Sample No. 82522 A	
SiO_2	%	80.6		70.7		77.5		81.7		81.2	
Si	%		37.68		33.05		36.23		38.20		37.96
Al_2O_3	%	7.36		11.2		10.3		7.2		8.25	
Al	%		3.90		5.93		5.45		3.81		4.37
Fe_2O_3	%	4.61		5.71		3.32		4.62		2.50	
Fe	%		3.22		3.99		2.32		3.23		1.75
CaO	%	0.09		2.43		0.19		0.12		0.20	
Ca	%		0.06		1.74		0.14		0.09		0.14
MgO	%	0.62		1.53		0.75		0.31		0.46	
Mg	%		0.37		0.92		0.45		0.19		0.28
Na_2O	%	0.45		1.68		1.15		0.68		0.67	
Na	%		0.33		1.25		0.85		0.50		0.50
K_2O	%	3.29		3.66		4.66		2.12		4.83	
K	%		2.73		3.04		3.87		1.76		4.01
TiO_2	%	0.31		0.53		0.33		0.32		0.27	
Ti	%		0.19		0.32		0.20		0.19		0.16
MnO	%	0.05		0.12		0.05		0.03		0.02	
Mn	%		0.04		0.09		0.04		0.02		0.02
SrO	%	<0.01		0.01		<0.01		<0.01		<0.01	
Sr	%		0.00		0.01		0.00		0.00		<0.01
BaO	%	0.04		0.06		0.06		0.05		0.05	
Ba	%		0.04		0.05		0.05		0.04		0.04
Cr_2O_3	%	0.02		0.01		<0.01		0.01		0.01	
Cr	%		0.01		0.01		<0.01		0.01		0.01
P_2O_5	%	0.05		0.15		0.08		0.07		0.08	
P	%		0.02		0.07		0.03		0.03		0.03
$LOI_{1090°C}$	%	2.44		2.45		1.98		2.97		1.66	
SUM	%	99.9		100.2		100.4		100.2		100.2	

Note: The SUM is the total of the oxide constituents and the loss on ignition.

Source: KCA (2019)

Comminution testing consisted of Bond ball mill work index (BW_i) and Bond abrasion index (A_i) testing. Sample material was stage crushed to minus 6 mesh and utilized as the feed for the grindability tests. The results are presented in Table 13.26.

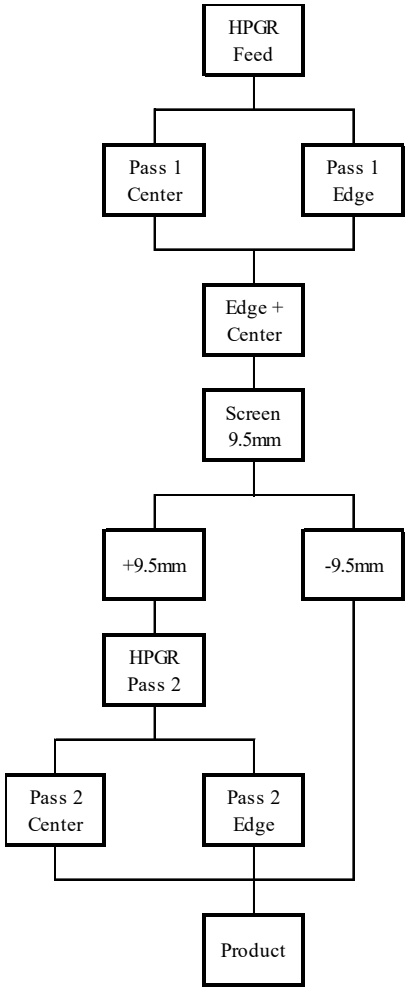
Table 13.26 Cerro del Gallo KCA Comminution Test Results

KCA Sample No.	Description	Mineral Type	A_i	BW_i, kWh/MT
82512 A	CO1	OX	0.1777	9.3
82513 A	CO2	MIX OX	0.5402	14.8
82515 A	CO4	MIX SULF 1	0.3498	12.7
82516 A	CO5	MIX SULF 1	0.4933	12.1
82520 A	CO9	SULF	0.8189	14.2

Source: KCA (2019)

KCA also conducted HPGR test work. Material was crushed to 100% passing 19 mm with a Thyssenkrupp Pilotwal HPGR unit with no edge recycle, screened at 9.5 mm and oversize material re-crushed, as depicted in Figure 13.2. The crushing was done at one pressure and one moisture content. Results are presented in Table 13.27.

Figure 13.2: Cerro del Gallo KCA Laboratory HPGR Outline



Source: KCA (2019)

Table 13.27 Cerro del Gallo KCA HPGR Crushing Test Results

Mineral Type			OX	MIX OX	MIX S	SULF HG	SULF	OX HG	MIX SULF HG
POLYCOM size			Pilotwal	Pilotwal	Pilotwal	Pilotwal	Pilotwal	Pilotwal	Pilotwal
Roll surface			studs	studs	studs	studs	studs	studs	studs
Feed material:			82512 A	82513-14	82515-18	82519 A	82520 A	82521 A	82522 A
Feed size:		mm	0..20	0..20	0..20	0..20	0..20	0..20	0..20
Moisture, Feed		[% H2O]	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Moisture, Product		[% H2O]	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Roll diameter	D	[m]	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Roll length	L	[m]	0.30	0.30	0.30	0.30	0.30	0.30	0.30
Circumf. roll speed	u	[m/s]	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Zero gap	s	[mm]	1.5	1.6	1.6	1.5	1.6	1.6	1.6
Initial hydraulic pressure	p0	[bar]	105	104	103	103	102	103	103
Preset nitrogen pressure	pN2	[bar]	95	95	95	95	95	95	95
Initial grinding force	F0	[kN]	397.3	394.7	393.0	392.9	389.6	391.8	392.5
Feed material, moist	M	[kg]	137.5	141.0	148.3	149.5	140.5	143.5	146.0
Feed material, dry	M	[kg]	136	140	147	148	139	142	145
Test time	t	[min]	1.22	1.10	1.22	1.17	1.07	1.19	1.16
Throughput, moist	M	[t/h]	6.77	7.71	7.30	7.70	7.87	7.27	7.53
Throughput, dry	M	[t/h]	6.70	7.63	7.23	7.62	7.79	7.19	7.45
Specific. throughput (feed), moist	m	[ts/hm³]	230.9	263.0	248.8	262.1	266.9	247.9	256.8
Specific. throughput (feed), dry	m	[ts/hm³]	228.6	260.3	246.3	259.5	264.2	245.5	254.2
Working Gap	sa	[mm]	8.1	11.5	10.9	12.5	12.8	11.5	11.9
Hydraulic pressure	pB	[bar]	120.7	130.6	128.2	133.9	133.7	130.0	131.6
Grinding force	F	[kN]	459.0	496.6	487.5	508.9	508.1	494.1	500.3
Spec. grinding force	phi	[N/mm²]	3.06	3.31	3.25	3.39	3.39	3.29	3.34
Power @ counter - no load	P	[kW]	0.93	0.96	0.96	0.98	0.96	0.98	1.00
Power @ counter - gross	P	[kW]	14.20	14.50	13.98	15.74	15.11	12.91	14.27
Power @ counter, net	Pw	[kW]	13.27	13.54	13.02	14.77	14.15	11.93	13.27
Power @ shaft, net	Pw	[kW]	12.14	12.39	11.91	13.51	12.95	10.92	12.14
Specific energy input @ shaft, gross (moist feed)	w_{sp}	[kWh/t]	1.96	1.77	1.79	1.92	1.80	1.64	1.76
Specific energy input @ shaft, gross (dry feed)	w_{sp}	[kWh/t]	1.98	1.78	1.80	1.94	1.82	1.66	1.78
Specific energy input @ shaft, net (moist feed)	w_{sp}	[kWh/t]	1.79	1.62	1.63	1.75	1.65	1.50	1.61
Specific energy input @ shaft, net (dry feed)	w_{sp}	[kWh/t]	1.81	1.63	1.65	1.77	1.66	1.52	1.63
Distribution of material									
center		[%]	80.4	76.9	76.1	75.9	75.8	76.3	75.0
edge		[%]	19.6	23.1	23.9	24.1	24.2	23.7	25.0
Screen products									
Oversize (+9.5 mm)		[%]	8.1	21.0	12.5	12.8	15.4	11.2	12.8
Undersize (-9.5 mm)		[%]	91.9	79.0	87.5	87.2	84.6	88.8	87.2

Source: KCA (2019)

Cyanide bottle roll leach tests were run for a period of ninety-six (96) hours with solution sampling performed at 2, 4, 8, 24, 48, 72 and 96 hours. The tests utilized both pulverized (target 100% passing 0.150 millimeters) and milled material. NaCN was added and maintained at 1.0 grams per liter of solution. The pH of the solution was maintained at 10.5 to 11.0 with the addition of hydrated lime. Results of the cyanide bottle roll leach test work are summarized in Table 13.28 and Table 13.29.

Table 13.28 Cerro del Gallo KCA Laboratory Bottle Roll Leach Tests – Gold and Silver

KCA Sample No.	Description	Mineral Type	Type	Calculated p80, mm	Calculated Head, gms Au/MT	Au Extracted, %	Calculated Head, gms Ag/MT	Ag Extracted, %	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
82512 B	CO1	OX	Pulv.	--	0.280	88%	14.35	58%	0.40	2.75
82512 B	CO1	OX	Milled	0.074	0.300	88%	14.62	53%	1.48	2.00
82512 B	CO1	OX	Milled	0.053	0.219	91%	14.57	51%	0.77	2.25
	Average :				0.266	89%	14.51	54%	0.88	2.33
82513 B	CO2	MIX OX	Pulv.	--	1.187	94%	12.82	79%	1.29	2.50
82513 B	CO2	MIX OX	Milled	0.093	1.216	92%	13.26	74%	1.51	1.75
82513 B	CO2	MIX OX	Milled	0.054	1.159	93%	12.93	77%	2.00	2.00
	Average :				1.187	93%	13.00	77%	1.60	2.08
82514 A	CO3	MIX OX	Pulv.	--	0.810	93%	20.74	86%	1.46	1.50
82514 A	CO3	MIX OX	Milled	0.095	0.795	89%	20.61	84%	1.83	1.00
82514 A	CO3	MIX OX	Milled	0.054	0.769	90%	20.94	85%	2.09	1.25
	Average :				0.791	91%	20.76	85%	1.79	1.25
82515 A	CO4	MIX SULF 1	Pulv.	--	0.762	89%	14.73	82%	2.05	2.50
82515 A	CO4	MIX SULF 1	Milled	0.107	0.685	82%	14.33	73%	1.82	2.00
82515 A	CO4	MIX SULF 1	Milled	0.079	0.745	86%	14.46	76%	1.91	2.25
	Average :				0.731	86%	14.51	77%	1.93	2.25
82516 A	CO5	MIX SULF 1	Pulv.	--	0.401	81%	31.63	53%	5.58	2.00
82516 A	CO5	MIX SULF 1	Milled	0.101	0.392	76%	34.11	45%	5.84	1.25
82516 A	CO5	MIX SULF 1	Milled	0.071	0.424	81%	30.86	37%	6.40	2.00
	Average :				0.406	79%	32.20	45%	5.94	1.75
82517 A	CO6	MIX SULF 2	Pulv.	--	0.607	88%	23.03	67%	2.45	2.25
82517 A	CO6	MIX SULF 2	Milled	0.106	0.467	79%	22.51	52%	2.06	1.50
82517 A	CO6	MIX SULF 2	Milled	0.086	0.466	84%	22.70	53%	2.48	1.75
	Average :				0.513	84%	22.75	57%	2.33	1.83
82518 B	CO7	MIX SULF 2	Pulv.	--	0.826	87%	6.83	73%	2.90	1.75
82518 B	CO7	MIX SULF 2	Milled	0.100	0.784	82%	6.74	58%	3.20	1.25
82518 B	CO7	MIX SULF 2	Milled	0.063	0.821	85%	6.56	63%	3.22	1.50
	Average :				0.810	85%	6.71	65%	3.11	1.50
82519 B	CO8	SULF HG	Pulv.	--	1.811	90%	2.42	67%	1.41	1.25
82519 B	CO8	SULF HG	Milled	0.203	1.828	77%	2.52	48%	0.88	1.00
82519 B	CO8	SULF HG	Milled	0.132	1.792	83%	2.42	50%	0.74	1.00
	Average :				1.810	83%	2.45	55%	1.01	1.08
82520 A	CO9	SULF	Pulv.	--	1.936	93%	6.05	77%	1.67	1.25
82520 A	CO9	SULF	Milled	0.103	1.876	89%	6.17	61%	1.87	1.00
82520 A	CO9	SULF	Milled	0.070	1.833	91%	5.92	65%	2.18	1.00
	Average :				1.882	91%	6.05	68%	1.91	1.08
82521 A	CO10	OX HG	Pulv.	--	1.471	95%	10.57	89%	1.18	3.25
82521 A	CO10	OX HG	Milled	0.098	1.461	91%	10.11	86%	1.91	2.50
82521 A	CO10	OX HG	Milled	0.065	1.410	93%	10.05	88%	1.96	2.75
	Average :				1.447	93%	10.24	88%	1.68	2.83
82522 A	CO11	MIX SULF HG	Pulv.	--	3.215	96%	3.73	83%	1.74	1.50
82522 A	CO11	MIX SULF HG	Milled	0.116	3.243	93%	3.55	78%	1.59	1.00
82522 A	CO11	MIX SULF HG	Milled	0.068	3.053	94%	3.56	78%	1.99	1.25
	Average :				3.170	94%	3.61	80%	1.77	1.25

Source: KCA (2019)

Table 13.29 Cerro del Gallo KCA Laboratory Bottle Roll Leach Tests – Copper

KCASample No.	Description	MineralType	Type	Calculated p80, mm	Head Copper, mg Cu/kg	Est. Cu Extracted, %	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
82512 B	CO1	OX	Pulv.	--	480	26%	0.40	2.75
82512 B	CO1	OX	Milled	0.074	480	23%	1.48	2.00
82512 B	CO1	OX	Milled	0.053	480	26%	0.77	2.25
	Average :				480	25%	0.88	2.33
82513 B	CO2	MIX OX	Pulv.	--	600	60%	1.29	2.50
82513 B	CO2	MIX OX	Milled	0.093	600	64%	1.51	1.75
82513 B	CO2	MIX OX	Milled	0.054	600	86%	2.00	2.00
	Average :				600	70%	1.60	2.08
82514 A	CO3	MIX OX	Pulv.	--	461	64%	1.46	1.50
82514 A	CO3	MIX OX	Milled	0.095	461	65%	1.83	1.00
82514 A	CO3	MIX OX	Milled	0.054	461	80%	2.09	1.25
	Average :				461	70%	1.79	1.25
82515 A	CO4	MIX SULF 1	Pulv.	--	691	78%	2.05	2.50
82515 A	CO4	MIX SULF 1	Milled	0.107	691	69%	1.82	2.00
82515 A	CO4	MIX SULF 1	Milled	0.079	691	74%	1.91	2.25
	Average :				691	74%	1.93	2.25
82516 A	CO5	MIX SULF 1	Pulv.	--	2310	59%	5.58	2.00
82516 A	CO5	MIX SULF 1	Milled	0.101	2310	69%	5.84	1.25
82516 A	CO5	MIX SULF 1	Milled	0.071	2310	60%	6.40	2.00
	Average :				2310	63%	5.94	1.75
82517 A	CO6	MIX SULF 2	Pulv.	--	992	69%	2.45	2.25
82517 A	CO6	MIX SULF 2	Milled	0.106	992	53%	2.06	1.50
82517 A	CO6	MIX SULF 2	Milled	0.086	992	60%	2.48	1.75
	Average :				992	61%	2.33	1.83
82518 B	CO7	MIX SULF 2	Pulv.	--	1200	75%	2.90	1.75
82518 B	CO7	MIX SULF 2	Milled	0.100	1200	76%	3.20	1.25
82518 B	CO7	MIX SULF 2	Milled	0.063	1200	72%	3.22	1.50
	Average :				1200	74%	3.11	1.50
82519 B	CO8	SULF HG	Pulv.	--	711	42%	1.41	1.25
82519 B	CO8	SULF HG	Milled	0.203	711	23%	0.88	1.00
82519 B	CO8	SULF HG	Milled	0.132	711	25%	0.74	1.00
	Average :				711	30%	1.01	1.08
82520 A	CO9	SULF	Pulv.	--	731	64%	1.67	1.25
82520 A	CO9	SULF	Milled	0.103	731	56%	1.87	1.00
82520 A	CO9	SULF	Milled	0.070	731	63%	2.18	1.00
	Average :				731	61%	1.91	1.08
82521 A	CO10	OX HG	Pulv.	--	686	44%	1.18	3.25
82521 A	CO10	OX HG	Milled	0.098	686	39%	1.91	2.50
82521 A	CO10	OX HG	Milled	0.065	686	40%	1.96	2.75
	Average :				686	41%	1.68	2.83
82522 A	CO11	MIX SULF HG	Pulv.	--	671	69%	1.74	1.50
82522 A	CO11	MIX SULF HG	Milled	0.116	671	63%	1.59	1.00
82522 A	CO11	MIX SULF HG	Milled	0.068	671	71%	1.99	1.25
	Average :				671	68%	1.77	1.25

Source: KCA (2019)

Compacted permeability test work was performed on tailings material from each column utilizing material previously crushed, conventional and HPGR, material. The purpose of the compacted permeability test work was to examine the permeability of the crushed material under compaction loading equivalent to heap heights of 10, 20, 40, and 80 meters of overall heap height.

The flow rate, slump, pellet breakdown and solution color and clarity are all monitored to provide meaningful indications and to help judge what represents a “Pass” or “Fail.” The results are summarized in Table 13.30.

Notes on the pass/fail criteria:

1. In KCA’s compacted agglomeration tests, a slump of over 10% may be an indication of heap instability and may require additional cement. If things worked perfectly, a lower slump with higher cement levels could be expected.

2. A typical heap leach solution application rate of 10 to 12 liters per hour per square meter is utilized when examining the agglomeration data. When examining results from this type of agglomeration test a measured flow of ten times (10X) the heap design rate is considered a “pass.” A measured flow less than 10X the heap design flow is not necessarily a failure. If there are enough tests with enough consistency between tests, and all other points indicate a “pass,” and then sometimes a test will pass with less than the 10X flow. However, a test will not likely pass at 1X and probably not at 4X.

3. In examining the Pellet Breakdown, about 10% is marginally acceptable and anything higher is a failure. In general, a higher range is allowable in Pellet Breakdown as this is a subjective value based on the visual observation of the pellets after the test by the technicians performing the test. When the samples tested are not agglomerated using cement, this test is not applicable.

4. Solution color and clarity typically is an indicator of agglomerate failure and fines migration. This information is utilized in coordination with both slump as well as Pellet Breakdown to determine if the test passes.

The results of the compacted permeability test work indicate the potential need for additional controls during heap leach operation to monitor slump, which may lead to higher cement additions to achieve the design criteria of an 80 meter heap leach pad overall height. Cement addition for the process was chosen based on compacted permeability flow rate results.

Table 13.30 Cerro del Gallo KCA Laboratory Compacted Permeability Test Results

KCA Sample No.	Sample Description	Mineral Type	Crush Type	Calc. p80 Size, mm	Test Phase	Cement Added, kg/MT	Cure Time, days	Effective Height, meter	Flow Rate, LpHr/m^2	Flow Result Pass/Fail	Saturated Permeability, cm/sec	Cum. Slump, % Slump
82565	CO1	OX	HPGR	3.3	Primary	5	1	20	0	Fail	0.0E+00	0%
					Stage Load			40	0	Fail	0.0E+00	N/A
					Stage Load			80	0	Fail	0.0E+00	N/A
82568	CO2	MIX OX	HPGR	4.3	Primary	0	1	20	0	Fail	0.0E+00	0%
					Stage Load			40	0	Fail	0.0E+00	N/A
					Stage Load			80	0	Fail	0.0E+00	N/A
82574	CO4	MIX SULF 1	HPGR	4.7	Primary	0	1	20	0	Fail	0.0E+00	0%
					Stage Load			40	0	Fail	0.0E+00	N/A
					Stage Load			80	0	Fail	0.0E+00	N/A
82953 A	CO4	MIX SULF 1	HPGR	4.7	Primary	5	3	10	505	Pass	1.4E-02	10%
					Stage Load			20	29	Fail	8.1E-04	16%
					Stage Load			40	2	Fail	5.6E-05	21%
					Stage Load			80	0	Fail	0.0E+00	N/A
82901	CO9	SULF	HPGR	5.6	Primary	0	1	20	10	Fail	2.8E-04	0%
					Stage Load			40	6	Fail	1.7E-04	3%
					Stage Load			80	0	Fail	0.0E+00	6%
82953 B	CO9	SULF	HPGR	5.6	Primary	0	3	20	233	Pass	6.5E-03	3%
					Stage Load			40	76	Pass	2.1E-03	8%
					Stage Load			80	39	Fail	1.1E-03	11%
82955 B	CO9	SULF	HPGR	5.6	Primary	5	1	10	3,339	Pass	9.3E-02	2%
					Stage Load			20	2,394	Pass	6.7E-02	7%
					Stage Load			40	1,084	Pass	3.0E-02	12%
					Stage Load			80	277	Pass	7.7E-03	17%
82552	CO1	OX	Conventional	5.9	Primary	10	1	10	3,627	Pass	1.0E-01	7%
					Stage Load			20	1,306	Pass	3.6E-02	13%
					Stage Load			40	218	Pass	6.1E-03	19%
					Stage Load			80	23	Fail	6.4E-04	24%
82555	CO2	MIX OX	Conventional	6.2	Primary	10	1	10	3,591	Pass	1.0E-01	2%
					Stage Load			20	2,989	Pass	8.3E-02	9%
					Stage Load			40	1,054	Pass	2.9E-02	14%
					Stage Load			80	200	Pass	5.6E-03	19%
82558	CO7	MIX SULF 2	Conventional	6.3	Primary	10	1	10	4,423	Pass	1.2E-01	0%
					Stage Load			20	2,428	Pass	6.7E-02	3%
					Stage Load			40	1,652	Pass	4.6E-02	8%
					Stage Load			80	107	Pass	3.0E-03	14%
82561	CO8	SULF HG	Conventional	6.6	Primary	10	1	10	4,430	Pass	1.2E-01	0%
					Stage Load			20	4,379	Pass	1.2E-01	2%
					Stage Load			40	4,255	Pass	1.2E-01	6%
					Stage Load			80	4,138	Pass	1.1E-01	9%
82967 A	CO1/CO10	OX OX HG	HPGR	4.1	Primary	5	7	10	282	Pass	7.8E-03	5%
					Stage Load			20	7	Fail	1.9E-04	9%
					Stage Load			40	0	Fail	0.0E+00	N/A
					Stage Load			80	0	Fail	0.0E+00	N/A
82967 A	CO1/CO10	OX OX HG	HPGR	4.1	Primary	10	7	10	942	Pass	2.6E-02	2%
					Stage Load			20	59	Fail	1.6E-03	8%
					Stage Load			40	0	Fail	0.0E+00	13%
					Stage Load			80	0	Fail	0.0E+00	N/A

KCA Sample No.	Sample Description	Mineral Type	Crush Type	Calc. p80 Size, mm	Test Phase	Cement Added, kg/MT	Cure Time, days	Effective Height, meter	Flow Rate, LpHr/m^2	Flow Result Pass/Fail	Saturated Permeability, cm/sec	Cum. Slump, % Slump
82967 A	CO1/CO10	OX OX HG	HPGR	4.1	Primary	15	7	10	12,348	Pass	3.4E-01	2%
					Stage Load			20	2,145	Pass	6.0E-02	8%
					Stage Load			40	231	Pass	6.4E-03	14%
					Stage Load			80	45	Fail	1.3E-03	20%
82967 A	CO1/CO10	OX OX HG	HPGR	4.1	Primary	20	7	10	17,140	Pass	4.8E-01	2%
					Stage Load			20	12,305	Pass	3.4E-01	6%
					Stage Load			40	2,039	Pass	5.7E-02	13%
					Stage Load			80	140	Pass	3.9E-03	21%
82968 A	CO4/CO5/ CO7	MIX SULF 1 MIX SULF 2	HPGR	4.7	Primary	5	7	10	2,181	Pass	6.1E-02	6%
					Stage Load			20	378	Pass	1.1E-02	13%
					Stage Load			40	107	Pass	3.0E-03	17%
					Stage Load			80	0	Fail	0.0E+00	21%
82968 A	CO4/CO5/ CO7	MIX SULF 1 MIX SULF 2	HPGR	4.7	Primary	10	7	10	13,877	Pass	3.9E-01	2%
					Stage Load			20	4,628	Pass	1.3E-01	7%
					Stage Load			40	857	Pass	2.4E-02	13%
					Stage Load			80	142	Pass	3.9E-03	19%
82968 A	CO4/CO5/ CO7	MIX SULF 1 MIX SULF 2	HPGR	4.7	Primary	15	7	10	16,864	Pass	4.7E-01	2%
					Stage Load			20	11,598	Pass	3.2E-01	6%
					Stage Load			40	2,628	Pass	7.3E-02	12%
					Stage Load			80	315	Pass	8.8E-03	19%
82968 A	CO4/CO5/ CO7	MIX SULF 1 MIX SULF 2	HPGR	4.7	Primary	20	7	10	17,255	Pass	4.8E-01	0%
					Stage Load			20	16,308	Pass	4.5E-01	4%
					Stage Load			40	12,200	Pass	3.4E-01	10%
					Stage Load			80	4,158	Pass	1.2E-01	17%
82969 A	CO6	VSED	HPGR	4.6	Primary	5	7	10	3,312	Pass	9.2E-02	5%
					Stage Load			20	610	Pass	1.7E-02	11%
					Stage Load			40	172	Pass	4.8E-03	15%
					Stage Load			80	37	Fail	1.0E-03	20%
82969 A	CO6	VSED	HPGR	4.6	Primary	10	7	10	13,565	Pass	3.8E-01	3%
					Stage Load			20	4,121	Pass	1.1E-01	7%
					Stage Load			40	1,011	Pass	2.8E-02	13%
					Stage Load			80	195	Pass	5.4E-03	19%
82969 A	CO6	VSED	HPGR	4.6	Primary	15	7	10	16,989	Pass	4.7E-01	2%
					Stage Load			20	15,357	Pass	4.3E-01	5%
					Stage Load			40	10,259	Pass	2.8E-01	11%
					Stage Load			80	3,141	Pass	8.7E-02	17%
82969 A	CO6	VSED	HPGR	4.6	Primary	20	7	10	17,232	Pass	4.8E-01	1%
					Stage Load			20	16,374	Pass	4.5E-01	5%
					Stage Load			40	13,180	Pass	3.7E-01	10%
					Stage Load			80	4,792	Pass	1.3E-01	18%
82970 A	CO8/CO9/ CO11	VSED/INT	HPGR	4.7	Primary	5	7	10	12,815	Pass	3.6E-01	1%
					Stage Load			20	6,526	Pass	1.8E-01	6%
					Stage Load			40	2,648	Pass	7.4E-02	10%
					Stage Load			80	766	Pass	2.1E-02	14%
82970 A	CO8/CO9/ CO11	VSED/INT	HPGR	4.7	Primary	10	7	10	16,245	Pass	4.5E-01	0%
					Stage Load			20	14,525	Pass	4.0E-01	3%
					Stage Load			40	9,772	Pass	2.7E-01	8%
					Stage Load			80	3,894	Pass	1.1E-01	14%
82970 A	CO8/CO9/ CO11	VSED/INT	HPGR	4.7	Primary	15	7	10	16,361	Pass	4.5E-01	1%
					Stage Load			20	15,449	Pass	4.3E-01	3%
					Stage Load			40	12,959	Pass	3.6E-01	7%
					Stage Load			80	6,930	Pass	1.9E-01	13%
82970 A	CO8/CO9/ CO11	VSED/INT	HPGR	4.7	Primary	20	7	10	16,841	Pass	4.7E-01	1%
					Stage Load			20	16,295	Pass	4.5E-01	3%
					Stage Load			40	14,121	Pass	3.9E-01	8%
					Stage Load			80	8,588	Pass	2.4E-01	14%

Source: KCA (2019)

Column leach tests were conducted utilizing both HPGR crushed material (p100 of 19 mm, average p80 of 4.7 mm) and conventionally crushed (p100 of 9.5 mm, average p80 of 6.3 mm). The results are presented in Table 13.31, **Error! Reference source not found.** and **Error! Reference source not found.**.

Table 13.31 Cerro del Gallo KCA Laboratory Column Leach Test Summary

KCA Sample No.	Description	Mineral Type	Crushing Type	Head Sulfide Sulfur, %	Head Assay, mg Cu/kg	Est. Cu Ext., %[1]	Calculated Head, gms Au/MT	Extracted, % Au	Calculated Head, gms Ag/MT	Extracted, % Ag	Calculated Tail p80 Size, mm	Days of Leach	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT	Addition Cement, kg/MT
82512 B	CO1	OX	Conventional	<0.01	440	13%	0.297	73%	8.31	60%	5.9	143	2.97	2.85	--
82513 B	CO2	MIX OX	Conventional	0.05	540	45%	1.265	70%	8.87	77%	6.2	143	2.77	2.54	--
82518 B	CO7	MIX SULF 2	Conventional	1.13	960	58%	0.862	56%	4.14	62%	6.3	143	3.01	1.76	--
82519 B	CO8	SULF HG	Conventional	1.23	635	14%	1.881	41%	1.52	40%	6.6	143	1.94	1.26	--
82537 A	CO1	OX	HPGR	<0.01	430	11%	0.314	71%	11.11	51%	3.3	135	2.27	2.85	2.07
82538 A	CO2	MIX OX	HPGR	0.05	553	44%	1.353	74%	10.65	74%	4.3	135	2.44	3.05	--
82539 A	CO3	MiX OX	HPGR	0.20	338	48%	0.881	69%	25.04	89%	4.3	135	2.87	1.53	--
82540 A	CO4	MIX SULF 1	HPGR	0.94	625	58%	0.771	69%	12.02	82%	4.7	135	3.15	2.64	--
82541 A	CO5	MIX SULF 1	HPGR	1.60	1922	67%	0.372	61%	19.92	44%	5.0	128	5.15	2.52	--
82542 A	CO6	MIX SULF 2	HPGR	1.76	789	47%	0.474	55%	18.25	60%	4.4	128	2.88	2.29	--
82543 A	CO7	MIX SULF 2	HPGR	1.13	982	65%	0.905	62%	6.43	54%	5.0	128	3.28	2.13	--
82544 A	CO8	SULF HG	HPGR	1.23	623	20%	1.893	54%	4.38	23%	5.3	128	2.07	1.26	--
82545 A	CO9	SULF	HPGR	0.84	616	48%	1.634	65%	5.02	63%	5.6	128	2.67	1.26	--
82546 A	CO10	OX HG	HPGR	0.04	566	33%	1.575	81%	9.39	75%	4.5	135	2.31	3.04	2.03
82547 A	CO11	MIX SULF HG	HPGR	0.43	519	56%	3.262	60%	3.22	71%	5.7	122	2.03	1.52	2.03

Source: KCA (2019)

The recovery performance between the HPGR and Conventionally crushed samples is shown in Table 13.32 and varied by mineral type. Gold recoveries increased in the HPGR crushed material for Sulfide High Grade, Mixed Sulfide and Mixed Oxides but also decreased for the Oxide sample. Silver recovery was lower overall in the HPGR crushed material most likely due to kinetics which slowed due to the increased copper recoveries. The new ongoing test work addressed this issue by increasing the NaCN solution concentrations in the column leach tests and introducing SART to process pregnant solution and better mimic the expected production system.

Table 13.32 Cerro del Gallo Current Test Work Conventional versus HPGR Crushing

Mineral Type	Crushing Type	Head Assay, mg Cu/kg	Est. Cu Ext., %[1]	Calculated Head, gms Au/MT	Extracted, % Au	Calculated Head, gms Ag/MT	Extracted, % Ag	Calculated Tail p80 Size, mm	Days of Leach	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT	Addition Cement, kg/MT
OX	Conventional	440	13%	0.297	73%	8.31	60%	5.9	143	2.97	2.85	--
MIX OX	Conventional	540	45%	1.265	70%	8.87	77%	6.2	143	2.77	2.54	--
MIX SULF 2	Conventional	960	58%	0.862	56%	4.14	62%	6.3	143	3.01	1.76	--
SULF HG	Conventional	635	14%	1.881	41%	1.52	40%	6.6	143	1.94	1.26	--
OX	HPGR	430	11%	0.314	71%	11.11	51%	3.3	135	2.27	2.85	2.07
MiX OX	HPGR	446	46%	1.117	72%	17.85	82%	4.3	135	2.66	2.29	
MIX SULF 2	HPGR	1080	59%	0.631	62%	14.16	60%	4.8	130	3.62	2.40	
SULF HG	HPGR	623	20%	1.893	54%	4.38	23%	5.3	128	2.07	1.26	--
HPGR minus Converntional Crushing												
OX			-2%		-2%		-9%					
MIX OX			-1%		4%		-3%					
MIX SULF 2			-10%		13%		27%					
SULF HG			32%		31%		42%					

Source: KCA (2019)

Historic test work also supports the HPGR crushing over conventional crushing. Oxide and Mixed Oxides samples crushed to minus 8 mm and column leached 115 days were crushed with both conventional crushers and HPGR and compared. The results are presented in Table 13.33 and indicate that metal dissolutions for the HPGR crushed samples achieved higher average metal dissolutions. Note that Conventional S2 (Conv. S2) samples utilized higher cement during agglomeration (6 kg/t) and also included higher cyanide addition.

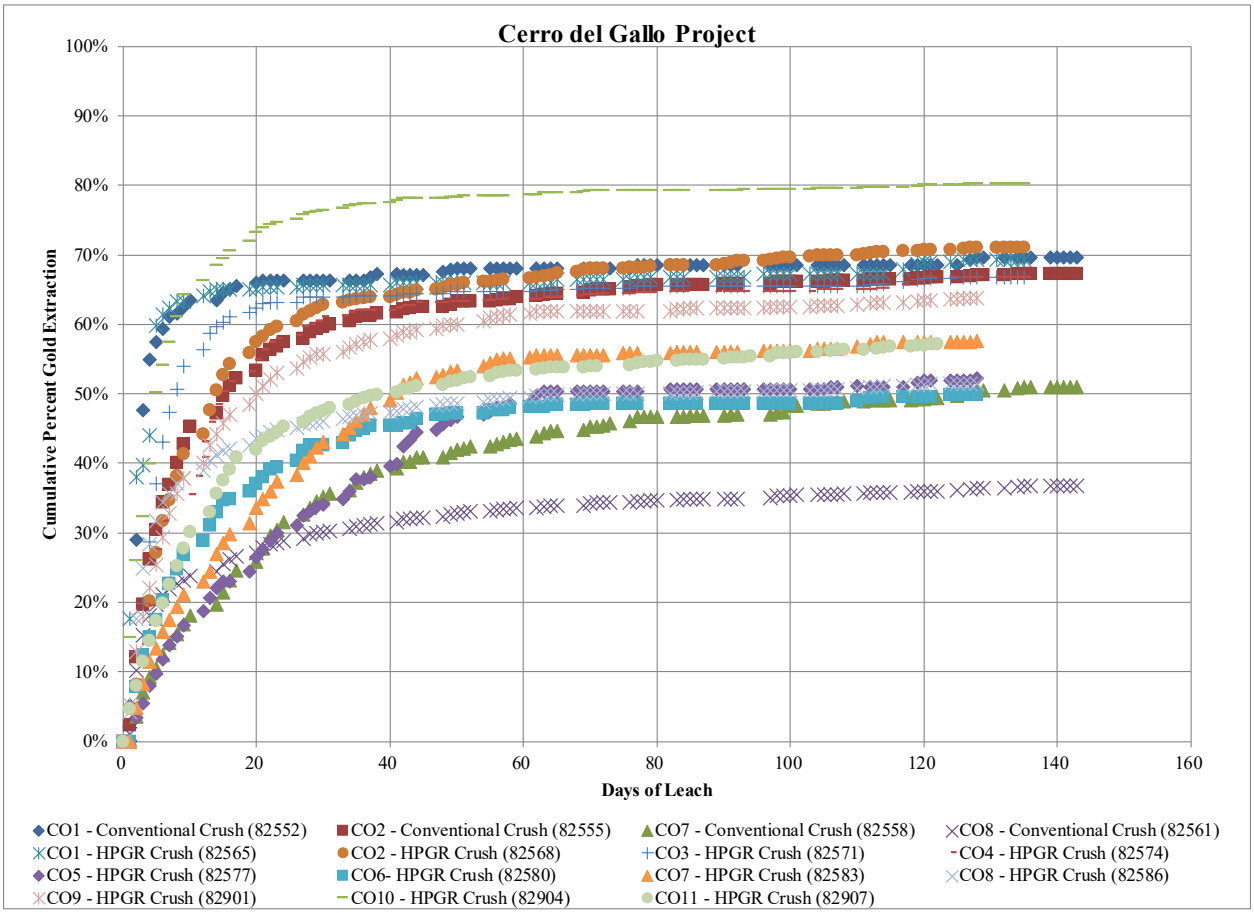
Table 13.33 Cerro del Gallo Historical Test Work Conventional versus HPGR Crushing

Crush Type	Reagents (kg/t)			Assays (g/t)						Dissolutions (%)		
	NaCN Add.	NaCN Cons.	Lime Add.	Calc. Head (Au)	Res. (Au)	Calc. Head (Ag)	Res. (Ag)	Calc. Head (Cu)	Res. (Cu)	Au	Ag	Cu
Conv.	1.36	1.08	1.28	1.20	0.58	13	8	786	553	52.2	38.0	31.5
HPGR	1.38	1.15	1.02	1.21	0.42	12	6	618	376	65.6	52.1	38.8
Conv. S2	4.39	4.39	1.03	1.24	0.53	12	6	1355	348	57.6	49.9	73.4

Source: Primero (2012)

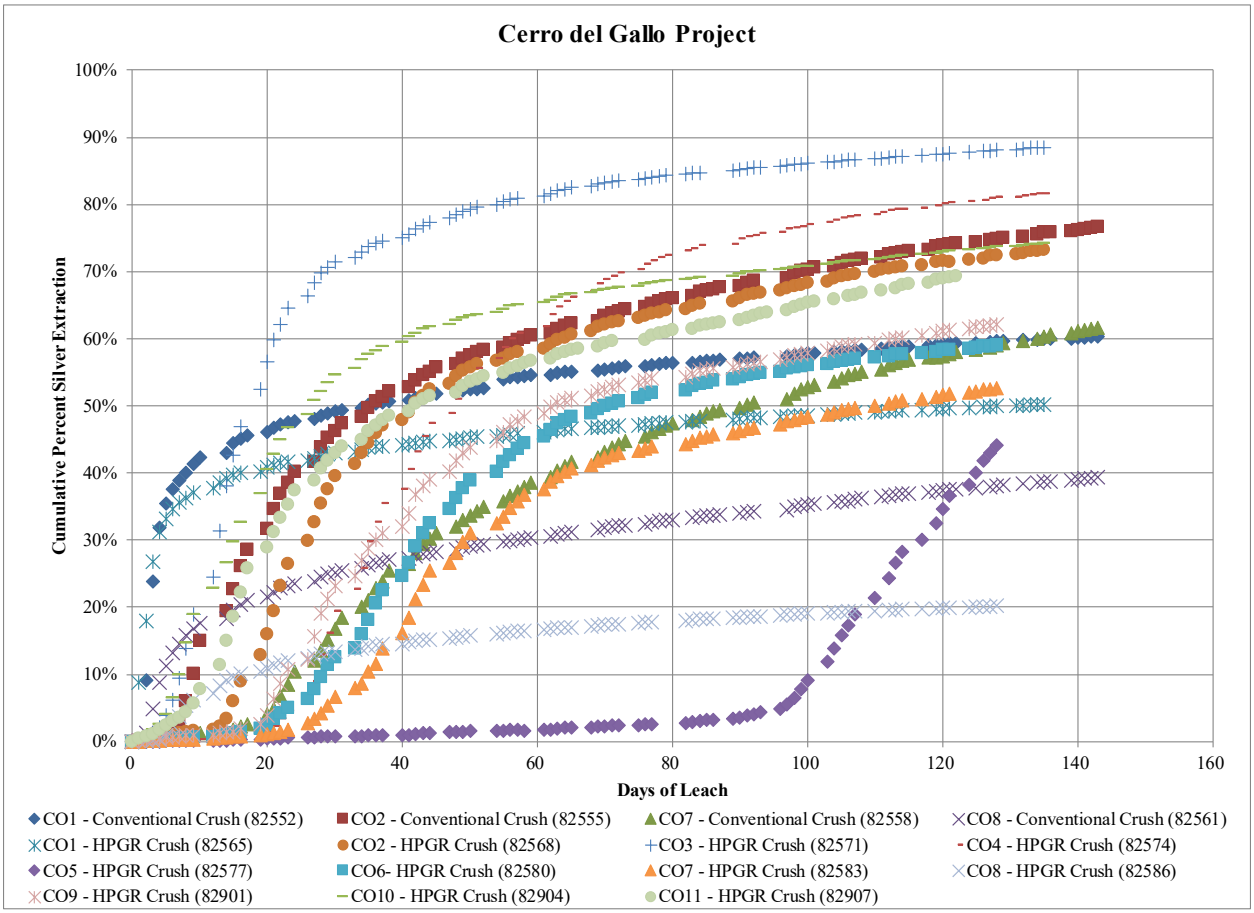
Given the test work results from the current and historic column leach tests, HPGR crushing will most likely achieve higher recoveries in the extraction of gold, silver and copper given the proper NaCN solution concentrations to aid in the dissolution kinetics.

Figure 13.3: Cerro del Gallo KCA Laboratory Column Leach Tests Curves – Gold



Source: KCA (2019)

Figure 13.4: Cerro del Gallo KCA Laboratory Column Leach Tests Curves – Silver



Source: KCA (2019)

13.6 2020 Metallurgical Testing

The 2020 test work was conducted by KCA and included bottle roll tests at various NaCN concentrations (0.5, 1, 2 and 5, gpL NaCN), compacted permeability tests with varying cement additions (5, 10, 15 and 20 kg/t cement), and SART test work including additional column leach tests in closed cycle. The tests utilized HPGR crushed material. The goal of the test work is to expand upon the successful results of the 2018 and 2019 metallurgical test work conducted by KCA and optimize process parameters for the project.

13.6.1 Bottle Roll Tests

Cyanide bottle roll leach tests were run for a period of two-hundred and forty (240) hours with solution sampling performed at 2, 4, 8, 24, 48, 72, 96, 120, 144, 168, 192, 216, and 240 hours. The tests utilized HPGR crushed material. NaCN was added and maintained at 0.5, 1.0, 2.0, and 5.0 grams per liter of solution. The pH of the solution was maintained at 10.5 to 11.0 with the addition of hydrated lime. Results of the cyanide bottle roll leach test work are summarized in Table 13.34.

The previous, pulverized bottle roll leach tests conducted by KCA as compared to the new HPGR crushed bottle roll tests give an idea of percent of recoverable gold extracted at HPGR product sizes. Table 13.35 shows the percent of cyanide recoverable gold that the HPGR product could potentially get, and the theoretical maximum recoverable gold shown in the pulverized samples.

Table 13.34 Cerro del Gallo KCA Laboratory Bottle Roll Test Results

KCA Sample No.	Mineral Type	Calc. p80 Size, mm	Target NaCN, g/L	Calculated Head, gms Au/MT	Au Extracted, %	Calculated Head, gms Ag/MT	Ag Extracted, %	Calculated Head, gms Cu/MT	Cu Extracted, %	Cons-umption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
82537 A	OX	6.45	0.50	0.281	79%	9.34	46%	425	23%	0.39	3.00
82537 A	OX	6.45	1.00	0.231	70%	10.21	49%	455	23%	0.57	2.50
82537 A	OX	6.45	2.00	0.438	84%	10.72	48%	491	22%	0.99	2.25
82537 A	OX	6.45	5.00	0.283	71%	10.91	49%	463	22%	1.69	2.00
		6.45	**2.13**	**0.308**	**76%**	**10.30**	**48%**	**458**	**23%**	**0.91**	**2.44**
82538 A	MIX OX	6.81	0.50	1.170	69%	10.01	56%	590	44%	1.01	2.00
82538 A	MIX OX	6.81	1.00	1.093	68%	10.50	59%	713	49%	1.16	2.00
82538 A	MIX OX	6.81	2.00	1.268	62%	10.31	64%	693	56%	1.61	1.50
82538 A	MIX OX	6.81	5.00	1.282	66%	9.71	75%	645	60%	2.41	1.25
82539 A	MiX OX	6.26	0.50	0.769	58%	25.59	65%	365	50%	0.73	1.75
82539 A	MiX OX	6.26	1.00	0.764	64%	26.24	79%	420	62%	0.92	1.75
82539 A	MiX OX	6.26	2.00	0.717	65%	27.59	76%	456	60%	1.27	1.75
82539 A	MiX OX	6.26	5.00	0.899	65%	25.20	78%	436	62%	1.95	1.00
		6.57	**2.13**	**0.995**	**65%**	**18.14**	**69%**	**540**	**55%**	**1.38**	**1.63**
82540 A	MIX SULF 1	7.26	0.50	0.683	59%	13.76	54%	695	57%	1.41	3.00
82540 A	MIX SULF 1	7.26	1.00	0.690	60%	12.68	62%	731	62%	1.09	2.50
82540 A	MIX SULF 1	7.26	2.00	0.716	56%	12.27	66%	803	64%	2.19	2.25
82540 A	MIX SULF 1	7.26	5.00	0.759	65%	12.54	72%	773	70%	3.37	2.25
82541 A	MIX SULF 1	5.72	0.50	0.332	44%	25.54	24%	2474	68%	4.46	2.75
82541 A	MIX SULF 1	5.72	1.00	0.328	52%	27.66	33%	2509	73%	5.31	2.25
82541 A	MIX SULF 1	5.72	2.00	0.332	49%	25.90	39%	2573	81%	6.39	1.75
82541 A	MIX SULF 1	5.72	5.00	0.392	63%	28.38	66%	2562	87%	8.09	1.25
82542 A	MIX SULF 2	5.88	0.50	0.428	42%	19.68	40%	939	46%	1.59	2.50
82542 A	MIX SULF 2	5.88	1.00	0.461	43%	19.07	47%	966	53%	1.82	2.50
82542 A	MIX SULF 2	5.88	2.00	0.469	47%	20.65	47%	1047	58%	2.53	2.25
82542 A	MIX SULF 2	5.88	5.00	0.419	44%	20.14	51%	993	62%	3.91	1.50
82543 A	MIX SULF 2	5.96	0.50	0.830	48%	6.21	39%	1110	59%	2.05	2.50
82543 A	MIX SULF 2	5.96	1.00	0.838	50%	6.13	49%	1211	67%	2.81	2.25
82543 A	MIX SULF 2	5.96	2.00	0.787	57%	4.95	57%	1160	69%	3.18	1.75
82543 A	MIX SULF	5.96	5.00	0.971	52%	5.87	58%	1213	73%	4.83	1.25
		6.22	**2.13**	**0.590**	**52%**	**16.34**	**50%**	**1360**	**66%**	**3.44**	**2.16**
82544 A	SULF HG	5.99	0.50	1.866	44%	2.82	27%	680	19%	0.68	1.75
82544 A	SULF HG	5.99	1.00	1.788	43%	2.81	24%	700	18%	0.78	1.50
82544 A	SULF HG	5.99	2.00	1.859	47%	2.17	32%	776	19%	1.27	1.50
82544 A	SULF HG	5.99	5.00	1.822	48%	2.07	36%	708	25%	2.36	1.00
		5.99	**2.13**	**1.834**	**46%**	**2.46**	**30%**	**716**	**20%**	**1.27**	**1.44**
82545 A	SULF	5.25	0.50	1.549	46%	4.11	50%	634	42%	1.08	1.75
82545 A	SULF	5.25	1.00	1.706	46%	4.58	52%	695	48%	1.44	1.25
82545 A	SULF	5.25	2.00	1.550	62%	5.19	59%	812	57%	2.05	1.25
82545 A	SULF	5.25	5.00	1.613	57%	5.33	59%	795	62%	3.15	1.00
		5.25	**2.13**	**1.605**	**53%**	**4.80**	**55%**	**734**	**52%**	**1.93**	**1.31**
82546 A	OX HG	6.24	0.50	1.391	81%	8.65	64%	606	43%	0.93	3.50
82546 A	OX HG	6.24	1.00	1.580	75%	10.13	60%	643	49%	1.18	3.00
82546 A	OX HG	6.24	2.00	1.656	75%	8.07	74%	645	49%	1.65	2.75
82546 A	OX HG	6.24	5.00	1.485	77%	9.76	72%	664	50%	2.41	2.25
		6.24	**2.13**	**1.528**	**77%**	**9.15**	**68%**	**639**	**48%**	**1.54**	**2.88**
82547 A	MIX SULF HG	5.55	0.50	3.139	47%	3.32	53%	568	60%	0.82	1.75
82547 A	MIX SULF HG	5.55	1.00	3.005	53%	3.48	56%	634	67%	1.51	1.50
82547 A	MIX SULF HG	5.55	2.00	3.116	54%	4.31	47%	680	69%	2.25	1.25
82547 A	MIX SULF HG	5.55	5.00	3.118	55%	2.81	79%	660	73%	2.71	1.00
		5.55	**2.13**	**3.095**	**52%**	**3.48**	**59%**	**635**	**67%**	**1.82**	**1.38**

Source: KCA (2020)

Table 13.35 Cerro del Gallo KCA Laboratory Bottle Roll Test Results Comparison

KCA Sample No.	Mineral Type	Crushing Type	Calc. p80 Size, mm	Target NaCN, g/L	Calculated Head, gms Au/MT	Au Extracted, %	Calculated Head, gms Ag/MT	Ag Extracted, %	Calculated Head, gms Cu/MT	Cu Extracted, %	Leach Time, Hours	Consumption NaCN, kg/MT	Addition Ca(OH)2, kg/MT
New Test Work													
82537 A	OX	HPGR	6.45	0.5 to 5.0	0.308	76%	10.30	48%	458	23%	240	0.91	2.44
82539 A	MiX OX	HPGR	6.57	0.5 to 5.0	0.995	65%	18.14	69%	540	55%	240	1.38	1.63
82543 A	MIX SULF	HPGR	6.22	0.5 to 5.0	0.590	52%	16.34	50%	1360	66%	240	3.44	2.16
82544 A	SULF HG	HPGR	5.99	0.5 to 5.0	1.834	46%	2.46	30%	716	20%	240	1.27	1.44
82545 A	SULF	HPGR	5.25	0.5 to 5.0	1.605	53%	4.80	55%	734	52%	240	1.93	1.31
82546 A	OX HG	HPGR	6.24	0.5 to 5.0	1.528	77%	9.15	68%	639	48%	240	1.54	2.88
82547 A	MIX SULF HG	HPGR	5.55	0.5 to 5.0	3.095	52%	3.48	59%	635	67%	240	1.82	1.38
Old Test Work													
82512 B	OX	Milled	0.053	1.00	0.260	90%	14.60	52%	480	24%	96	1.13	2.13
82514 A	MIX OX	Milled	0.054	1.00	0.985	91%	16.94	80%	531	74%	96	1.86	1.50
82518 B	MIX SULF	Milled	0.063	1.00	0.598	82%	19.03	57%	1264	67%	96	3.37	1.69
82519 B	SULF HG	Milled	0.132	1.00	1.810	80%	2.47	49%	711	24%	96	0.81	1.00
82520 A	SULF	Milled	0.07	1.00	1.855	90%	6.05	63%	731	59%	96	2.03	1.00
82521 A	OX HG	Milled	0.065	1.00	1.436	92%	10.08	87%	686	39%	96	1.94	2.63
82522 A	MIX SULF HG	Milled	0.068	1.00	3.148	94%	3.56	78%	671	67%	96	1.79	1.13

Source: KCA (2020)

13.6.2 Column Leach Tests

Column leach tests were completed on HPGR crushed material. They were leached for a total of 117 days with a mixed sulfide sample number 82541 leached for 147 days. The results are presented in Table 13.36.

Table 13.36 Cerro del Gallo KCA Laboratory Column Leach Test with SART Results

KCA Sample No.	Mineral Type	Average Head, gms Au/MT	Overall Extracted, % Au	Overall Average, gms Ag/MT	Overall Extracted, % Ag	Overall Average, gms Cu/kg	Overall Extracted, % Cu	Days of Leach	Overall Consumption NaCN, kg/MT	Column Addition $Ca(OH)_2$, kg/MT	Addition Cement, kg/MT
82538	MIX OX	1.21	71%	12.0	62%	783	65%	117	1.84	0	10
82539	MIX OX	0.81	67%	21.9	78%	493	69%	117	1.45	0	10
82540	MIX SULF 1	0.75	69%	13.9	65%	903	73%	117	1.82	0	10
82541	MIX SULF 1	0.39	63%	27.8	52%	2568	82%	147	3.97	0	10
82544	SULF HG	1.78	46%	2.9	27%	1412	64%	117	1.91	0	5

Source: KCA (2020)

The ongoing column leach tests were compared to the previous column test work conducted with HPGR crushing at the same particle size. The tests were compared by material type. The results indicate that the gold recoveries are similar for mixed oxide and mixed sulfide but the sulfide material has a lower gold recovery in the latest test work. The silver recoveries are similar for the mixed oxide material but are higher in the new test work for both the mixed sulfide and sulfide. The difference in the previous column leach test work conducted by KCA versus the new test work is presented in

Table 13.37.

Table 13.37 Cerro del Gallo KCA Laboratory Column Leach Test Interim Results Comparison

Sample No.	Code (Material)	Head Grade, g/t		Calc. Head, g/kg	Crush Size, mm	Recovery, %			Reagent Consumption, kg/ton			Total Sulfur, %	Sulfide Sulfur, %	Time, days
		Au	Ag	Cu	P80	Au	Ag	Cu	NaCN	Cement	Lime			
82537 A	WEATHERED	0.27	14.1	480	3.3	71.0%	51.0%	11.0%	2.27	2	2.85	0.05	0.01	135
82538 A	MIXED_O	1.14	12.5	600	4.3	74.0%	74.0%	44.0%	2.44		3.05	0.14	0.05	135
82539 A	MIXED_O	0.73	20.3	461	4.3	69.0%	89.0%	48.0%	2.87		1.53	0.30	0.20	135
82540 A	MIXED_S	0.74	14.5	691	4.7	69.0%	82.0%	58.0%	3.15		2.64	1.13	0.94	135
82541 A	MIXED_S	0.38	29.7	2310	5	61.0%	44.0%	67.0%	5.15		2.52	1.80	1.60	128
82542 A	MIXED_S	0.50	22.4	992	4.4	55.0%	60.0%	47.0%	2.88		2.29	2.06	1.76	128
82543 A	MIXED_S	0.83	6.5	1200	5	62.0%	54.0%	65.0%	3.28		2.13	1.29	1.13	128
82544 A	FRESH	1.69	2.4	711	5.3	54.0%	23.0%	20.0%	2.07		1.26	1.47	1.23	128
82545 A	FRESH	1.90	5.8	731	5.6	65.0%	63.0%	48.0%	2.67		1.26	0.95	0.84	128
82546 A	WEATHERED	1.36	9.7	686	4.5	81.0%	75.0%	33.0%	2.31	2	3.04	0.12	0.04	135
82547 A	MIXED_S	3.06	3.4	671	5.7	60.0%	71.0%	56.0%	2.03	2	1.52	0.53	0.43	122
82538 A	MIXED_O	1.21	12.0	783	4.538	71.0%	62.0%	65.0%	1.84	10				
82539 A	MIXED_O	0.81	21.9	493	4.053	67.0%	78.0%	69.0%	1.45	10				
82540 A	MIXED_S	0.75	13.9	903	5.774	69.0%	65.0%	73.0%	1.82	10				
82541 A	MIXED_S	0.39	27.8	2568	5.426	63.0%	52.0%	82.0%	3.97	10				
82544 A	FRESH	1.78	2.9	1412	6.163	46.0%	27.0%	64.0%	1.91	5				
WEATHERED						**76%**	**63%**	**22%**						
MIXED_O						**70%**	**76%**	**67%**						
MIXED_S						**63%**	**61%**	**78%**						
FRESH						**55%**	**38%**	**64%**						

Source: KCA (2025)

13.6.3 SART Tests

The SART test work is integrated with the column leach tests where the resulting pregnant solution from the columns is characterized and undergoes sulfidization, acidification, and neutralization. Columns are arranged in a closed-cycle with the solution recycling. Solution is sampled before and after the SART test work. The SART results are presented in

Table 13.38.

Table 13.38 Cerro del Gallo KCA Laboratory SART Interim Results

KCA Sample No.	Mineral Type	Free NaCN Recovery, %	SART Recovery, % Au	SART Recovery, % Ag	SART Recovery, % Cu	Days of Leach
82538	MIX OX	94%	4%	59%	47%	117
82539	MIX OX	96%	4%	76%	56%	117
82540	MIX SULF 1	92%	6%	63%	53%	117
82541	MIX SULF 1	91%	<1%	51%	71%	147
82544	SULF HG[1]	89%	1%	17%	8%	117

1: Test only had 15 days of SART performed during column leach test.

Source: KCA (2020)

The process design criteria specifies 100% recovery for copper, 95% recovery for silver, and 0% recovery for gold. Following SART, pregnant solution will be sent to an ADR plant for recovery of remaining precious metals. There is a small risk of oversaturating the carbon columns in the ADR plant with an underperforming SART plant, which may be reduced by allowing for more carbon capacity or reduced carbon strip schedules.

There exists a potential risk that some gold may be present in the SART product, possibly due to poor washing. As such, the value received for any gold in the SART product may be lower than that received from the sale of dorè. Potential gold losses from treating pregnant solution are shown to be variable and may be in the range from <0.3% to 11.0%.

13.6.3.1 SART Copper Precipitate

The SART test work included a copper precipitate analysis for smelter treatment. Solution samples from the bottle roll leach tests (1 and 2 g/L NaCN, 240-hour sample) were selected from the four material types (Oxide, Mixed Oxide, Mixed Sulfide, Sulfide) and blended together in unequal proportions that mimic the final feed distribution (8.5%, 3.1%, 36.6%, and 51.8%, respectively). Once blended into a single composite sample, the solution was processed with SART procedures to produce a copper concentrate. The concentrate was washed and analyzed. Results for the copper precipitate analysis are presented in

Table 13.39.

Table 13.39 Cerro del Gallo KCA Laboratory SART Copper Precipitate Results

Constituent	Unit	KCA Sample No. 85243 B (Precip)			Unit	Average, %
		Whole Rock	ICP Split A	ICP Split B		
Au	mg/kg		10	12	%	**0.001**
Ag	mg/kg		4770	4820	%	**0.48**
Al	%	0.16	0.05	0.05	%	0.087
As	mg/kg		62	59	%	0.006
Ba	mg/kg	400	61	69	%	0.018
Bi	mg/kg		<2	<2	%	--
Ca	%	0.66	0.28	0.26	%	**0.40**
Cd	mg/kg		1,830	1,890	%	**0.19**
Co	mg/kg		22	22	%	0.002
Cr	mg/kg		20	19	%	0.002
Cu	%		52.7	52.8	%	**53**
Fe	%	2.76	1.19	1.24	%	**1.7**
Hg	mg/kg		293.4	317.9	%	0.031
K	%	0.21	0.07	0.06	%	**0.11**
Mg	%	0.04	0.02	0.01	%	0.023
Mn	mg/kg	600	278	283	%	0.039
Mo	mg/kg		50	50	%	0.005
Na	%	0.72	0.16	0.16	%	**0.35**
Ni	mg/kg		6	5	%	0.001
Pb	mg/kg		92	96	%	0.009
$S_{(total)}$	%		16.1	--	%	**16**
$S_{(sulfide)}$	%		7.22	--	%	--
$S_{(sulfate)}$	%		8.88	--	%	--
Sb	mg/kg		<2	<2	%	--
Se	mg/kg		124	123	%	0.012
Si	%	0.66			%	**0.66**
Sr	mg/kg		11	9	%	0.001
Te	mg/kg		138	140	%	0.014
Ti	%	0.01	<0.01	<0.01	%	--
V	mg/kg		7	7	%	0.001
W	mg/kg		231	229	%	0.023
Zn	%		7.76	7.64	%	**8**
LOI[1]	1090°C					27.900
Total -						108.640

Source: KCA (2019)

13.7 Metal Recovery and Reagent Consumption Projections

All the leach test work conducted by the various laboratories, historic (pre-2015) and recent (2018-2019), were compiled and analyzed to best determine field metal heap leach recoveries. Not all the results were included in the final tabulations due to the historical test work only included three main categories of the ore types (Weathered/Oxide, Mixed, Fresh/Sulfide) while the most current test work further differentiated the Mixed category into two separate material types (Mixed Oxide, Mixed Sulfide).

For the projected field recoveries and reagent consumptions calculations there are four separate material ore type categories that were considered. The most current test work included all the ore type categories and parameters that matched the process criteria, which includes column leach tests utilizing HPGR crushed material. Historic test work that matched the process criteria (column leached HPGR crushed material) were not able to be included in this analysis due to lack of information that connect the old samples to the mineralization breakdown selected for this project. The test work material samples that were not associated with an ore type category are shown as “TBC” in the test work overall summary, presented in Table 13.40 and Table 13.41, and their results generally stayed within the range of the categorized samples.

The resulting data from the test work was compiled and sorted by test type, sample category, crush size and crush type (conventional vs HPGR). Results are presented in Table 13.40 and Table 13.41.

Table 13.40 Cerro del Gallo Leach Test Work Overall Summary – Conventional Crush

CONV. CRUSH	Recoveries					Head Grades			Calculated Head			Consumptions			
Column Leach Tests	Au	Ag	Cu	p80	# Tests	Au	Ag	Cu	Au	Ag	Cu	NaCN	Cement	Lime	Duration
Heap Leach - Coarse	%	%	%	mm	COL	g/t	g/t	g/t	g/t	g/t	g/t	kg/t	kg/t	kg/t	days
WEATHERED	75.0	32.4	12.0	12.1	1	0.72	10.0	232	0.71	8.1	205	0.3	–	0.1	61
MIXED	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_O	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_S	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
FRESH	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
TBC	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
Column Leach Tests	Au	Ag	Cu	p80	# Tests	Au	Ag	Cu	Au	Ag	Cu	NaCN	Cement	Lime	Duration
Heap Leach - Fine	%	%	%	mm	COL	g/t	g/t	g/t	g/t	g/t	g/t	kg/t	kg/t	kg/t	days
WEATHERED	75.7	51.1	12.8	6.1	2	0.49	12.0	356	0.48	8.1	342	1.6	–	1.5	102
MIXED	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_O	70.0	77.0	45.0	6.2	1	1.14	12.5	600	1.15	8.8	600	2.8	–	2.5	143
MIXED_S	56.0	62.0	58.0	6.3	1	0.83	6.5	1200	0.77	4.1	1200	3.0	–	1.8	143
FRESH	41.0	40.0	14.0	6.6	1	1.69	2.4	711	1.76	1.5	711	1.9	–	1.3	143
TBC	55.0	44.8	53.0	6.4	13	1.24	15.4	672	1.22	12.7	1048	2.6	6.0	1.8	112
Bottle Roll Tests	Au	Ag	Cu	p80	# Tests	Au	Ag	Cu	Au	Ag	Cu	NaCN	Cement	Lime	Duration
Coarse Crush	%	%	%	mm	BRT	g/t	g/t	g/t	g/t	g/t	g/t	kg/t	kg/t	kg/t	days
WEATHERED	59.3	34.6	36.7	14.2	4	1.11	15.2	1079	1.32	16.8	982	1.6	0.0	4.9	30
MIXED	37.4	34.2	34.2	14.2	3	1.07	23.4	1079	1.13	26.7	981	1.1	0.0	1.9	30
MIXED_O	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_S	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
FRESH	19.1	11.6	7.7	15.4	2	0.62	13.0	1385	0.67	18.5	1238	0.9	0.0	0.4	30
TBC	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
Bottle Roll Tests	Au	Ag	Cu	p80	# Tests	Au	Ag	Cu	Au	Ag	Cu	NaCN	Cement	Lime	Duration
Fine Crush	%	%	%	mm	BRT	g/t	g/t	g/t	g/t	g/t	g/t	kg/t	kg/t	kg/t	days
WEATHERED	71.6	54.1	35.8	5.1	6	0.88	16.6	1079	0.97	13.5	952	1.6	0.0	4.7	34
MIXED	51.7	36.9	47.9	4.0	47	0.71	15.7	1079	0.67	29.0	424	2.6	0.0	2.5	39
MIXED_O	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_S	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
FRESH	35.2	16.6	25.3	4.4	7	0.76	9.4	1385	0.84	15.0	791	3.0	0.0	2.9	41
TBC	51.6	37.6	34.3	5.6	24	1.24	17.0	870	1.25	16.5	909	1.5	0.0	2.4	4
Bottle Roll Tests	Au	Ag	Cu	p80	# Tests	Au	Ag	Cu	Au	Ag	Cu	NaCN	Cement	Lime	Duration
Mill	%	%	%	mm	BRT	g/t	g/t	g/t	g/t	g/t	g/t	kg/t	kg/t	kg/t	days
WEATHERED	91.0	70.8	32.9	0.1	6	0.82	11.9	583	0.86	12.4	0	1.3	0.0	2.6	4
MIXED	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_O	91.8	80.8	69.8	0.1	6	0.94	16.4	531	0.99	16.9	0	1.7	0.0	1.7	4
MIXED_S	85.5	64.7	67.7	0.1	15	1.10	15.3	1173	1.13	16.0	0	3.0	0.0	1.7	4
FRESH	87.2	61.3	45.6	0.1	6	1.80	4.1	721	1.85	4.3	0	1.5	0.0	1.1	4
TBC	81.2	64.5	20.4	0.2	1	1.08	11.7	628	1.38	12.4	716	0.7	0.0	2.2	3
Cyanide Shake Tests	Au	Ag	Cu	p80	# Tests	Au	Ag	Cu	Au	Ag	Cu	NaCN	Cement	Lime	Duration
	%	%	%	mm	LEACH	g/t	g/t	g/t	g/t	g/t	g/t	kg/t	kg/t	kg/t	days
WEATHERED	82.7	82.8	10.9	0.1	34	0.93	12.0	536	0.00	0.0	0	0.0	0.0	0.0	1
MIXED	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_O	85.0	84.7	13.7	0.1	28	0.98	16.2	542	0.00	16.2	542	0.0	0.0	0.0	1
MIXED_S	77.8	69.0	7.1	0.1	71	1.05	14.8	1209	0.00	14.8	1209	0.0	0.0	0.0	1
FRESH	82.9	60.6	9.0	0.1	27	1.81	5.0	727	0.00	5.0	727	0.0	0.0	0.0	1
TBC	80.2	38.7	31.8	0.1	37	1.24	18.9	1003	1.19	18.9	1003	2.0	0.0	1.5	2

Source: KCA (2019)

Table 13.41 Cerro del Gallo Leach Test Work Overall Summary – HPGR Crush

HPGR CRUSH	Recoveries					Head Grades			Calculated Head			Consumptions			
Column Leach Tests	**Au**	**Ag**	**Cu**	**p80**	**# Tests**	**Au**	**Ag**	**Cu**	**Au**	**Ag**	**Cu**	**NaCN**	**Cement**	**Lime**	**Duration**
Heap Leach - Fine	**%**	**%**	**%**	**mm**	**COL**	**g/t**	**g/t**	**g/t**	**g/t**	**g/t**	**g/t**	**kg/t**	**kg/t**	**kg/t**	**days**
WEATHERED	76.0	63.0	22.0	3.9	2	0.82	11.9	583	0.90	10.1	–	2.3	2.1	2.9	135
MIXED	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_O	71.5	81.5	46.0	4.3	2	0.94	16.4	531	1.02	17.4	–	2.7	–	2.3	135
MIXED_S	61.4	62.2	58.6	5.0	5	1.10	15.3	1173	1.06	11.7	–	3.3	2.0	2.2	128
FRESH	59.5	43.0	34.0	5.5	2	1.80	4.1	721	1.68	4.5	–	2.4	–	1.3	128
TBC	65.5	54.8	44.2	4.9	6	1.20	9.1	649	1.21	11.5	618	1.2	–	1.0	115
Bottle Roll Tests	**Au**	**Ag**	**Cu**	**p80**	**# Tests**	**Au**	**Ag**	**Cu**	**Au**	**Ag**	**Cu**	**NaCN**	**Cement**	**Lime**	**Duration**
Fine Crush	**%**	**%**	**%**	**mm**	**BRT**	**g/t**	**g/t**	**g/t**	**g/t**	**g/t**	**g/t**	**kg/t**	**kg/t**	**kg/t**	**days**
WEATHERED	71.6	0.0	47.6	3.8	1	0.41	30.6	0	0.43	0.0	0	2.5	0.0	5.0	42
MIXED	49.9	0.0	45.0	3.9	12	0.78	15.1	0	0.84	0.0	0	2.5	0.0	3.2	42
MIXED_O	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
MIXED_S	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
FRESH	–	–	–	–	0	–	–	–	–	–	–	–	–	–	–
TBC	59.5	54.7	49.7	4.9	6	1.20	9.1	649	1.31	13.1	638	1.2	0.0	2.8	2
Cyanide Shake Tests	**Au**	**Ag**	**Cu**	**p80**	**# Tests**	**Au**	**Ag**	**Cu**	**Au**	**Ag**	**Cu**	**NaCN**	**Cement**	**Lime**	**Duration**
	%	**%**	**%**	**mm**	**LEACH**	**g/t**	**g/t**	**g/t**	**g/t**	**g/t**	**g/t**	**kg/t**	**kg/t**	**kg/t**	**days**
TBC	89.3	81.5	57.1	0.1	1	1.21	9.7	640	1.31	12.2	755	1.5	0.0	1.6	2

Source: KCA (2019)

Heap leach field projections for gold, silver and copper recovery were estimated based on a two, three and five percentage point deduction, respectively from laboratory results. The projected field recoveries assume a well-managed heap leach operation and that agglomeration was completed correctly. Reagent consumptions for sodium cyanide and cement were also compiled and projected for the field; for field projections the sodium cyanide consumption was calculated in METSIM and cement was established from the compacted permeability test work. Lime was not expected to be utilized due to the high levels of cement and its neutralization potential. A summary table showing the overall recoveries for each sample category is presented in Table 13.42, and the projected reagent consumptions in Table 13.43.

Table 13.42 Cerro del Gallo Heap Leach Recovery Projections

Ore Type (HPGR)	Feed Distribution LOM Average[1]	Column Test Recoveries		Projected Field Recoveries[2]		
		Au, %	Ag, %	Au, %	Ag, %	Cu%
Weathered (Oxide)	9.2%	76	63	74	60	17
Mixed Oxide	5.8%	72	82	68	73	62
Mixed Sulfide	38.0%	61	62	61	58	73
Fresh (Sulfide)	47.0%	60	43	53	35	59
Life of Mine (LOM) Average				59	49	62
[1] Based on HRC mine plan Oct 2025						
[2] Gold Discount, % 2						
[2] Silver Discount, % 3						
[2] Copper Discount, % 5						

Source: KCA (2025)

Table 13.43 Cerro del Gallo Heap Leach Projected Reagent Consumptions

Ore Type (HPGR)	Feed Distribution LOM Average[1]	Column Test Reagents, kg/t			Projected Field Reagents, kg/t		
		NaCN	Cement[2]	Lime	NaCN[3]	Cement	Lime
Weathered (Oxide)	9.2%	2.3	20.0	–	0.87	10.0	–
Mixed Oxide	5.8%	2.7	10.0	–	0.34	10.0	–
Mixed Sulfide	38.0%	3.3	10.0	–	0.87	10.0	–
Fresh (Sulfide)	47.0%	2.4	7.5	–	0.60	10.0	–
Life of Mine (LOM) Average					0.71	10	–
[1] Based on HRC mine plan Oct 2025							
[2] Based on compacted permeability test work, 80 m							
[3] Based on SART Column Tests and METSIM simulation							

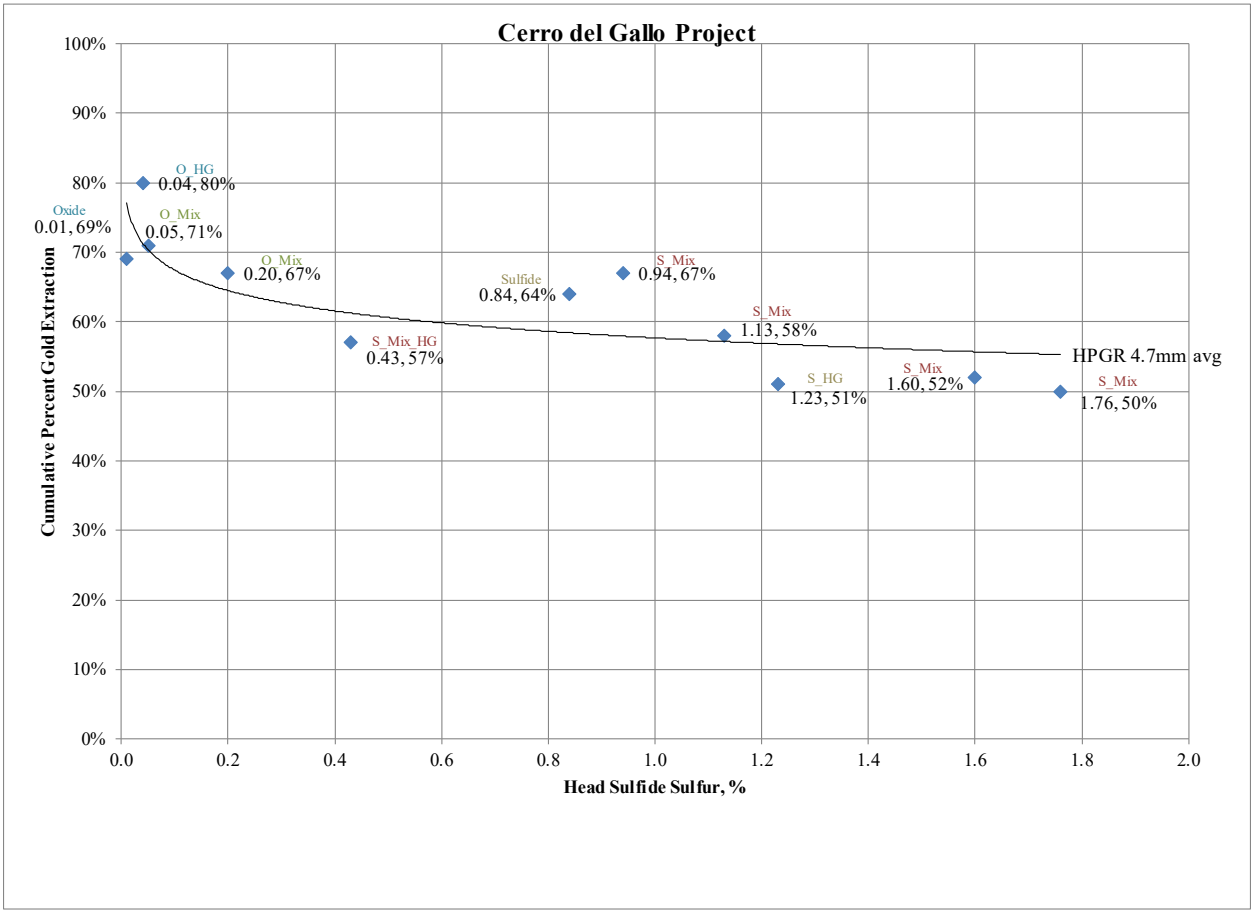
Source: KCA (2025)

13.7.1 Sulfur Overview

The sulfide sulfur was analyzed and compared to the results of the latest column leach test work. Sample categories (Weathered/Oxide, Mixed Oxide, Mixed Sulfide, Fresh/Sulfide) were compared with sulfide sulfur content and metal extraction. The general trends indicate higher gold and silver metal extraction as the sulfide sulfur content decreases. More importantly, the four separate sample categories are less distinct where only a clear difference between oxide samples and sulfide samples is noticed.

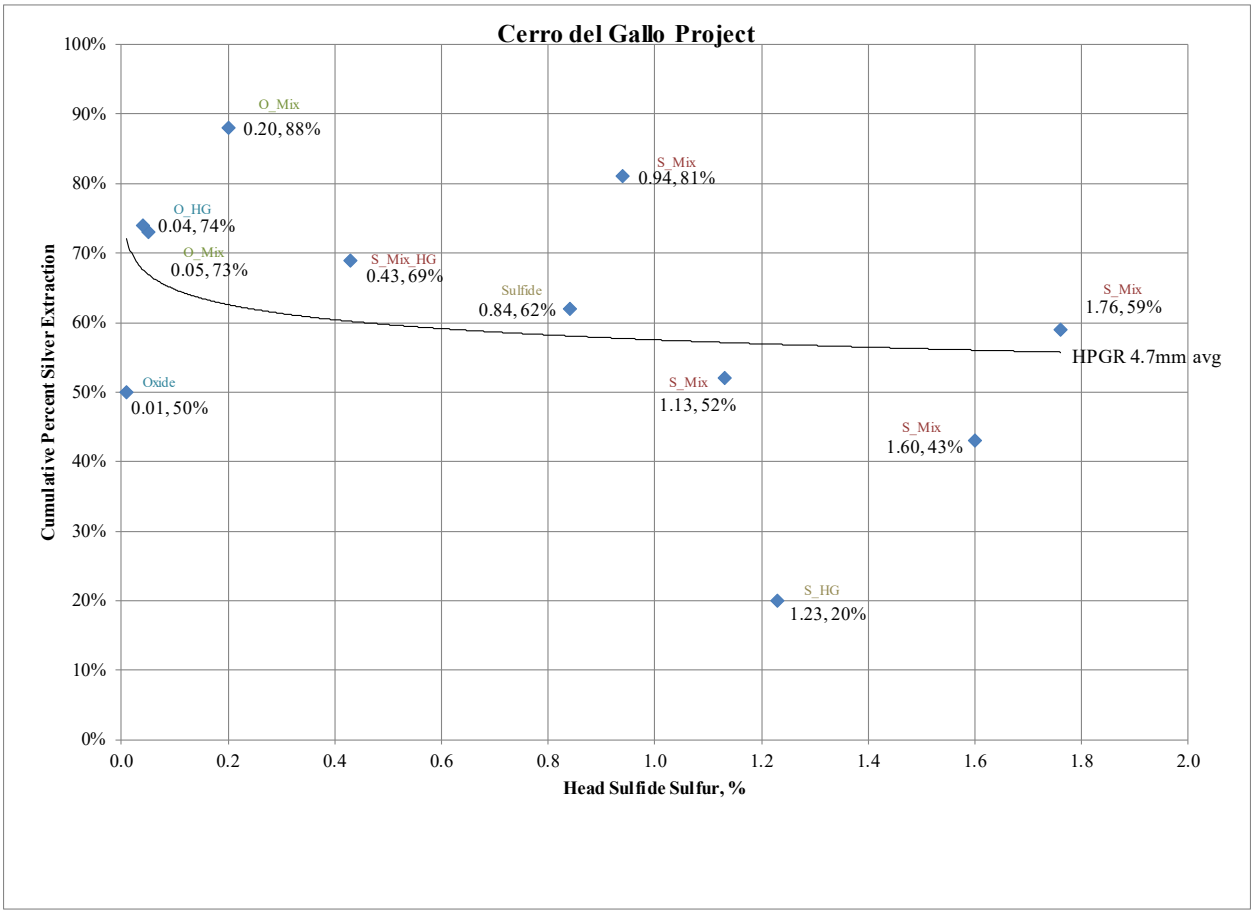
The sulfide content graphs are illustrated in Figure 13.5 through Figure 13.8.

Figure 13.5: Cerro del Gallo Sulfide Content versus Recovery (HPGR) – Gold



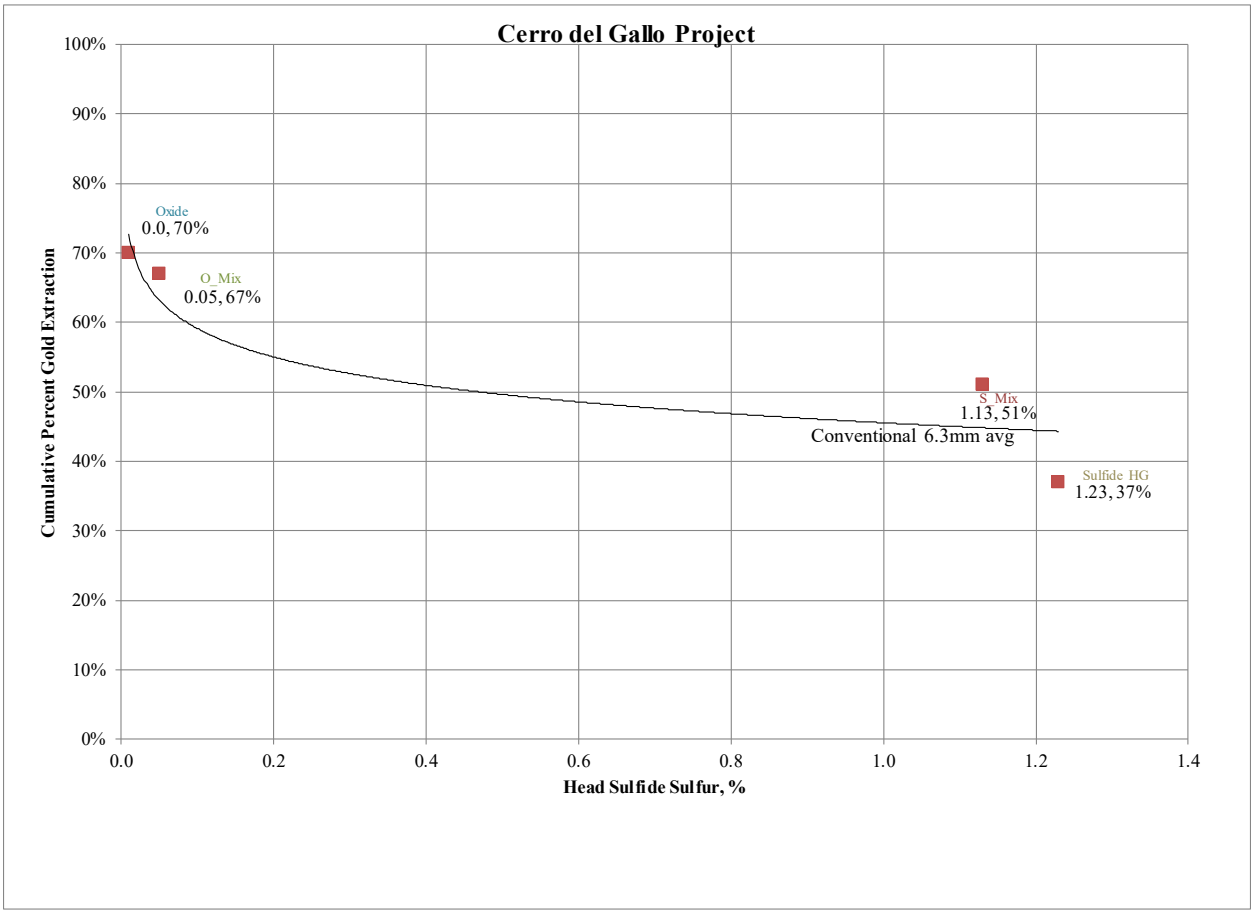
Source: KCA (2019)

Figure 13.6: Cerro del Gallo Sulfide Content versus Recovery (HPGR) – Silver



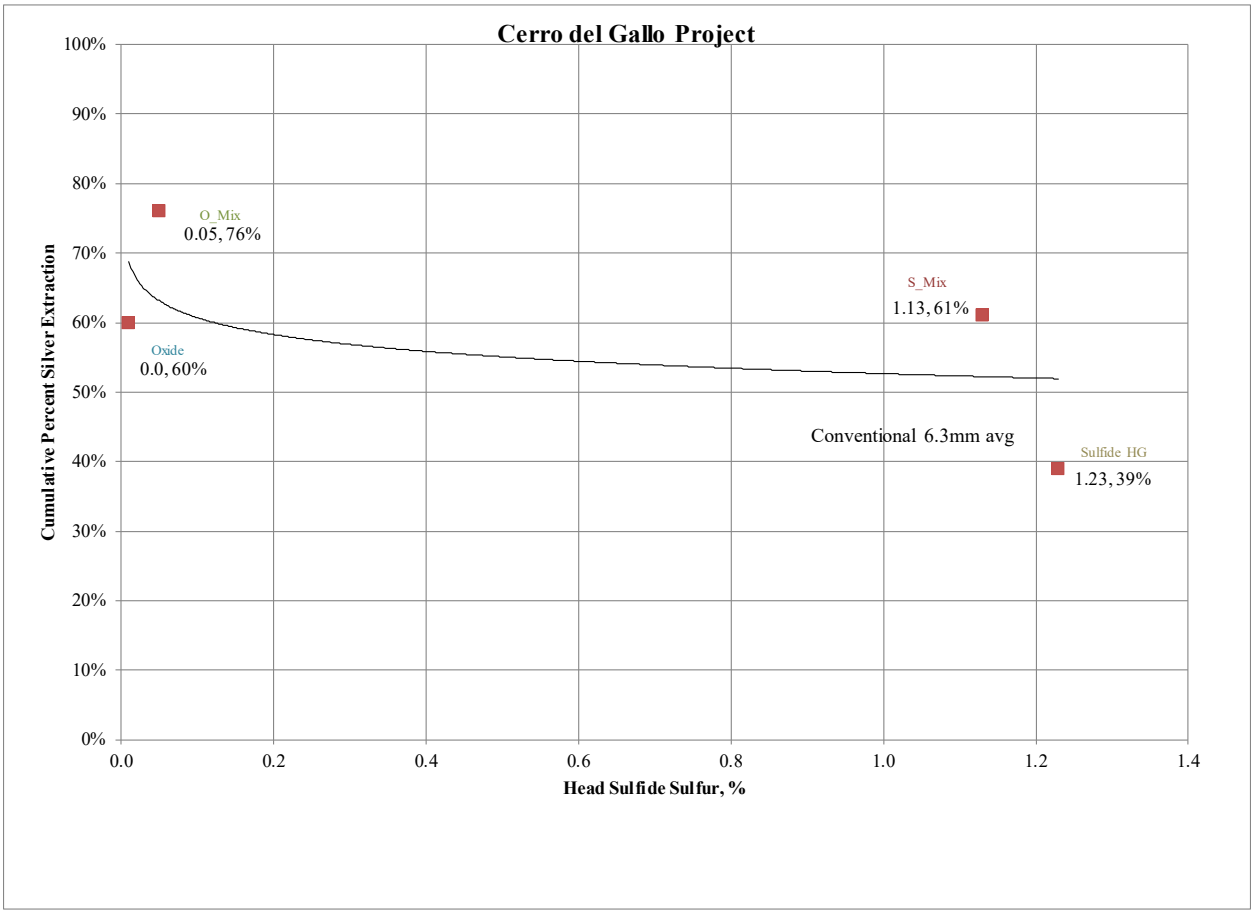
Source: KCA (2019)

Figure 13.7: Cerro del Gallo Sulfide Content versus Recovery (Conventional) – Gold



Source: KCA (2019)

Figure 13.8: Cerro del Gallo Sulfide Content versus Recovery (Conventional) – Silver

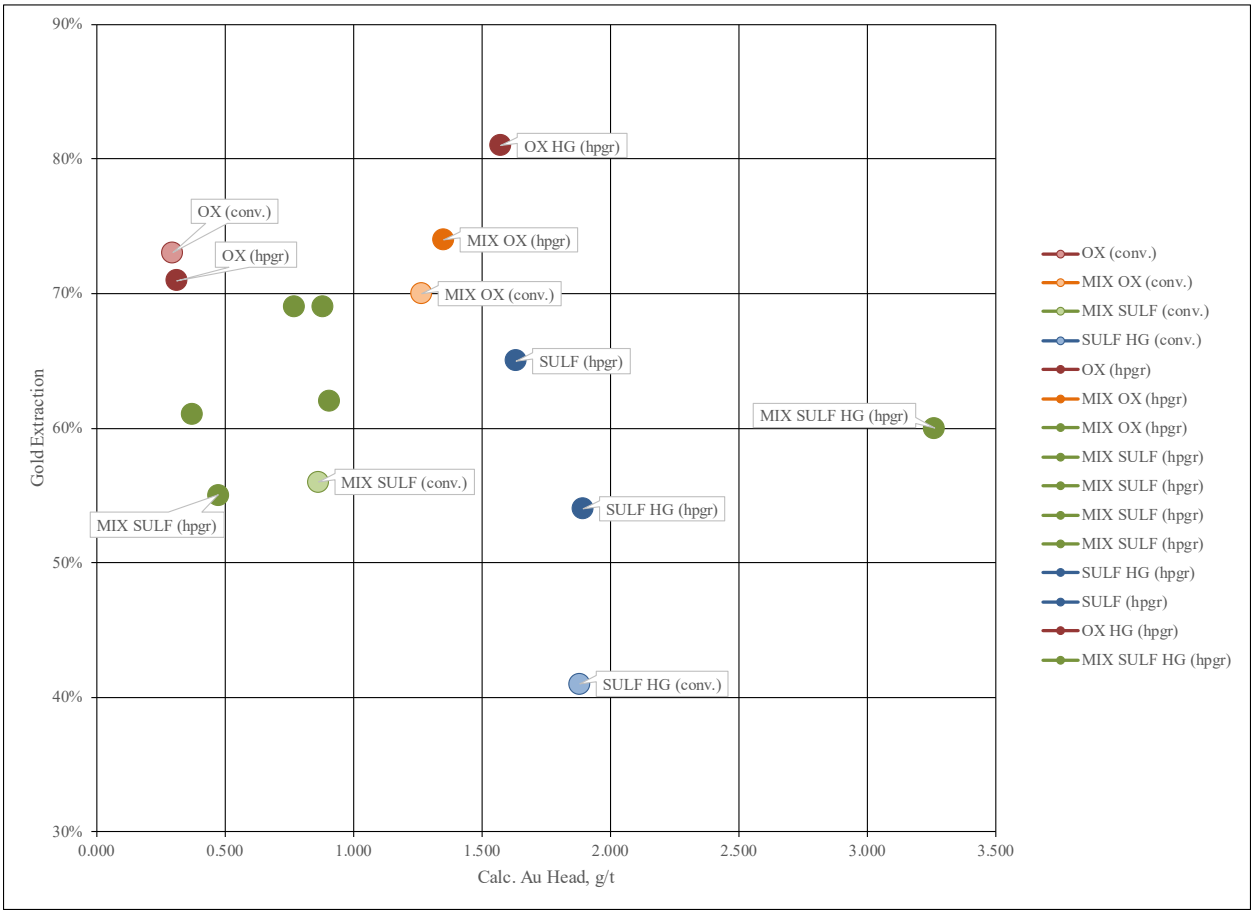


Source: KCA (2019)

13.7.2 Head Grades versus Metal Extraction

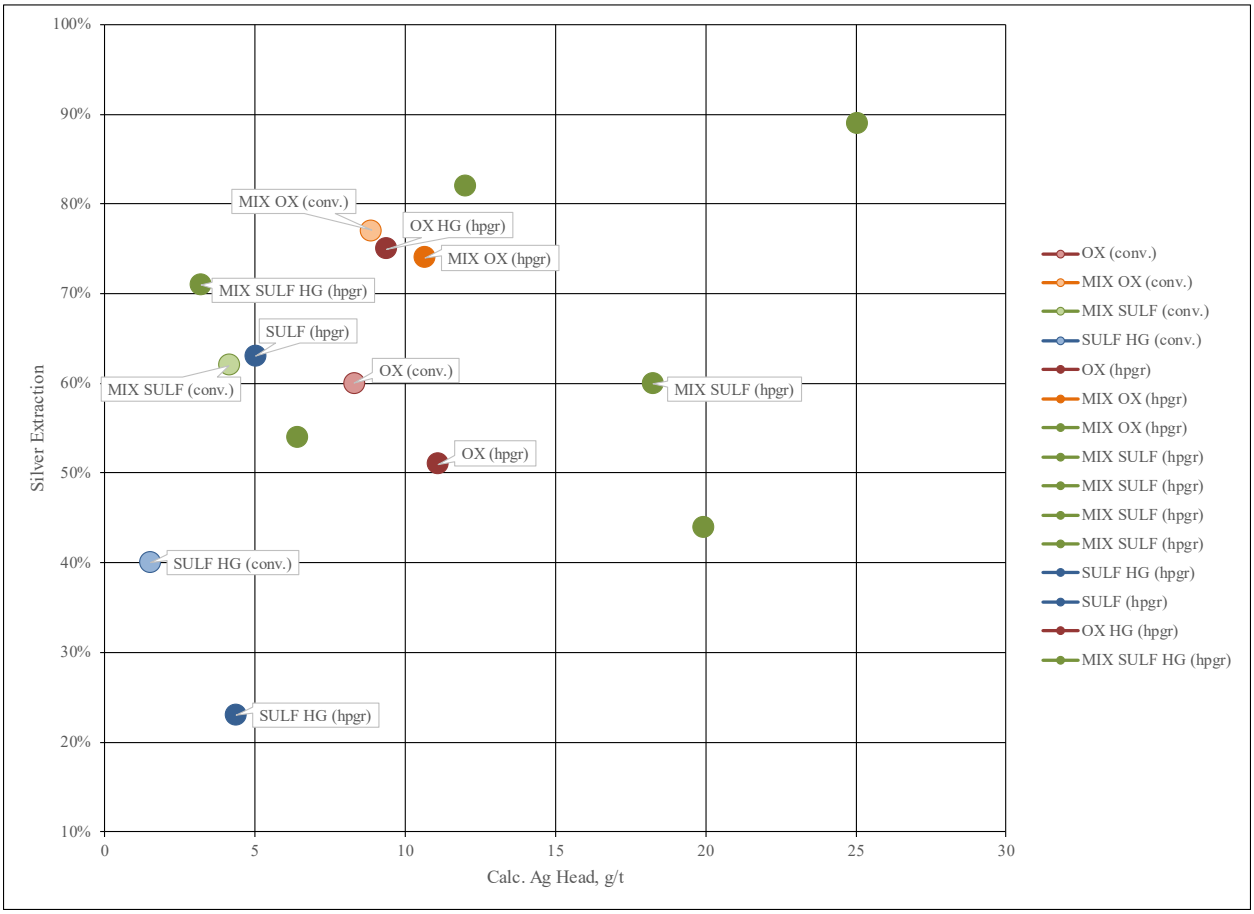
Head grades of the sample categories (Weathered/Oxide, Mixed Oxide, Mixed Sulfide, Fresh/Sulfide) were compared with metal extraction from the latest column leach test work. The resulting graphs for gold, silver, and copper are presented in Figure 13.9 through Figure 13.11. They include both conventionally and HPGR crushed material. No measurable trends were observed for gold, silver and copper.

Figure 13.9: Cerro del Gallo Head Grade versus Recovery – Gold



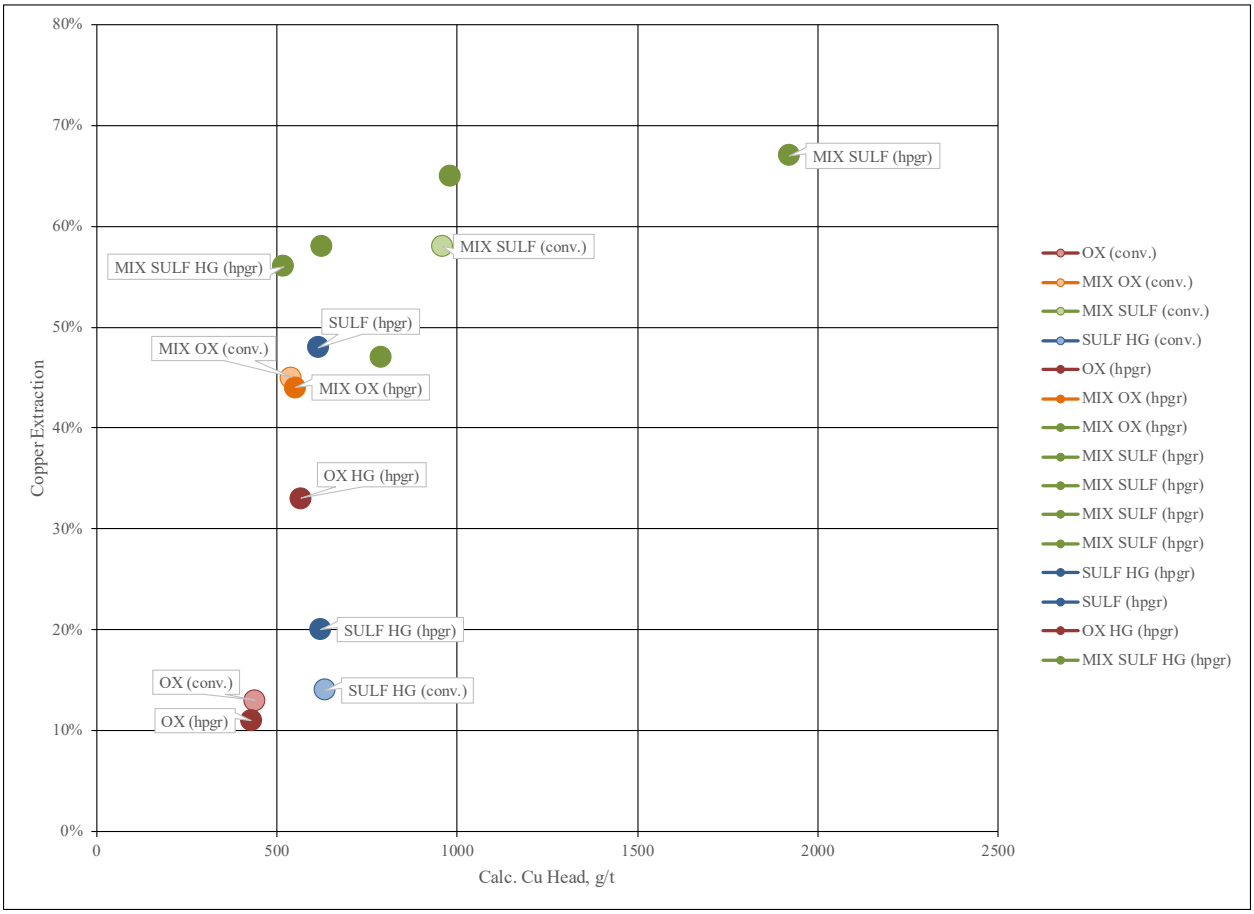
Source: KCA (2019)

Figure 13.10: Cerro del Gallo Head Grade versus Recovery – Silver



Source: KCA (2019)

Figure 13.11: Cerro del Gallo Head Grade versus Recovery – Copper



Source: KCA (2019)

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

Construction of the Cerro del Gallo (CdG) 2025 mineral resource model included the construction of a geology model and an estimation model for gold, silver and copper.

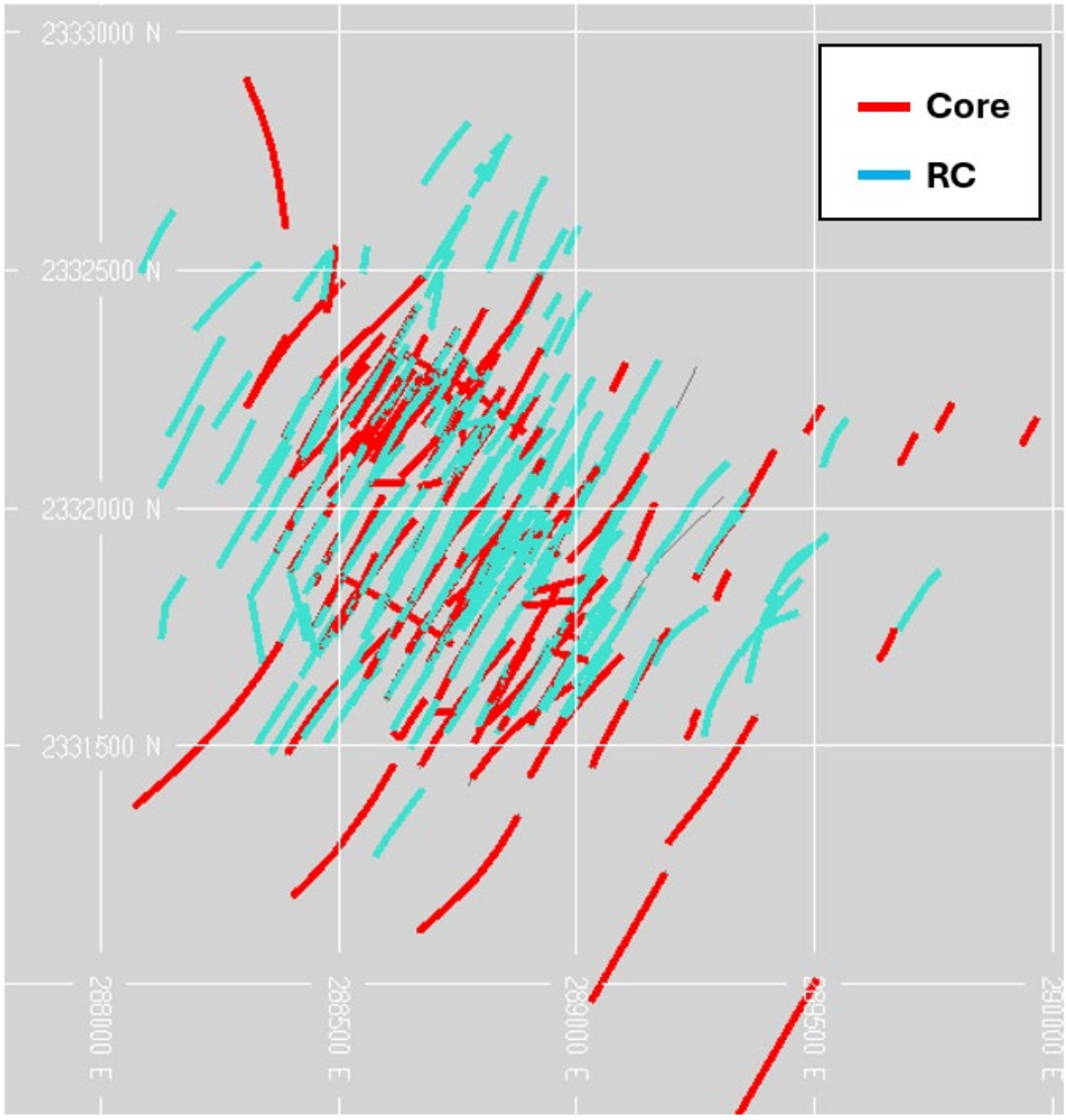
Three geological models were constructed in Leapfrog including lithology domains, oxidation domains and grade shell domains for gold, silver and copper.

Mineral Resources estimates were completed for gold, silver and copper using Vulcan software.

14.2 Drill Hole Data

Drill hole data for CdG district was provided to MTS in an Access database. The CdG Access database was audited by MTS (discussed in Sections 10.0 to 12.0). The audited drill hole data were provided as csv files and were imported into Leapfrog and Vulcan software. The csv files included tables for collar locations, downhole surveys, assays, lithology, oxidation, structural features and drill hole type. Only drillhole data for the CdG area was used for the resource model. Figure 14.1 is a plan map of the CdG area showing core and RC drill holes.

Figure 14.1: Drill Holes Used for CdG Resource Model



Source: MTS 2025

14.3 Geology Model

The geology modeling for the CdG resource model was completed in Leapfrog software. The geological models include lithology domains, oxidation state domains and grade shell domains for gold, silver and copper.

14.3.1 Lithology Domains

The CdG database field for lithology included codes for lithology, alteration and structural features. In developing the lithology model, the database lithology field was divided into fields for Model Lithology, Model Alteration and Model Structural features.

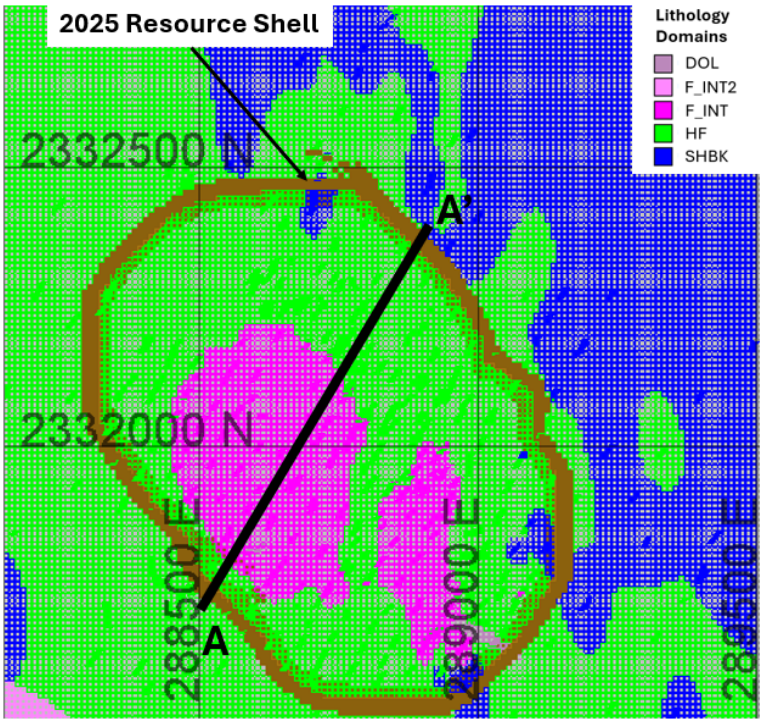
The final lithological model field included domains intrusive rocks (F_INT and F_INT2), hornfels + tuff domain (HF), a black shale domain (SHBK) and a dolerite domain (DOL). Wireframes were constructed in Leapfrog and imported into Vulcan software to code the block model. Table 14.1 summarizes the five lithology domains and domain codes used for the 2025 CdG lithology domains.

Figure 14.2 and Figure 14.3 show a plan section and cross-section of the lithology domains. The cross-section A-A' is shown on Figure 14.2. Note the F_INT2 domain is in the extreme southwest of the model area (Figure 14.3). This F_INT2 intrusive is separate from the main intrusive (F_INT) with only limited drilling intercepts (3 drill holes).

Table 14.1: Lithology Domains

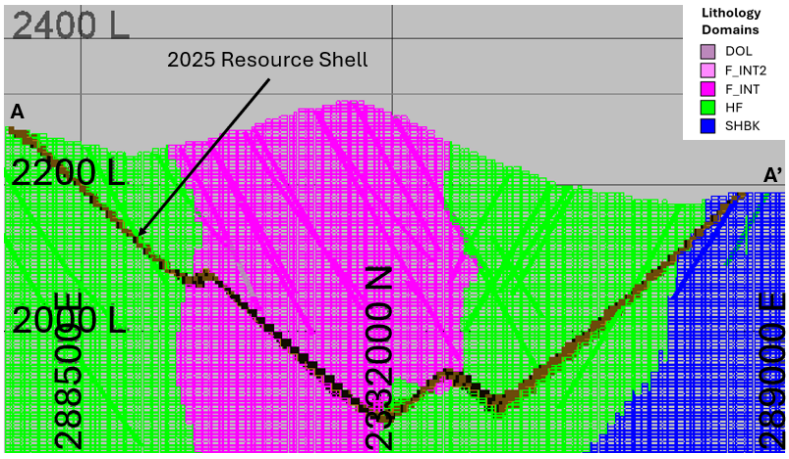
Lithology Domain	Code	Integer Code
Dolerite	DOL	535
Intrusive 2	F_INT2	531
Intrusive	F_INT	530
Horn fels	HF	514
Black Shale	SHBK	510

Figure 14.2: 2100 Level Showing Lithology Domains



Source: MTS 2025

Figure 14.3: Cross-Section A-A" Showing Lithology Domains



Source: MTS 2025

14.3.2 Oxidation Model

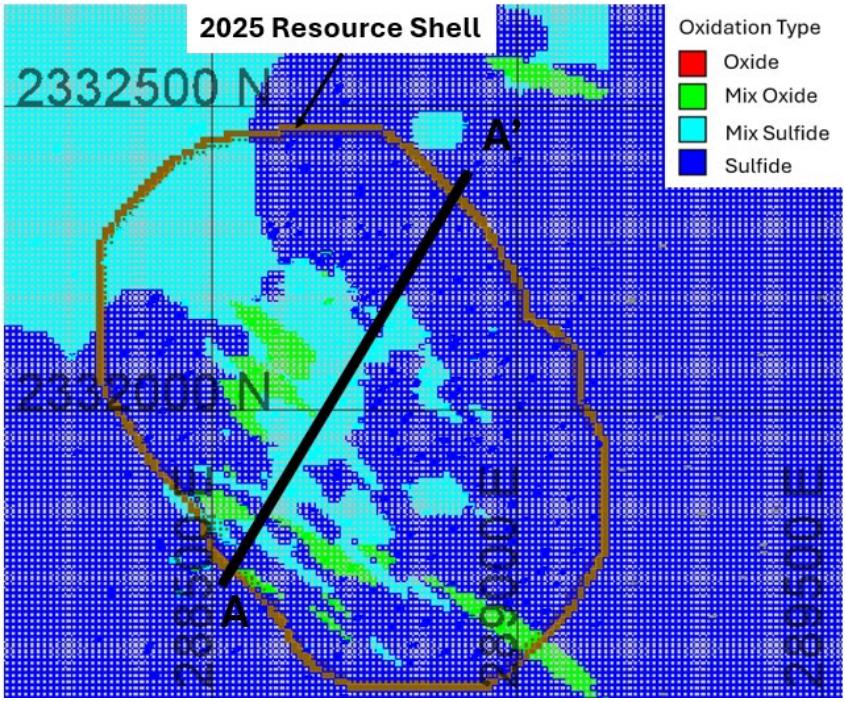
The CdG oxidation model included four material types (Table 14.2). The material types were modelled in Leapfrog. The wireframes were imported into Vulcan and used to code the block model. Figure 14.4 and **Error! Reference source not found.** show a plan

section and cross-section through the oxidation model respectively. Cross-section A-A' is shown on Figure 14.4.

Table 14.2: Oxidation Domains

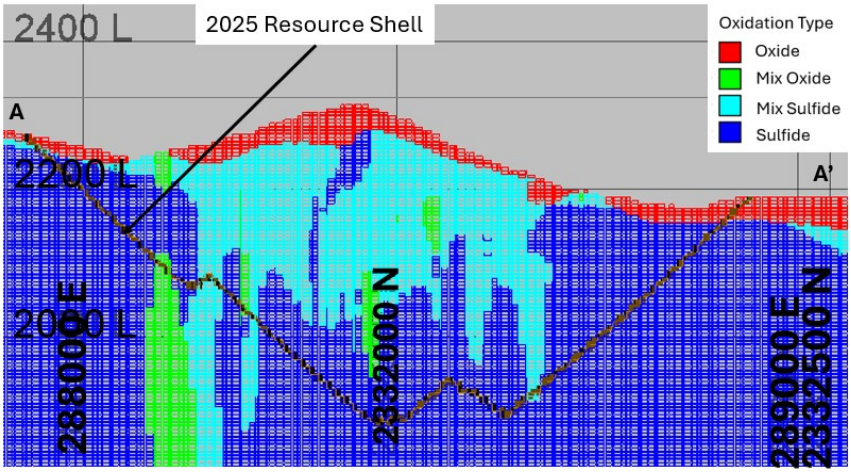
Oxidation Domain	Code
Oxide	803
Mixed Oxide	802
Mixed Sulfide	801
Sulfide	800

Figure 14.4: 2100 Level Showing Oxidation Domains



Source: MTS 2025

Figure 14.5: Cross-Section A-A" Showing Oxidation Domains



Source: MTS 2025

14.3.3 Grade Shell Indicator Model

Grade shell domain models for gold, silver and copper were constructed in Leapfrog using Leapfrogs RBF indicator function. The grade shell domains were used to constrain higher grades areas during grade estimation. The purpose of the grade shells was to constrain higher-grade mineralization from encroaching into lower-grade regions.

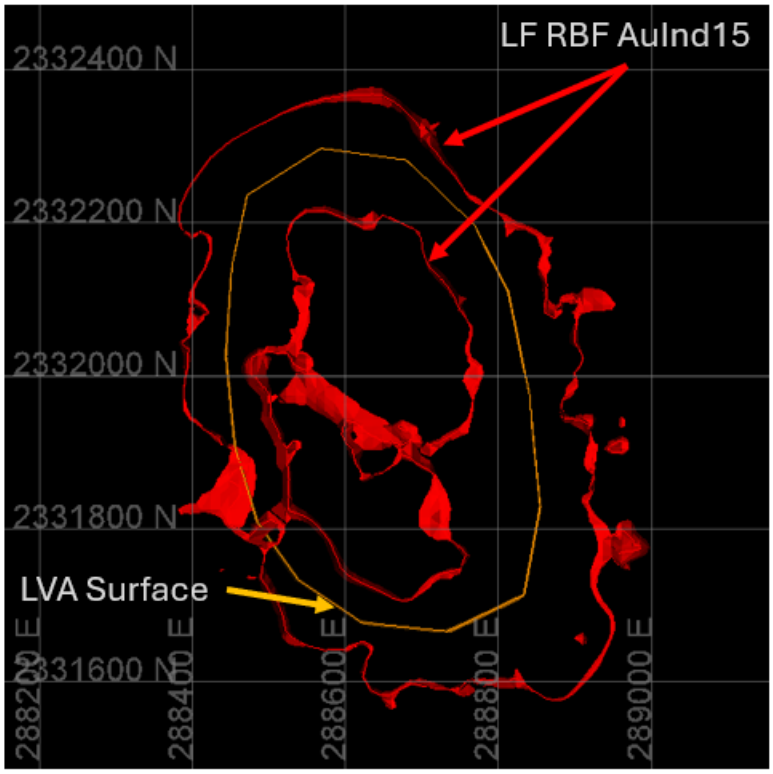
Table 14.3 summarizes the grade threshold used for gold, silver and copper. The indicator threshold for the RBF function was 0.50. Each block in the block model was coded for LG or HG. The HG and LG domains were estimated independently.

Table 14.3: Leapfrog RBF Indicator Function

Element	Grade Threshold	Indicator Threshold	LG Indicator	HG Indicator
Gold	0.15 ppm	0.50	0	1
Silver	15 ppm	0.50	0	1
Copper	200 ppm	0.50	0	1

The HG gold domain (Figure 14.6) was used to assist in the construction of the LVA surface discussed in Section 14.7.

Figure 14.6: Plan View of LVA Surface and Gold RBF



Source: MTS 2025

14.4 Exploratory Data Analysis

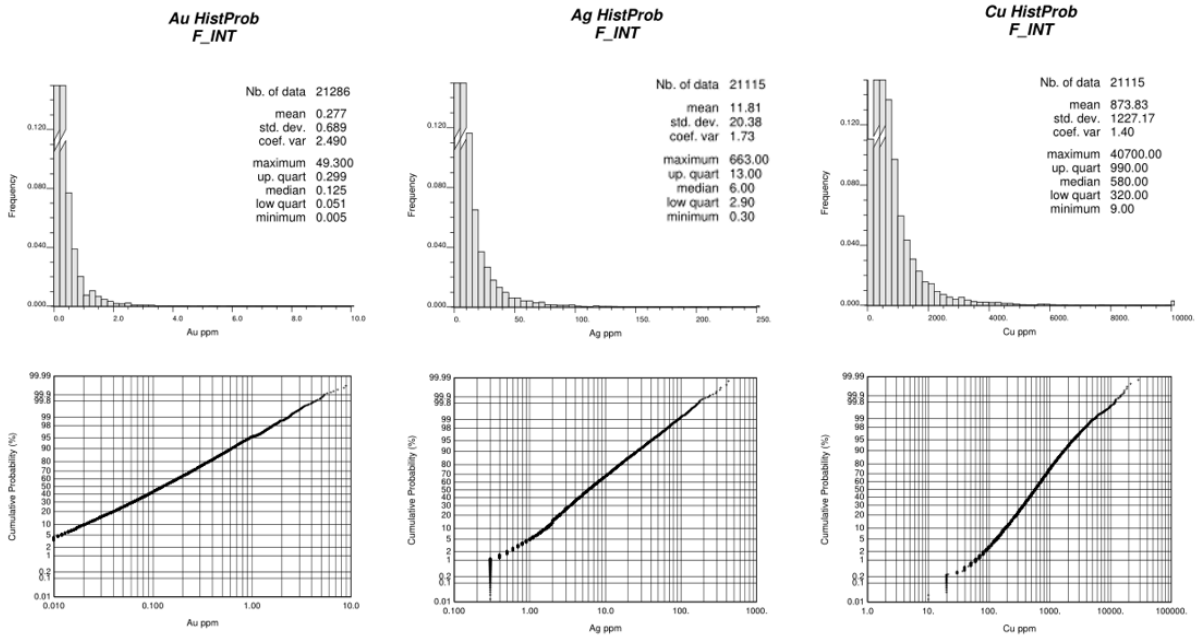
Exploratory Data Analysis (EDA) was completed on assays and composites using box plots, histograms, cumulative probability plots and contact profiles.

14.4.1 Assays

Box plots, histograms and probability plots of assay data were constructed for CdG to investigate domaining and grade outliers.

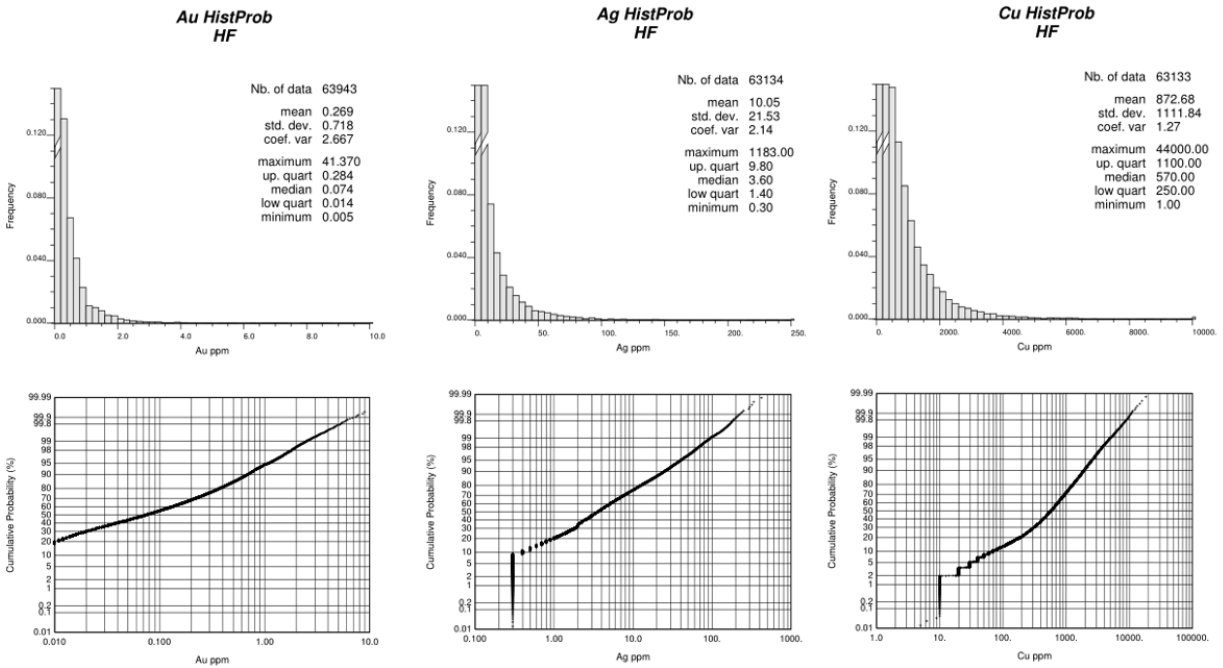
Cumulative probability plots were generated from the gold, silver and copper assays for the mineralized and non-mineralized populations sub-divided by each lithology domain. Figure 14.7 to Figure 14.9 shows histogram and probability plots of assays for gold, silver and copper. After reviewing probability plots, the QP chose not to implement grade capping at the assay level.

Figure 14.7: Histograms and Probability Plots for Gold, Silver and Copper for F_INT Domain



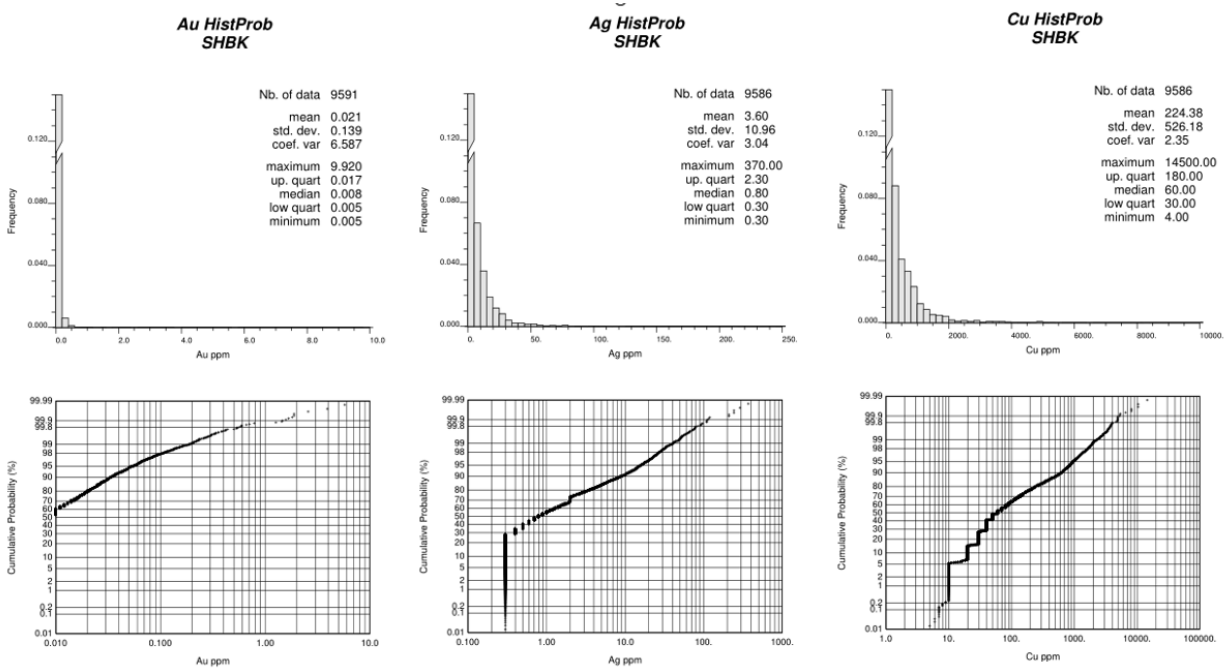
Source: MTS 2025

Figure 14.8: Histograms and Probability Plots for Gold, Silver and Copper for the HF Domain



Source: MTS 2025

Figure 14.9: Histograms and Probability Plots for Gold, Silver and Copper for the SHBK Domain



Source: MTS 2025

14.4.2 Composites

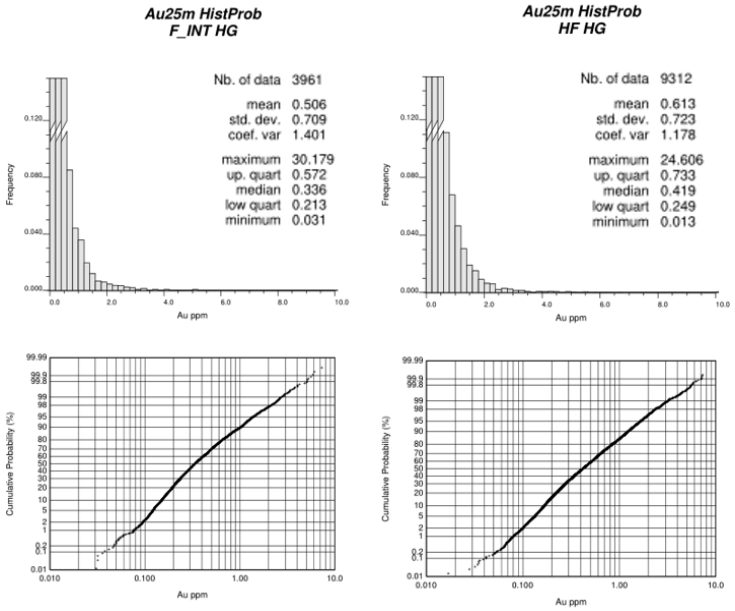
The CdG assay data were composited into 2.5 m and 5 m fixed length composites. The lithology domains, oxide domains and grade shell domains were back tagged into the composite files using the domain wireframes.

The 2.5 m composites were used for grade estimation. The 5 m composites were used for estimating a Nearest Neighbor (NN) model used for model validation.

EDA was completed on 2.5 m composites using box plots, histograms, probability plots and contact profiles. Figure 14.10 to Figure 14.12 show histogram and probability plots for gold, silver and copper within the F_INT and HG domains.

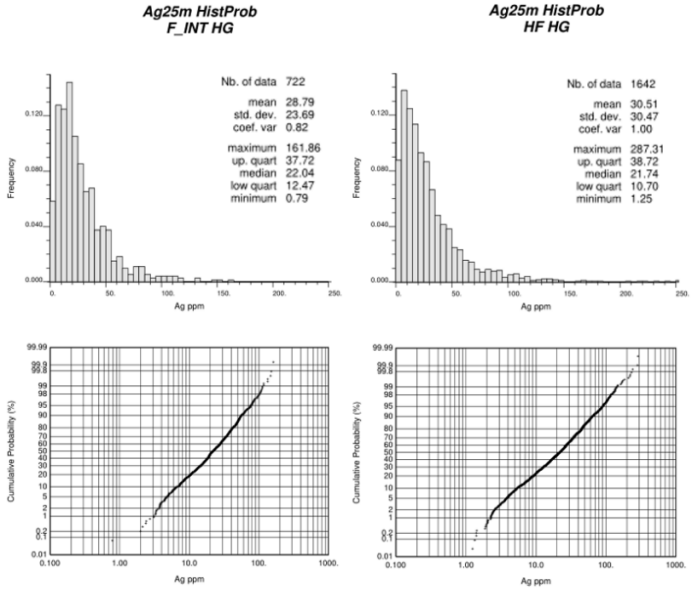
Table 14.4 to Table 14.6 summarize the 2.5 m composite statistics by lithology and grade domains.

Figure 14.10: Gold Histogram and Probability Plot for 2.5 m Composites (F_INT and HF) HG Domain



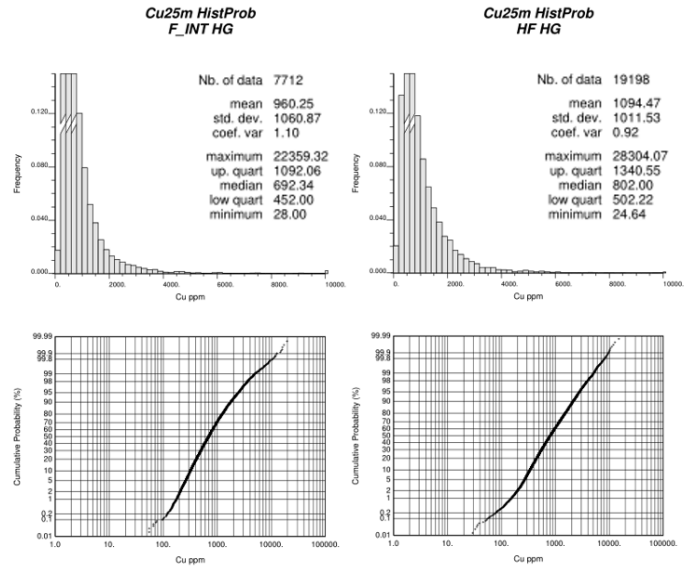
Source: MTS 2025

Figure 14.11: Silver Histogram and Probability Plot for 2.5 m Composites (F_INT and HF) HG Domain



Source: MTS 2025

Figure 14.12: Copper Histogram and Probability Plot for 2.5 m Composites (F_INT and HF) , HG Domain



Source: MTS 2025

Table 14.4: Summary of 2.5 m composites for Gold

Domain	Number	Min (g/t)	Max (g/t)	Mean (g/t)	CV
F_INT (HG)	3961	0.031	1.272	0.506	1.401
F_INT (LG)	5134	0.005	0.75	0.105	1.633
HF (HG)	9312	0.613	24.606	0.613	1.178
HF (LG)	18095	0.068	14.327	0.068	3.364
SHBK (HG)	4	0.151	0.273	0.190	0.264
SHBK (LG)	3197	0.005	3.010	0.023	4.183

Table 14.5: Summary of 2.5 m composites for Silver

Domain	Number	Min (g/t)	Max (g/t)	Mean (g/t)	CV
F_INT (HG)	722	0.79	161.86	28.79	0.82
F_INT (LG)	8300	0.30	405.90	10.34	1.37
HG (HG)	1642	1.25	287.31	30.51	1.00
HF (LG)	23468	0.30	797.34	8.65	1.78
SHBK (HG)	32	4.62	161.97	24.45	1.12
SHBK (LG)	3170	0.030	156.28	3.56	2.19

Table 14.6: Summary of 2.5 m composites for Copper

Vein/ Structure	Number	Min(%t)	Max(%t)	Mean (g/t)	CV
F_INT (HG)	7712	28	22359	960	1.10
F_INT (LG)	1310	16	2166	270	0.87
HG (HG)	19198	25	28304	1094	0.92
HF (LG)	5912	4	8676	189	1.53
SHBK (HG)	522	75	7051	947	0.79
SHBK (LG)	2850	10	2466	105	1.41

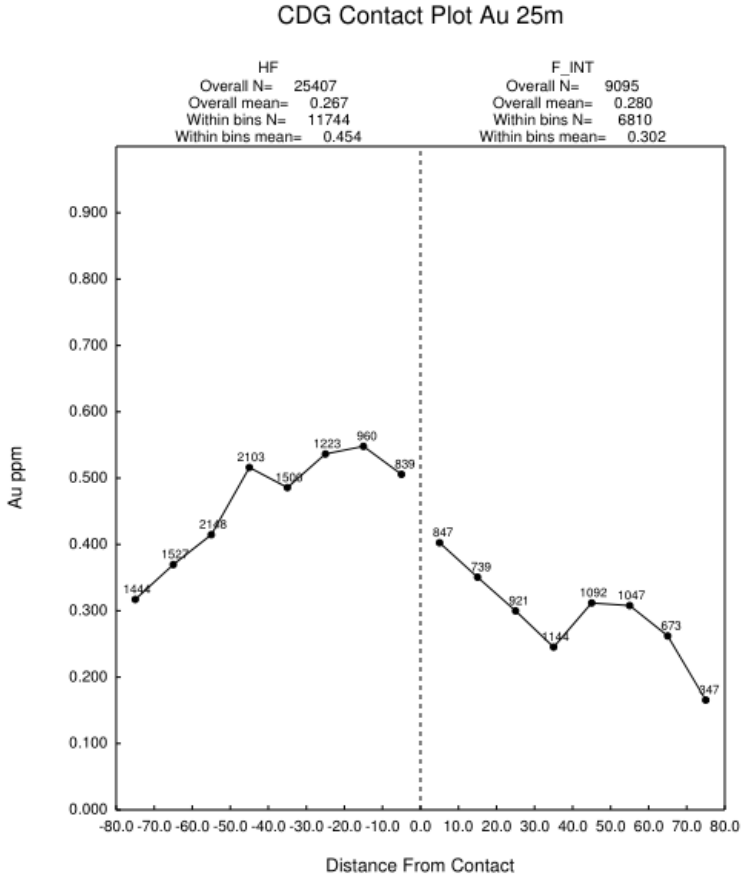
14.4.2.1 Capping of Outliers

A top cutting was not implemented on assays or composites. An outlier restriction was implemented during grade estimation (see Section 14-9).

14.4.2.2 Contact Profiles

Contact profiles were constructed between the F_INT, HF and SHBK domains. Generally, the contact profiles suggested hard contacts between the domains. However, model validation suggested a firm contact between the F_INT and HF domains.

Figure 14.13: Gold Contact Profile for HF to F_INT



Source: MTS 2025

14.5 Grade Restrictions, Top-Cuts and Outlier Evaluation

Capping limits were investigated at the 2.5 m composite level using probability and histogram plots. The plots for the HG and LG portions of the 2.5 m composited data showed limited outliers.

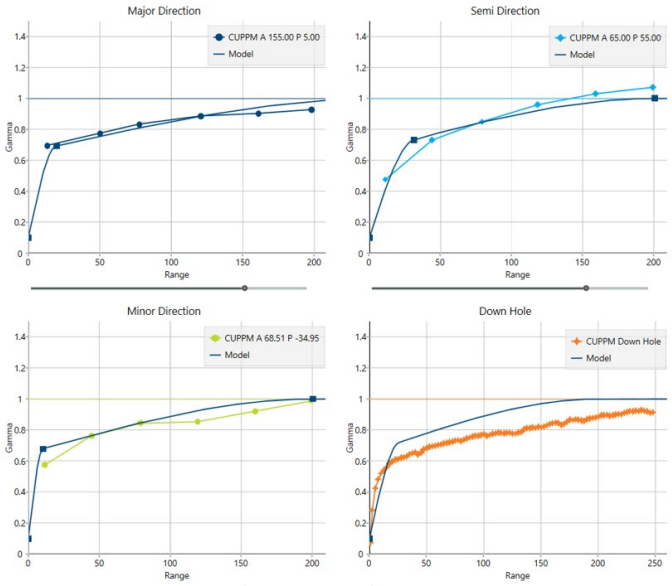
Because of limited outliers, the QP chose to implement an outlier restriction strategy during grade estimation (Section 14.9).

14.6 Variography

Grade Variograms (correlograms) were generated for gold, silver and copper using 2.5 m composites. The variogram models were generated using Vulcan data analysis tools for each element to be estimated (Au, Ag, Cu). A single variogram was used for each element. During grade estimation, the orientation of the variogram was adjusted for each block using the LVAB, LVAP and LVAD coding (Section 14.7.1).

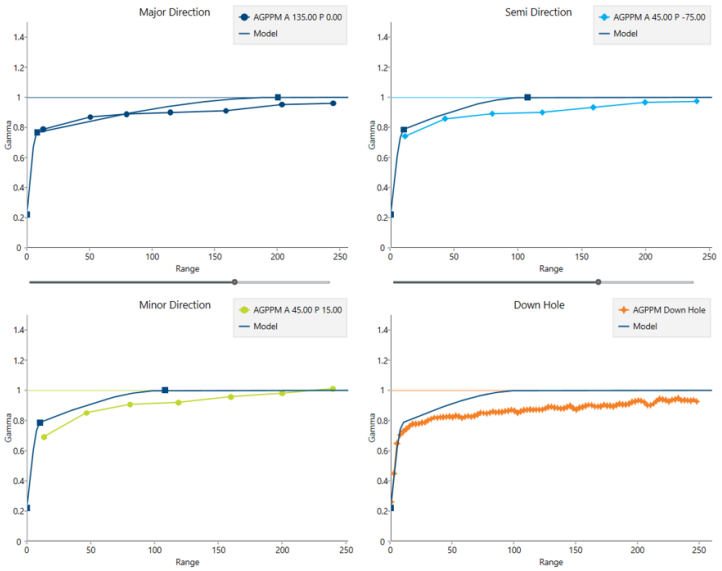
Figure 14.14 to Figure 14.16 show the variograms for gold, silver and copper respectively. Table 14.7 summarizes the variogram models.

Figure 14.14: Gold Correlogram



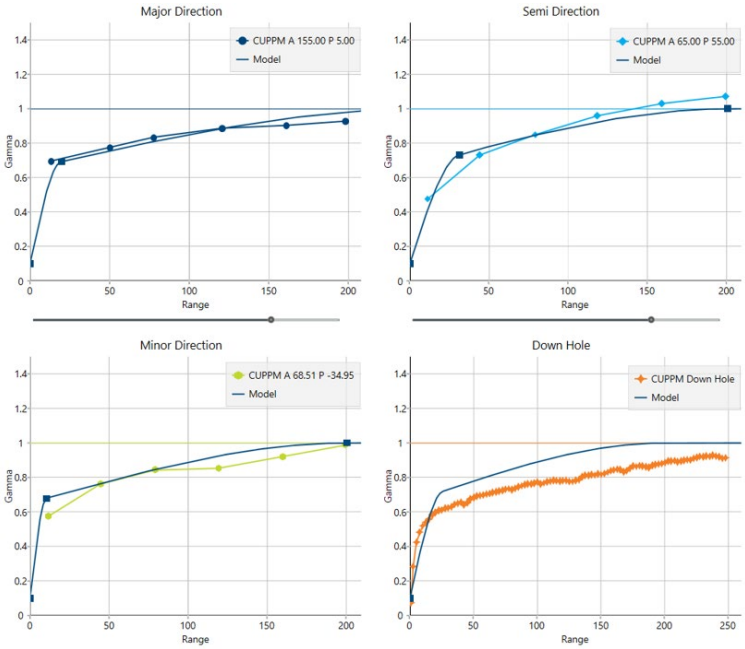
Source: MTS 2025

Figure 14.15: Silver Correlogram



Source: MTS 2025

Figure 14.16: Copper Correlogram



Source: MTS 2025

Table 14.7: Correlogram Parameters for Gold, Silver and Copper

Element	CO	C1	Maj	Semi	Minor	Bear	Plunge	Dip	C2	Maj	Semi	Minor	Bear	Plunge	Dip
Gold	0.20	0.39	68	40	10	170	74	-68	0.41	110	250	204	204	40	42
Silver	0.22	0.53	8	10	10	135	0	75	0.25	200	107	135	135	0	75
Copper	0.10	0.55	20	50	10	165	0	-70	0.35	250	200	165	165	0	-70

Table 14.8: Estimation Plan for Gold, Silver and Copper

Ind	Pass	Min	Max	From DH	Maj	Semi	Minor	Bearing	Plunge	Dip
1	1	7	15	3	25	25	25	LVAB	LVAP	LVAD
1	2	7	15	3	50	50	25	LVAB	LVAP	LVAD
1	3	5	12	3	100	100	50	LVAB	LVAP	LVAD
1	4	3	8	3	250	250	50	LVAB	LVAP	LVAD
1	6	NN Fill								
0	1	7	15	3	25	25	25	LVAB	LVAP	LVAD
0	2	7	15	3	50	50	25	LVAB	LVAP	LVAD
0	3	5	12	3	75	75	50	LVAB	LVAP	LVAD
0	4	3	5	3	150	150	50	LVAB	LVAP	LVAD
0	5	3	5	3	300	300	100	LVAB	LVAP	LVAD
0	6	NN Fill								

Note: Estimation was completed by lithology domain and grade shell domain (Ind)

14.7 Block Model

The CdG block model was constructed using 10 x 10 x 5 m blocks (x,y,z) in Vulcan software. Table 14.9 summarizes the extent and block dimensions of the CdG block model.

Table 14.9: Block Model Limits

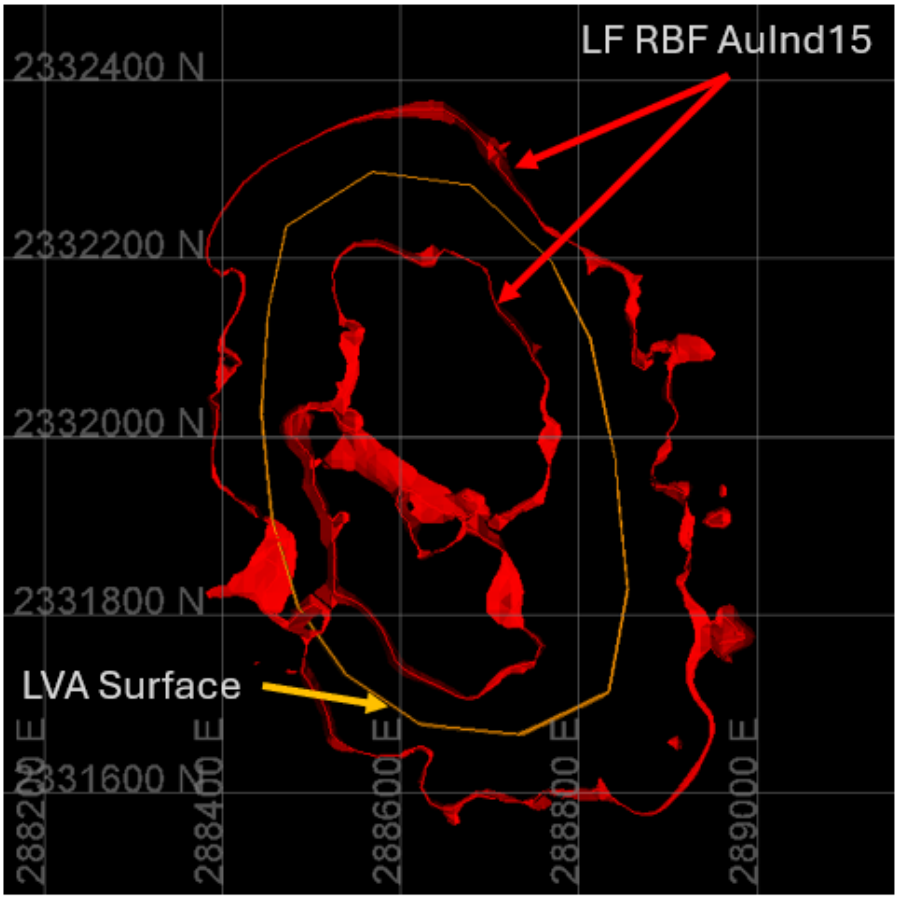
Parameter	Minimum	Maximum	Extent (m)	Block Size (m)	Number
Easting	287,700	290,000	2,300	10	230
Northing	2,330,900	2,332,850	1,950	10	195
Elevation	1,500	2,325	825	5	165

14.7.1.1 Local Varying Anisotropy (LVA) Surface

The mineralization generally forms a halo around the CdG intrusive rocks (F_INT). To accommodate changes in search and variogram orientation during grade estimation, a Local Varying Anisotropy (LVA) was implemented to control the variogram and search orientations during grade estimation. The gold indicator shell was chosen to assist in the construction of the LVA surface.

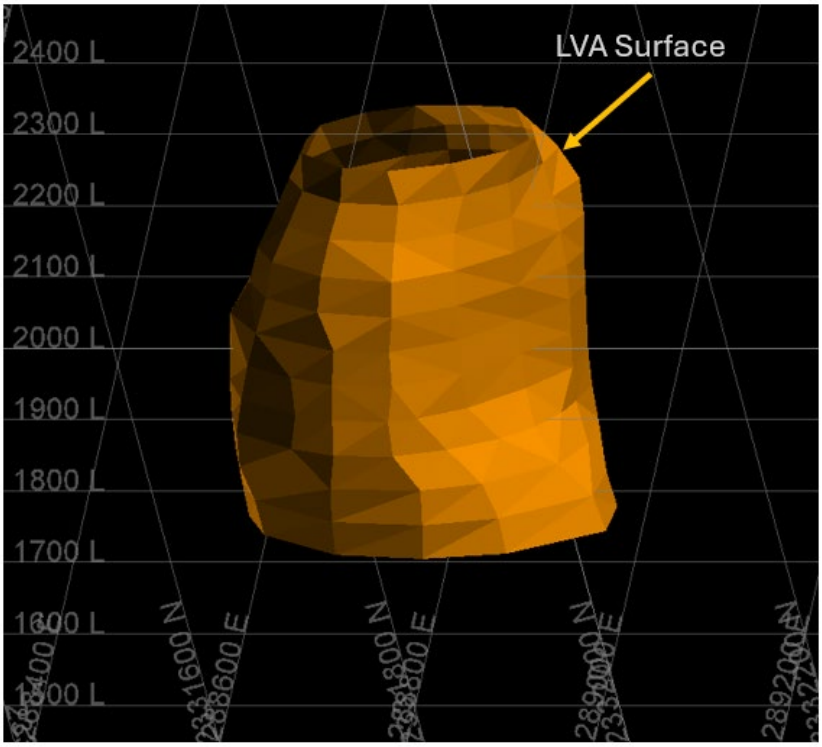
Bench polygons were constructed at 20 m intervals using the gold grade shell as a guide (Figure 14.17). The bench polygons were used to construct a LVA wireframe (Figure 14.18). The wireframe was used to populate LVA fields for bearing, plunge and dip in each block of the block model. The LVA data were used to control variogram and search orientations at each block during grade estimation.

Figure 14.17: Plan View of LVA Surface and Gold RBF



Source: MTS 2025

Figure 14.18: LVA Surface



Source: MTS 2025

14.8 Grade Estimation Methodology

Block grades were interpolated for gold, silver and copper by lithology domain both inside and outside of the grade shell domains.

A multi-pass ordinary kriging (OK) estimation method was used. An ID3 estimate was also completed for comparison. The block estimations were accomplished using only drill hole composites located within the respective lithology and grade shell domains.

Table 14.10 summarizes the parameters that were used for grade estimation for gold, silver and copper.

The directions of variogram anisotropy were modified for each block based on the LVA parameters in each block. The search ellipse was also controlled by the LVA parameters.

14.9 Outlier Grade Restrictions

Constraining higher-grade samples were achieved by outlier restriction. Higher grades were permitted within the search distance of the outlier restriction. The composite grades were then capped at the grade threshold level beyond the distance threshold. Distance and grade thresholds used during grade estimation are summarized in Table 14.10. An estimated 1% of metal was removed using the outlier restriction.

Table 14.10: Outlier Restriction Strategy by Element

Element	Distance Threshold (m)	Grade Threshold (ppm)		
		F_INT	HF	SHBK
Gold	10	7	7	0.4
Silver	10	200	200	80
Copper	10	12000	10000	3000

14.10 Sample Sharing

Based on contract profiles, a firm boundary with sample sharing of 30 m was implemented for the HF and F_INT lithology domains during estimation. Table 14.11 summarizes the sample sharing strategy between the F_INT and HF lithology domains. During the estimation of the F_INT domain, samples in the HF domain within 30 m of the contract were used for estimation. During the estimation of the HF domain, samples in the F-INT domain within 30 m of the contract were used for estimation.

Table 14.11: Block Model Limits

Firm Boundary							
Domain	Share	Rotation			Search Ellipse		
F_INT	HF	LVAB	LVAP	LVAD	30	30	30
HF	F_INT	LVAB	LVAP	LVAD	30	30	30

14.11 Density

Density determinations were collected during drilling programs completed before 2015. The data was collected by various methods (Refer to Section 11-2).

Density was assigned by lithology domain and oxide domain. A summary of the density assignments are summarized in Table 14.12.

There is a paucity of density data for some Oxide material types. Additional density data should be collected from future drill programs.

Table 14.12: Summary of Density Data

Lithology Domain	Oxide Domain	Number	SG
Default	**NA**	**NA**	**2.66**
Dolerite	**NA**	**NA**	**2.66**
F_INT2	**NA**	**NA**	**2.68**
F_INT	Oxide	13	2.49
	Mixed Oxide	14	2.66
	Mixed Sulfide	33	2.62
	Sulfide	147	2.64
HF	Oxide	16	2.57
	Mixed Oxide	8	2.64
	Mixed Sulfide	57	2.66
	Sulfide	446	2.68
SHBK	Oxide	None	2.66
	Mixed Oxide	None	2.66
	Mixed Sulfide	None	2.66
	Sulfide	14	2.75
Total	All	777	2.66

14.12 Model Validation

The CdG block model was validated by visual and statistical methods. Model validation included global bias check, local bias check (swath plots), change of support analysis (HERCO) and visual inspection.

14.12.1 Bias Check

A global bias check was completed using a zero cutoff. Table 14.13 summarizes the global bias checks by Indicated resource class and by combined Indicated+Inferred resource class.

For Indicated Mineral Resources the global bias for gold and silver is within 1%. Copper is within 2%. For the combined Indicated+Inferred class, gold and silver are within 2% and copper is within 3%.

The QP concluded that the mineral resource model is unbiased.

Table 14.13: Bias Check for Gold, Silver and Copper Estimates

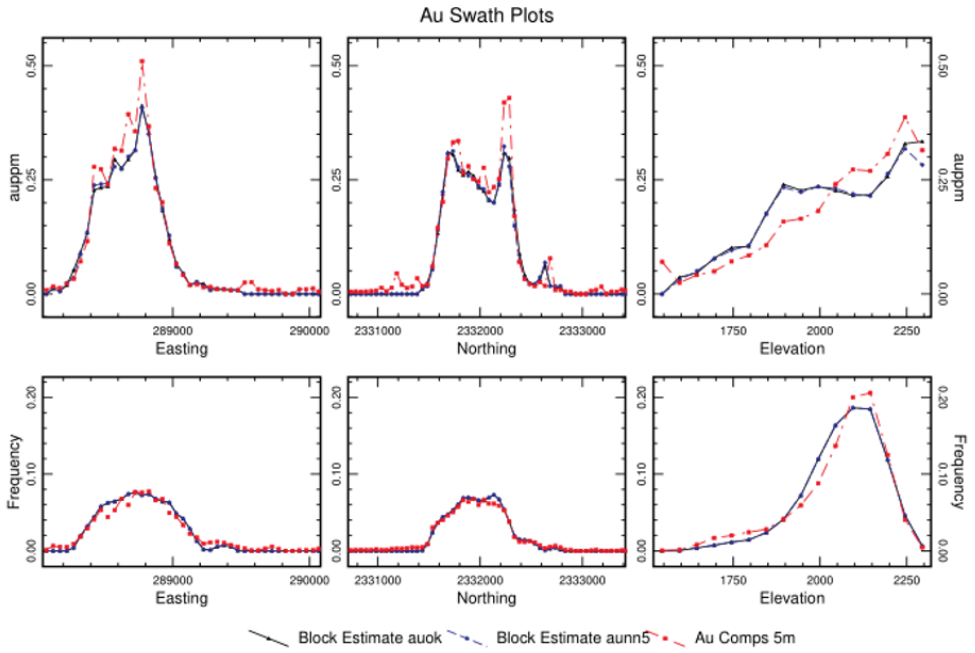
Indicated Classification						
Domain	**Num Blocks**	**OK Mean (ppm)**	**NN Mean (ppm)**	**CV OK**	**CV NN**	**%Rel Difference**
Gold	387,625	0.227	0.228	1.23	1.63	-0.4%
Silver	387,625	9.91	9.86	0.86	1.37	0.5%
Copper	387,625	847	833	0.76	1.04	1.7%
Indicated + Inferred Classification						
Gold	1,043,214	0.127	0.125	1.60	2.07	1.6%
Silver	1,043,214	6.5	6.48	1.10	1.62	0.3%
Copper	1,043,214	613	600	0.75	1.09	2.2%

14.12.2 Swath Plots

Swath plots were completed to check for local bias. The swath plots are presented in Figure 14.19 (gold), Figure 14.20 (silver) and Figure 14.21 (copper). The upper portion of the swath plots present a comparison of the block OK estimated grade, NN estimated grade and grades of the 5 m composites. The lower portion of the swath plots indicated the quantity of data within each swath.

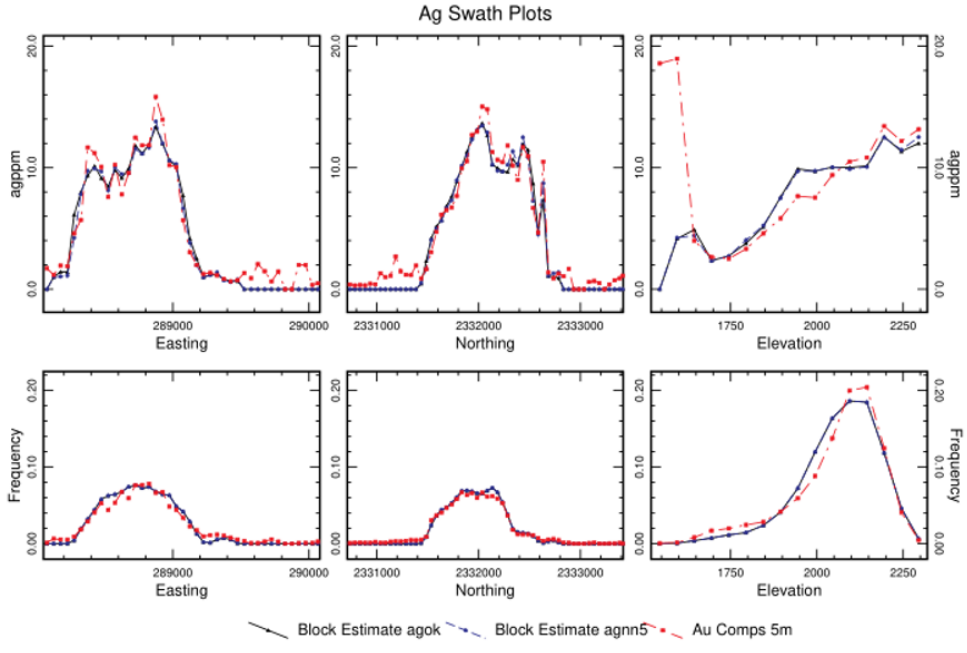
The swath plots show the OK and the NN estimates compare closely. The plots diverge locally where there are limited data.

Figure 14.19: Gold Swath Plot for Indicated Blocks



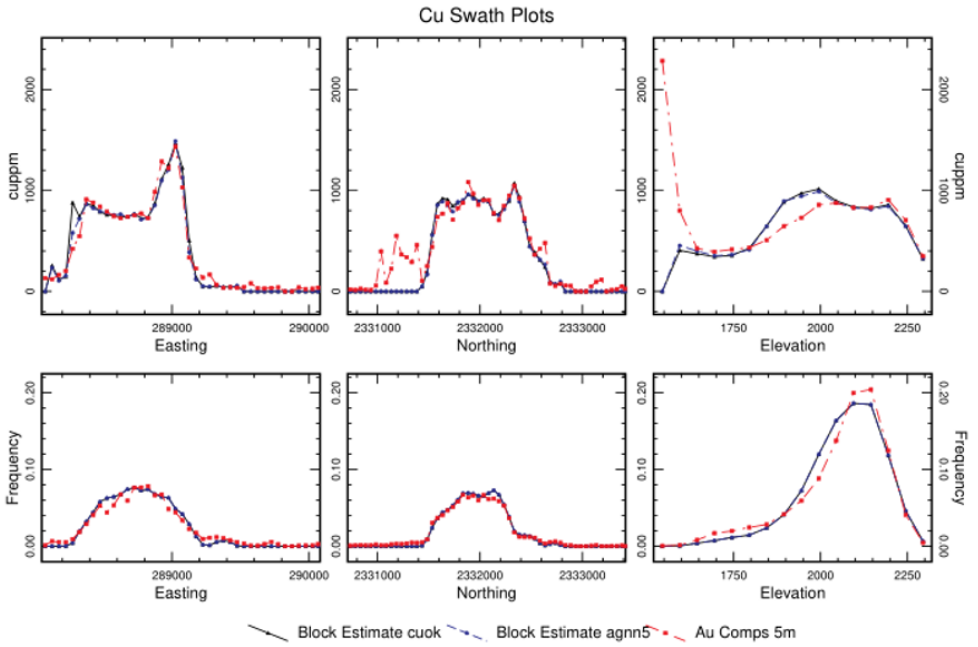
Source: MTS October 2025

Figure 14.20: Silver Swath Plot for Indicated Blocks



Source: MTS October 2025

Figure 14.21: Copper Swath Plot for Indicated Blocks

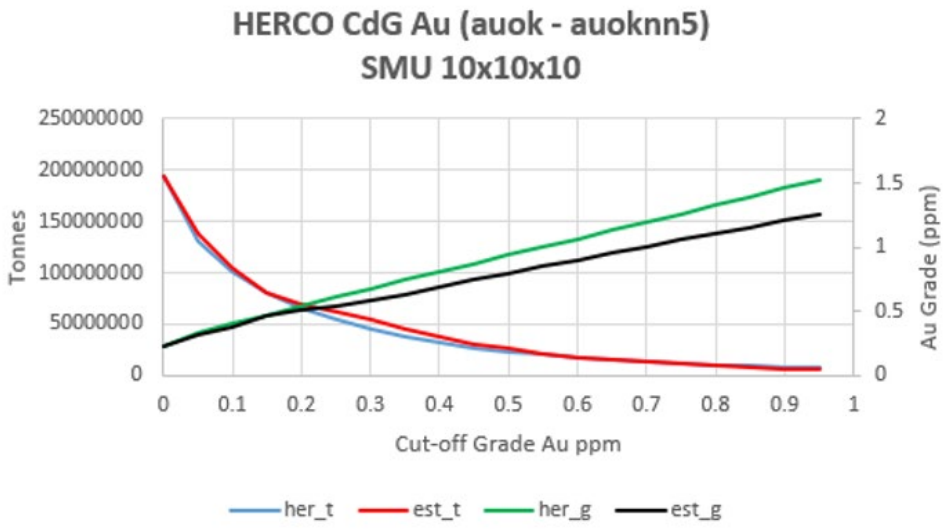


Source: MTS October 2025

14.12.3 Change of Support

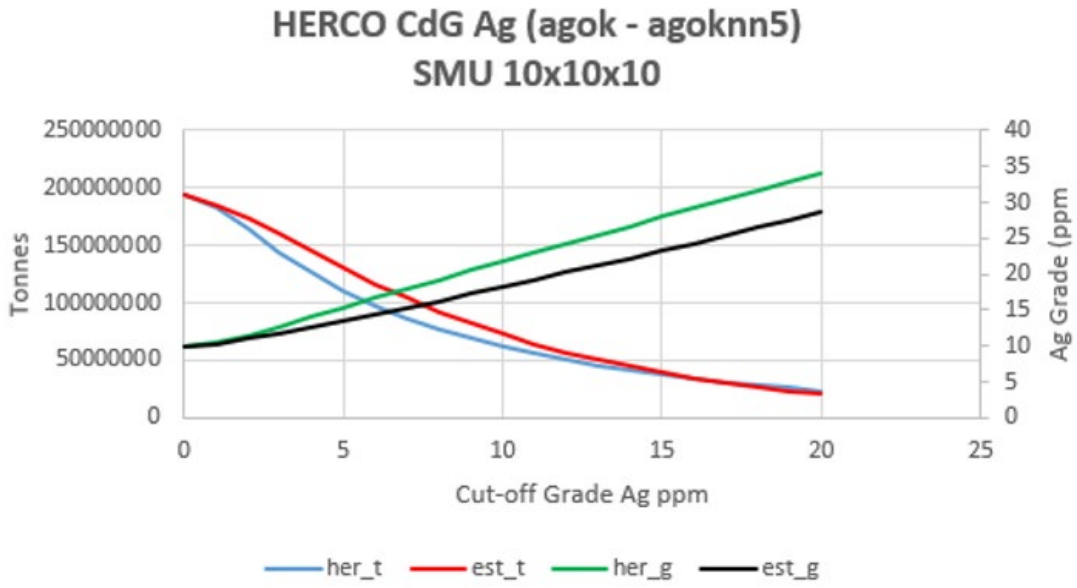
Change of support Hermite Polynomial (HERCO) comparisons plots were prepared for gold, silver and copper. A 10 x 10 x 5 (x,y,z) Selective Mining Unit (SMU) was used. The HERCO analysis was limited to blocks classified as Indicated Mineral Resources. No resources are classified as Measured Mineral Resources at CdG. The HERCO plots for gold, silver and copper show acceptable profiles (Figure 14.22 to Figure 14.24).

Figure 14.22: Hermite Polynomials (Herco) Plot for Gold; Indicated Only



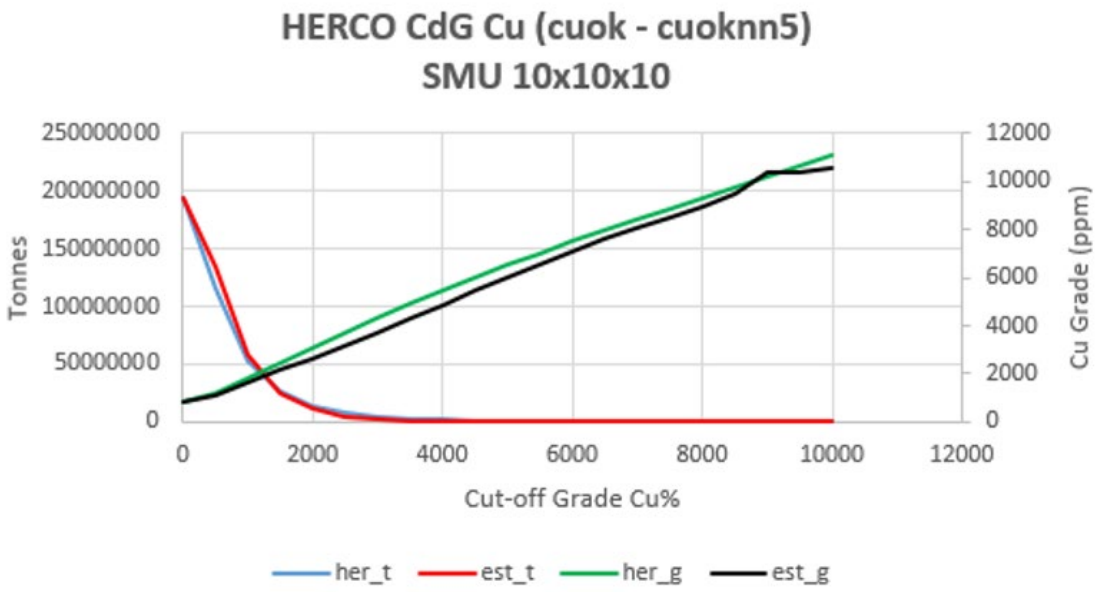
Source: MTS 2025

Figure 14.23: Hermite Polynomials (Herco) Plot for Silver; Indicated Only



Source: MTS 2025

Figure 14.24: Hermite Polynomials (Herco) Plot for Copper; Indicated Only



Source: MTS, 2025

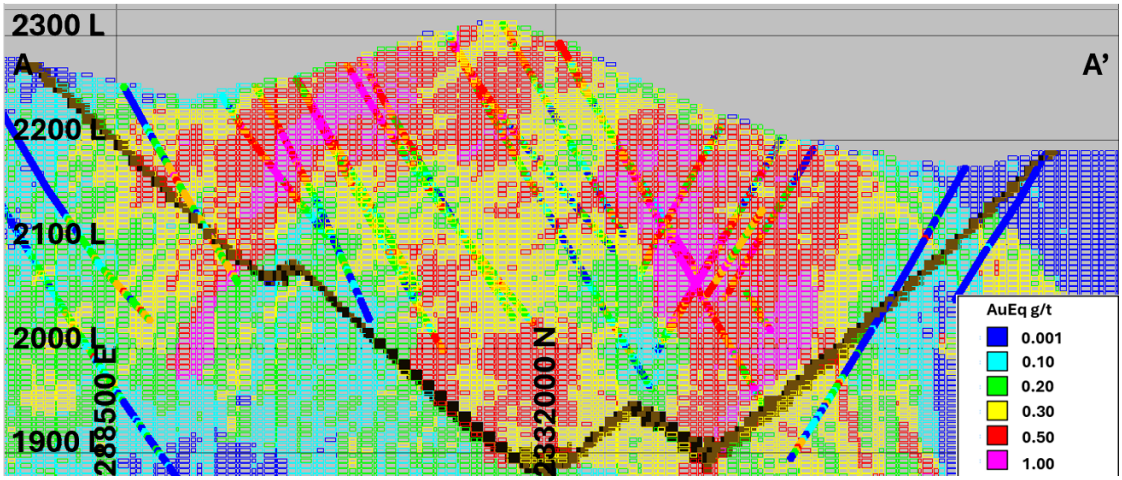
14.12.4 Visual Inspection

Block grades for gold, silver and copper were compared against drill hole composite grades in cross section and plan views. Figure 14.25 and Figure 14.26 show a typical cross section and a plan view through the deposit showing grades of drill hole composites and estimated block grades.

The QP concludes that the estimated block grades generally reflect the drill hole composite data. Observations of silver indicated there were local issues where block grades do not compare with drill data.

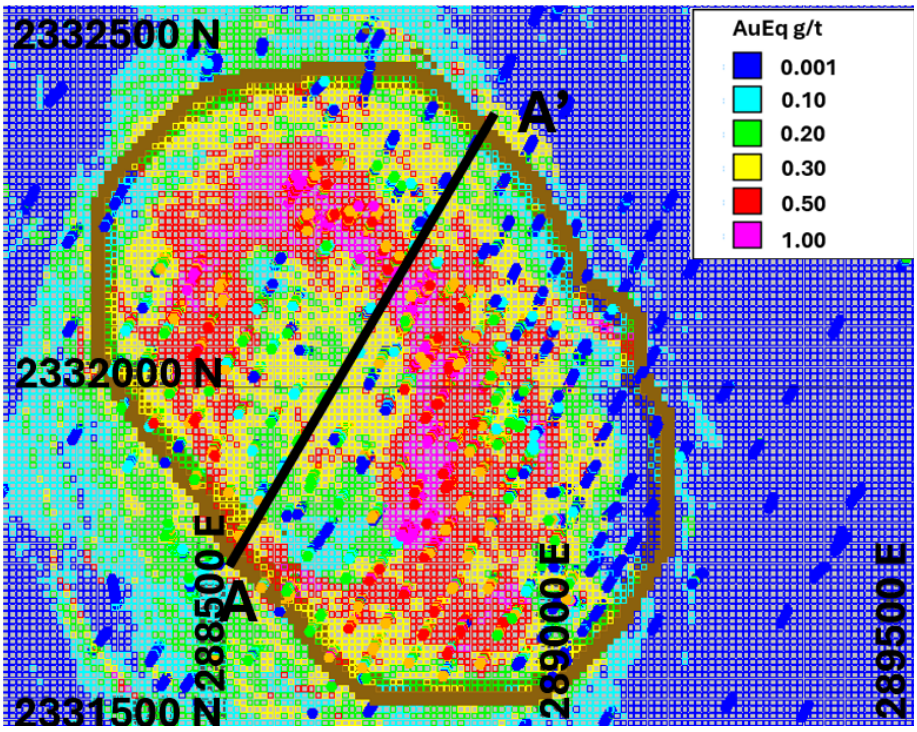
The QP recommends additional geology interpretations be applied to the silver mineralization. Relogging of core holes is recommended with focus on controls for silver mineralization and correlation of mineralization between drill holes.

Figure 14.25: Cerro de Gallo Cross Section Showing Gold Grades for Blocks and Composites (Looking Northwest}



Source: MTS, 2025.

Figure 14.26: Cerro de Gallo Bench 2100 m Showing Gold Equivalent Grades for Blocks, Composites and Resource Shell



Source: MTS, 2025.

14.13 Mineral Resource Classification

As a part of Mineral Resource classification, the QP used the criteria that grade, tonnage and metal estimates should have a 90% confidence interval of ±15% for Measured Mineral Resources within a quarterly production increment. Indicated Mineral Resources should have a 90% confidence interval of ±15% within an annual production increment.

The results of the analysis suggested that a nominal drill spacing of 25 m would be required to classify Measured Mineral Resources. A nominal drill spacing of 55 m was required for classification of Indicated Mineral Resources.

Because of issues observed during the database audit, the QP chose not to include a Measured Mineral Resource in the classification.

The QP compared block grade estimates to drill hole data in cross-sections and plan section. The QP chose to relax the Indicated Mineral Resource criteria to a maximum nominal drill spacing to 75 m.

A drill hole spacing of 75 m (blocks with three drill holes falling within 75 m) was used to classify model blocks as Indicated Mineral Resources. A nominal drill hole spacing of 150 m (three drill holes within 150 m) was chosen for Inferred Mineral Resources. Review of cross-sections noted some estimation blow-outs of Inferred Mineral resources occurred at depth but these are well below the 2025 Mineral Resource shell. These areas should be investigated further if they are included in future resource shells.

14.14 Reasonable Prospects of Eventual Economic Extraction

A conceptual resource shell was generated using the Lerchs–Grossmann (LG) algorithm and an NSR cutoff. NSR cutoffs were determined for each oxide material type. Metal recoveries were also estimated for each oxide material type. Metal prices for the resource shell were $2,500/oz for gold, 30.50/oz for silver and $4.60/lb for copper. Table 14.14 summarizes the parameters that were used to generate the conceptual pit and

Table 15.1 summarizes the cost parameters used for the L-G optimizations.

Table 14.14: Conceptual Pit Parameters for Lerchs-Grossman Analysis

Parameter		Oxide	Mixed Oxide	Mixed Sulfide	Sulfide
Gold Recover (AuRec)		74%	68%	61%	53%
Silver Recover (AgRec)		60%	73%	58%	35%
Copper Recovery (CuRec)		17%	62%	73%	59%
Gold Price		$2500/oz			
Silver Price		$30.50/oz			
Copper Price		$4.60/lb			
Gold Payable		98.5%			
Silver Payable		90.1%			
Copper Payable		88.2%			

14.15 Au Equivalent

Based on the stated metal prices and recoveries (Table 14.14) the gold equivalent grades were calculated with the equation:

- AuEq = Au Grade + (((Cu Price in US$/lb * 22.0462 * Cu Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Cu Grade) + (((Ag Price in US$/g * Ag Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Ag Grade).

The average overall payables from the smelter and refineries were estimated at 98.8% for gold, 90.1% for silver and 88.2% for copper.

14.16 Mineral Resource Statement

Mineral Resources are reported in situ using the 2014 CIM Definition Standards, and are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The effective date for the estimates is 31 Jul 2025. The Qualified Person for the estimate is Mr. Timothy O. Kuhl, SME Reg Member, Principal Geologist with MTS.

14.17 Factors that May Affect the Mineral Resource Estimates

Various factors may affect the mineral resource estimate.
Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- Changes to the long-term gold and silver prices and exchange rates;
- Changes in interpretation of mineralization geometry and continuity of mineralization zones.
- Changes to the geological interpretation.
- Changes to the interpretation of the oxidation model.
- Changes to design parameter assumptions that pertain to the conceptual pit design that constrains the Mineral Resources.
- Modifications to geotechnical parameters and mining recovery assumptions.
- Changes to metallurgical recovery assumptions.
- Changes to environmental, permitting, and social license assumptions.
- Ability to obtain or maintain land access agreements, including specifically for the Pima 3 concession at Veta Madre.

Table 14.15: Cerro del Gallo Mineral Resources

Classification	Material Type	NSR Cutoff	Tonnes (kt)	Grade				Contained Metal			
				Au g/t	Ag g/t	Cu %	AuEq g/t	Gold (koz)	Silver (koz)	Copper (t)	AuEq (koz)
Indicated	Oxide	$11.81	10,733	0.41	17.92	0.09	0.60	141	6,184	9,659	207
	Mix Oxide	$10.66	13,613	0.28	11.12	0.08	0.50	123	4,867	10,890	219
	Mix Sulfide	$11.81	70,066	0.40	13.70	0.09	0.68	901	30,862	63,060	1,532
	Sulfide	$11.23	145,572	0.38	13.77	0.11	0.62	1778	64,447	160,129	2,902
	Total		**239,984**	**0.38**	**13.78**	**0.10**	**0.63**	**2,944**	**106,359**	**243,739**	**4,859**
Inferred	Oxide	$11.81	2,042	0.19	21.08	0.09	0.40	12	1,384	1,838	26
	Mix Oxide	$10.66	1,604	0.14	16.12	0.07	0.40	7	831	1,123	21
	Mix Sulfide	$11.81	10,501	0.28	13.75	0.11	0.57	95	4,642	11,552	192
	Sulfide	$11.23	10,300	0.33	11.74	0.07	0.51	109	3,888	7,210	169
	Total		**24,448**	**0.28**	**13.67**	**0.09**	**0.52**	**224**	**10,746**	**21,722**	**408**

Notes to accompany Mineral Resources table:

1. Mineral Resources are reported within a resource shell constrained by the property boundary using the 2014 CIM Definition Standards.
2. Mineral Resources have an effective date of 31 July 2025. The Qualified Person for the estimate is Mr. Timothy O. Kuhl, Reg Mem SME and Principal Geologist with Mine Technical Services.
3. An NSR is used for reporting Mineral Resources by material type. NSR cutoffs of $11.81 for Oxide, $10.66 for Mixed Oxide, $11.81 for Mixed Sulfide and $11.23 for Sulfide were used. The NSR is determined based on estimated processing costs of US$9.10/t, general and administrative costs of US$0.90t, production taxes and royalty costs of US$1.40/t. Metal prices of US$2,500/oz Au, US$30.50/oz Ag, and US$4.60/lb Cu were used in calculating the NSR. In addition, a gold recovery of 74%, a silver recovery of 60% and a copper recovery of 17% were used for Oxide material, a gold recovery of 68%, a silver recovery of 73% and a copper recovery of 62% were used for Mixed Oxide material, a gold recovery of 61%, a silver recovery of 58% and a copper recovery of 73% were used for Mixed Sulfide material and a gold recovery of 53%, a silver recovery of 35% and a copper recovery of 59% were used for Sulfide material in the NSR calculation.
4. Based on the stated metal prices and recoveries the gold equivalent grades were calculated as AuEq = Au Grade + (((Cu Price in US$/lb * 22.0462 * Cu Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Cu Grade) + (((Ag Price in US$/g * Ag Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Ag Grade). The average overall payables from the smelter and refineries were estimated at 98.8% for gold, 90.1% for silver and 88.2% for copper.
5. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
6. Tonnage and grade estimates are in metric units.
7. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

14.18 QP Comments on Section 14

There are no environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

There is an upside potential for mineralization that is currently classified as Inferred can be upgraded to higher-confidence Mineral Resource categories.

The QP recommends a relogging of drill core with focus on:

- lithology domains,
- oxidation domains,
- silver mineralization and controls.

Lithology and oxidation domains are important for metallurgical types and recoveries. The determination of the controls of silver mineralization is critical for developing an improved silver model.

The QP recommends additional density determinations be obtained from future drill programs.

15 Mineral Reserve Estimate

15.1 Introduction

Mineral Reserves were estimated for the Cerro de Gallo Project which is planned to be an open pit operation with a heap leach pad. Pregnant solution from the heap leach will be processed in a Sulfidization, Acidification, Recirculation, Thickening (SART) plant where a copper - silver precipitate product will be produced, followed by a carbon adsorption-desorption-recovery (ADR) plant to produce a doré product.

The Mineral Reserves are based on three phases of pit designs developed by the QP for the pre-feasibility study. The open pit designs completed on the Cerro de Gallo deposit were evaluated with the updated Measured and Indicated Mineral Resources and were demonstrated to be economically viable, therefore Measured and Indicated Mineral Resources within the pit designs have been converted to Proven and Probable Reserves. All Inferred material was classified as waste and scheduled to the appropriate waste rock storage facility (WRSF). The mineral reserves use the terminology, definitions and guidelines given in the CIM Standards on Mineral Resource and Mineral Reserves (May 2014).

15.2 Estimation Parameters

The pit designs used in this study were developed based on pit optimizations run on the updated operating costs and metal prices used in this report. The pit optimizations were constrained by a major drainage along the northwest side of the pit, at depth due to a limited metallurgical testing and to the maximum leach pad capacity of 92MT.

The Mineral Reserves are reported using a $12.50/t Net Smelter Return (NSR) cut-off inside the pit designs, which are discussed in more detail in the following sub-sections together with the pit design parameters pit optimization constraints.

15.2.1 Pit Slopes

The pit slopes follow recommendations provided by Mines Group Inc. (“Mines Group”); see discussion in Section 16.2.

The recommended inter-ramp slope angles range from 45° to 50°, for the purpose of the pit optimizations, the inter-ramp slope angle was set to a maximum of 42° to account for the inclusion of haulage ramps in the final pit designs.

15.2.2 Ore Loss and Dilution

Ore loss and dilution are discussed in Section 16.5

15.2.3 Pit Optimization

The Mineral Resources for the deposit were evaluated using an Lerchs–Grossmann (L–G) pit optimizer to generate optimized pit shells. Pit shells were generated based on varying metal prices, with base prices of US$2,200/oz Au, US$26.50/oz Ag and US$4.00/lb Cu. Starting with the current topography, a total of 60 pit shells were generated to determine optimal break points for developing pit phases and for determining the ultimate final pit phase for the deposit.

Table 15.1 shows the cost estimates used for each pit optimization. The operating costs were determined based on historical costs provided by Heliostar, and Hard Rock Consulting's and KCA's industry knowledge and prior experience.

15.2.3.1 NSR Calculation

The CdG ore contains cyanide soluble copper that results in high cyanide consumption during the heap leaching process. A SART plant is included in the process that releases cyanide associated with the copper-cyanide complex, allowing a significant portion of it to be recycled back to the leach process as free cyanide, and produces a copper-silver precipitate that can be sold to a smelter. Due to the production of the copper precipitate an NSR cutoff was chosen to estimate the Mineral Reserves.

In order to apply the cutoff for the tabulation of the mineral reserve, each block in the mineral resource model was assigned an NSR (Net Smelter Return) value calculated in $/tonne.

The average total percent recovery of each element was estimated for each material type from metallurgical testing and provided by KCA. The NSR calculation also takes into account all distributions, concentrate treatment charges, refining, payable deductions, and shipping charges for Dore and precipitates produced as shown in

Table 15.1.

Table 15.1: Reserve Pit Optimization Parameters

Reserve Parameters	**Units**			
Operating Costs				
Leach Oxide	$/t ore	$9.11		
Leach Mixed Oxide	$/t ore	$7.96		
Leach Mixed Sulfide	$/t ore	$9.11		
Leach Fresh	$/t ore	$8.53		
G&A	$/t ore	$1.95		
Selling/Finishing/Prod Taxes	$/t ore	$0.75		
Mining Cost – Surface	$/t mined	$2.15		
Mining Cost - Incremental Increase for each 5m depth	$/t mined	$0.014		
	Units	**Au**	**Ag**	**Cu**
Commodity Prices	$/oz or $/lb	$2,200	$26.50	$4.00
Total Process Recoveries				
Oxide	%	74.0%	60.0%	17.0%
Mixed Oxide	%	68.0%	73.0%	62.0%
Mixed Sulfide	%	61.0%	58.0%	73.0%
Sulfide	%	53.0%	35.0%	59.0%
Distributions				
SART Cu Concentrate	%	3.1%	98.0%	98.0%
Dore	%	96.9%	2.0%	2.0%
Payables				
Copper Con Payables	%	90.0%	90.0%	90.0%
Dore Payables	%	99.2%	95.0%	0.0%
Treatment/Refining Charges				
Copper Con Moisture	%	10.0%		
Copper Con Deduct	% Cu	1.2%		
Copper Con Deduct	Au g/t	1.00		
Copper Con Deduct	Ag g/t	30.00		
Copper Con Refining	Cu $/lb	$0.08		
Copper Con Refining	Ag $/oz	$0.45		
Copper Con Refining	Au $/oz	$6.00		
Copper Con Trucking & Shipping $/t conc	wet	$225.00		
Copper Con Treatment $/t conc	wet	$210.00		
Copper Con Penalties $/t conc	dry	$50.00		
Dore Shipping	$/oz Au & Ag	$1.50		

15.2.4 Pit Limitations

A major drainage runs along the northwest side of the ultimate pit. For the purpose of the pit optimizations, a boundary was defined that is offset approximately 10 meters from the center of the drainage toward the deposit. The boundary was used directly in the optimization process as a constraint to ensure the pit would not encroach on the drainage. This boundary is shown in purple with the ultimate optimized pit shell shown in green in Figure 15.1 below. The pit optimizations were also limited by depth due to lack of metallurgical testing of at depth within the deposit. A boundary was defined that identified

blocks that were greater than 60 meters from a metallurgical sample, all blocks below this surface were treated as waste during the optimization process.

Figure 15.1: Drainage Boundary and Optimized Pit

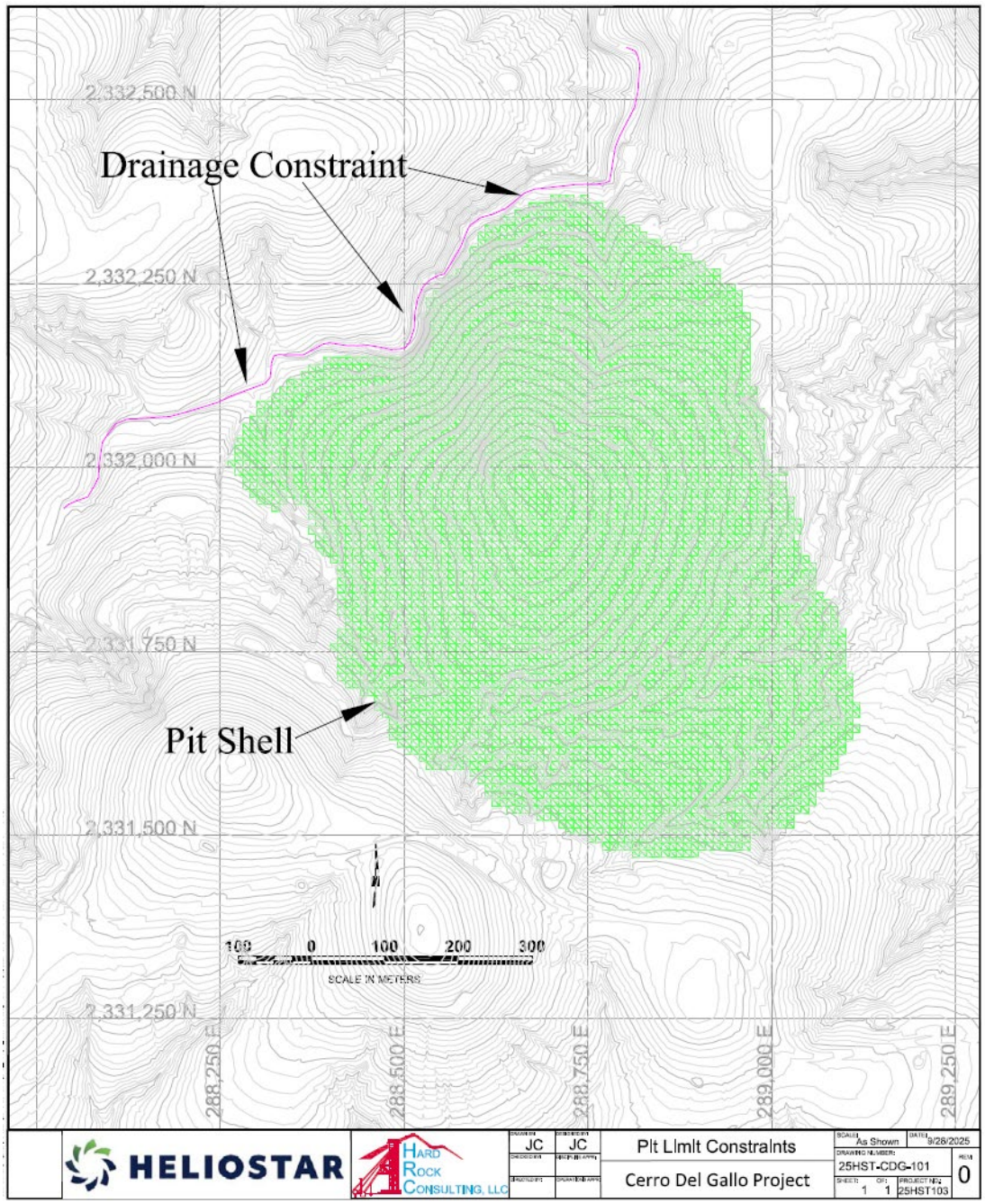
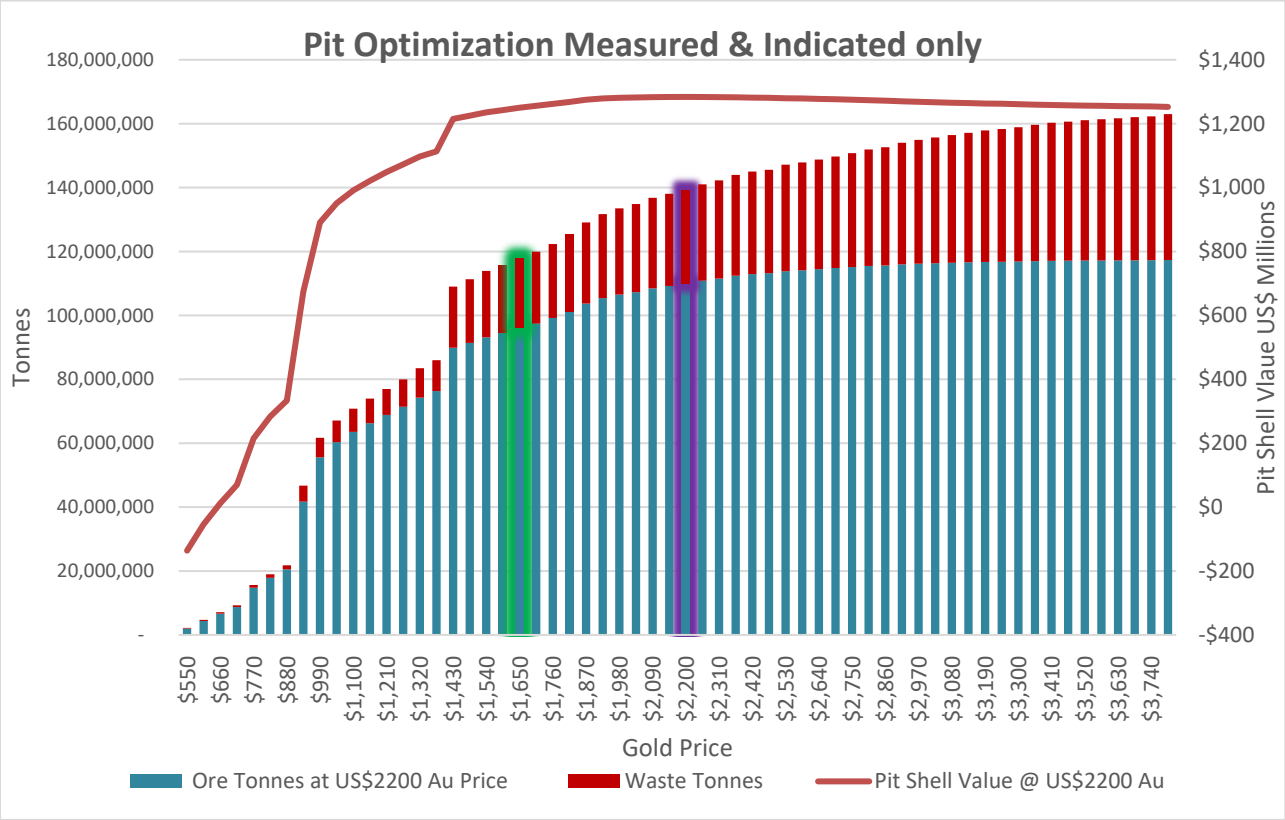


Figure 15.2 shows the optimization results for the Measured and Indicated material within the resource model for CdG. Values in the figure are based on optimized pit shells before the design process, and do not include the haulage ramps and catch benches.

The final pit was also limited to current leach pad design capacity of 92Mt. This resulted in a pit at a $1,650 AuEq price being chosen as the basis for the ultimate pit design. The pit was designed as three phases for the Mineral Reserve evaluation, with the first phase targeting the higher-grade ore on the southwest side of the deposit. The second phase targeted a high-grade zones on the northwest side of the deposit. The third and final phase was based on the $1,650 AuEq pit that was limited by the northwest drainage, at depth due to lack of met samples and the leach pad capacity. The pit phase designs are shown in Figure 16.3 through Figure 16.5.

Figure 15.2: Pit Optimization Results



Note: Figure prepared by Hard Rock Consulting, 2025. The base case pit shell at a gold price of US$2,200/oz is highlighted in purple. The heap leach pad ore capacity pit shell selected for the final design is highlighted in green.

15.2.5 Cutoff

The Mineral Reserves are reported using a $12.50/t NSR cut-off inside the final pit design, which includes the estimated plant operating costs, all general and administrative costs, taxes and royalties as shown in Table 15.2. An average fixed processing cost was chosen for the NSR cutoff versus a variable cutoff by material type in order to limit the Mineral

Reserves to the leach pad capacity. A variable cutoff would have a minimal change to the reported Mineral Reserves and only add 3% more material to the estimation.

Table 15.2: Mineral Reserve NSR Cut-off

Cost Center	$US/ore tonne
Processing	9.10
G & A	0.90
Production Taxes	1.10
Royalties	1.40
Total NSR Cutoff	**12.50**

15.3 Mineral Reserves Statement

The Mineral Reserve estimates are reported using the 2014 CIM Definition Standards. The QP for the estimate is Mr. Jeffrey Choquette P.E., of Hard Rock Consulting. Mineral Reserves are reported at the point of delivery to the process plant and have an effective date of 31 July, 2025.

The Mineral Reserves are reported by material type using a $12.50/t NSR cut-off inside the final pit designs as shown in

Table **15.3**.

Gold equivalency (AuEq) grades were calculated using ratios of metal prices and metal recoveries in the following equation.

- AuEq = Au Grade + (((Cu Price in US$/lb * 22.0462 * Cu Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Cu Grade) + (((Ag Price in US$/g * Ag Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Ag Grade). The average overall payables from the smelter and refineries were estimated at 98.8% for gold, 90.1% for silver and 88.2% for copper.

Table 15.3: Mineral Reserves Statement

Classification	Material Type	Tonnes (kt)	Grade				Contained Metal			
			Au g/t	Ag g/t	Cu%	AuEq g/t	Gold (koz)	Silver (koz)	Copper (t)	AuEq (koz)
Probable	Oxide	9,198	0.46	18.46	0.08	0.65	137	5,459	7,714	193
Probable	Mix Oxide	4,411	0.42	10.74	0.09	0.64	59	1,524	4,115	91
Probable	Mix Sulfide	38,761	0.50	15.26	0.10	0.80	629	19,020	37,354	995
Probable	Sulfide	39,524	0.53	15.00	0.12	0.78	670	19,064	45,557	997
Probable	**Total**	**91,893**	**0.51**	**15.25**	**0.10**	**0.77**	**1,495**	**45,066**	**94,740**	**2,275**

Notes to accompany Mineral Reserves table:

1. Mineral Reserves are reported at the point of delivery to the process plant, using the 2014 CIM Definition Standards.
2. Mineral Reserves have an effective date of 31 July 2025. The Qualified Person for the estimate is Mr. Jeffrey Choquette, P.E., of Hard Rock Consulting.
3. An NSR cutoff of $12.50/t was used for reporting the Mineral Reserves which is based on estimated processing costs of US$9.10/t, general and administrative costs of US$0.90t, production taxes and royalty costs of US$1.40/t. Metal prices of US$2,200/oz Au, US$26.50/oz Ag, and US$4.00/lb Cu were used in calculating the NSR. In addition, a gold recovery of 74%, a silver recovery of 60% and a copper recovery of 17% were used for Oxide material, a gold recovery of 68%, a silver recovery of 73% and a copper recovery of 62% were used for Mixed Oxide material, a gold recovery of 61%, a silver recovery of 58% and a copper recovery of 73% were used for Mixed Sulfide material and a gold recovery of 53%, a silver recovery of 35% and a copper recovery of 59% were used for Sulfide material in the NSR calculation.
4. Based on the stated metal prices and recoveries the gold equivalent grades were calculated as AuEq = Au Grade + (((Cu Price in US$/lb * 22.0462 * Cu Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Cu Grade) + (((Ag Price in US$/g * Ag Recovery and Payable) / (Au Price in US$/g * Au Recovery and Payable)) * Ag Grade). The average overall payables from the smelter and refineries were estimated at 98.8% for gold, 90.1% for silver and 88.2% for copper.
5. Mineral Reserves are reported within the ultimate reserve pit design.
6. Tonnage and grade estimates are in metric units.
7. Mineral Reserve tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

15.4 Factors that May Affect the Mineral Reserves

The Mineral Reserves are currently constrained along the northwestern edge of the pit due to a major drainage feature, limited metallurgical testing at depth, and the capacity of the existing leach pad design. A feasibility study to assess the potential for diverting the drainage could enable pit expansion in this direction, potentially increasing Mineral Reserves. Additional metallurgical testing at depth will help refine recovery estimates, which may lead to either an increase or decrease in ore reserves. Furthermore, redesigning the leach pad—subject to favorable topography—could support a larger footprint or the development of a second leach pad, both of which may contribute to increased Mineral Reserves.

Other areas of uncertainty that may materially impact the Mineral Reserves include the following:

- Variations in the forecast commodity price;
- Variations to the assumptions used in the constraining L–G pit shells, including mining loss/dilution, metallurgical recoveries, geotechnical assumptions including pit slope angles, operating costs and smelter treatment charges;
- Variations in assumptions as to permitting, environmental, and social license to operate.

15.5 QP Comments on Section 15

Mineral Reserves are reported using the 2014 CIM Definition Standards.

As the economic analysis in Section 22 used higher metal prices than were used in the Mineral Reserve estimates, the QP performed a check to ensure that the Mineral Reserves returned positive economics at the Mineral Reserve commodity pricing. The results showed a positive after tax cashflow, thus verifying the Mineral Reserve estimates. There are no other environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of Mineral Reserves that are not discussed in this Report.

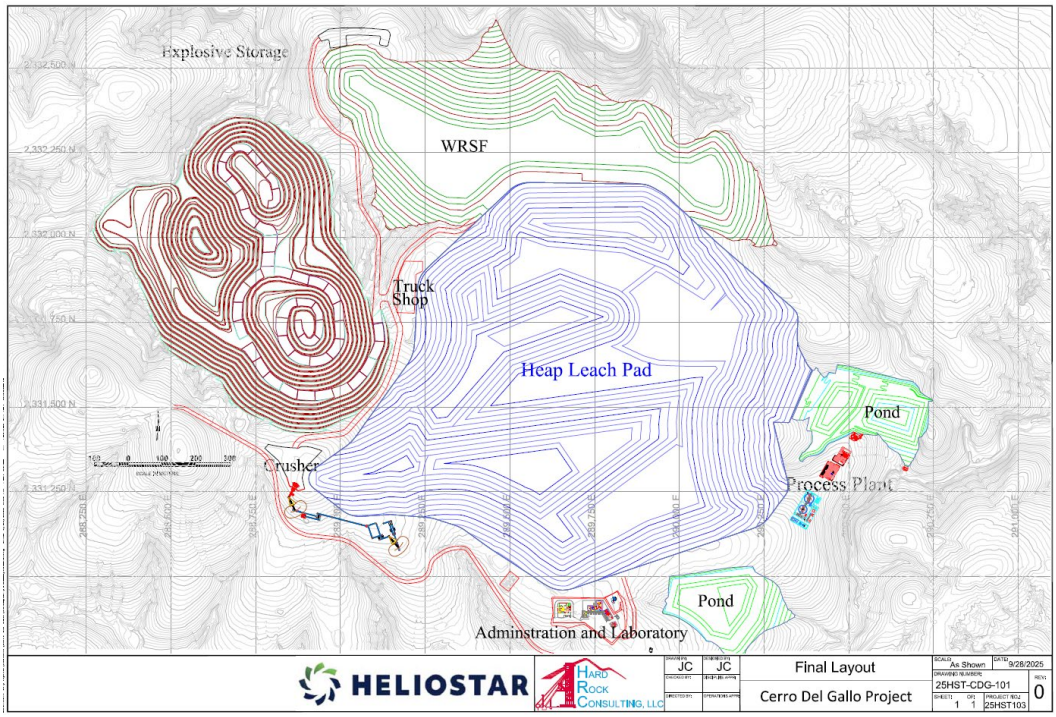
16 Mining Methods

16.1 Overview

The CdG deposit contains mineralization at or near the surface that is ideal for open pit mining methods. The mine plan includes a conventional truck and loader open pit operation. The major production unit operations will include drilling, blasting, loading, hauling, and dumping. These activities are planned to be completed with a contractor fleet. Ore from the pit will be crushed to 80% passing 5mm, agglomerated with cement, conveyor stacked onto a heap leach pad and leached using a low concentration sodium cyanide solution. Pregnant solution from the heap leach will be processed in a Sulfidization, Acidification, Recirculation, Thickening (SART) plant where a copper - silver precipitate product will be produced, followed by a carbon adsorption-desorption-recovery (ADR) plant to produce a doré product.

The processing throughput for the Project is 6 million tonnes of ore per year after the second year, with the first year at 5.6 million tonnes. The mine plan is based on Proven and Probable Mineral Reserves only. A total of 91.9 Mt will be processed during the 15.3-year mine life with an overall LOM plan strip ratio of 0.32:1. A total of 29.3 Mt will be sent to the waste rock storage facility (WRSF) located east of the pit and north of the heap leach pad as shown on Figure 16.1. The WRSF has sufficient capacity for the LOM plan.

Figure 16.1: Final Pit Design and Mine Infrastructure



Note: Figure prepared by HRC, 2025

16.2 Geotechnical Considerations

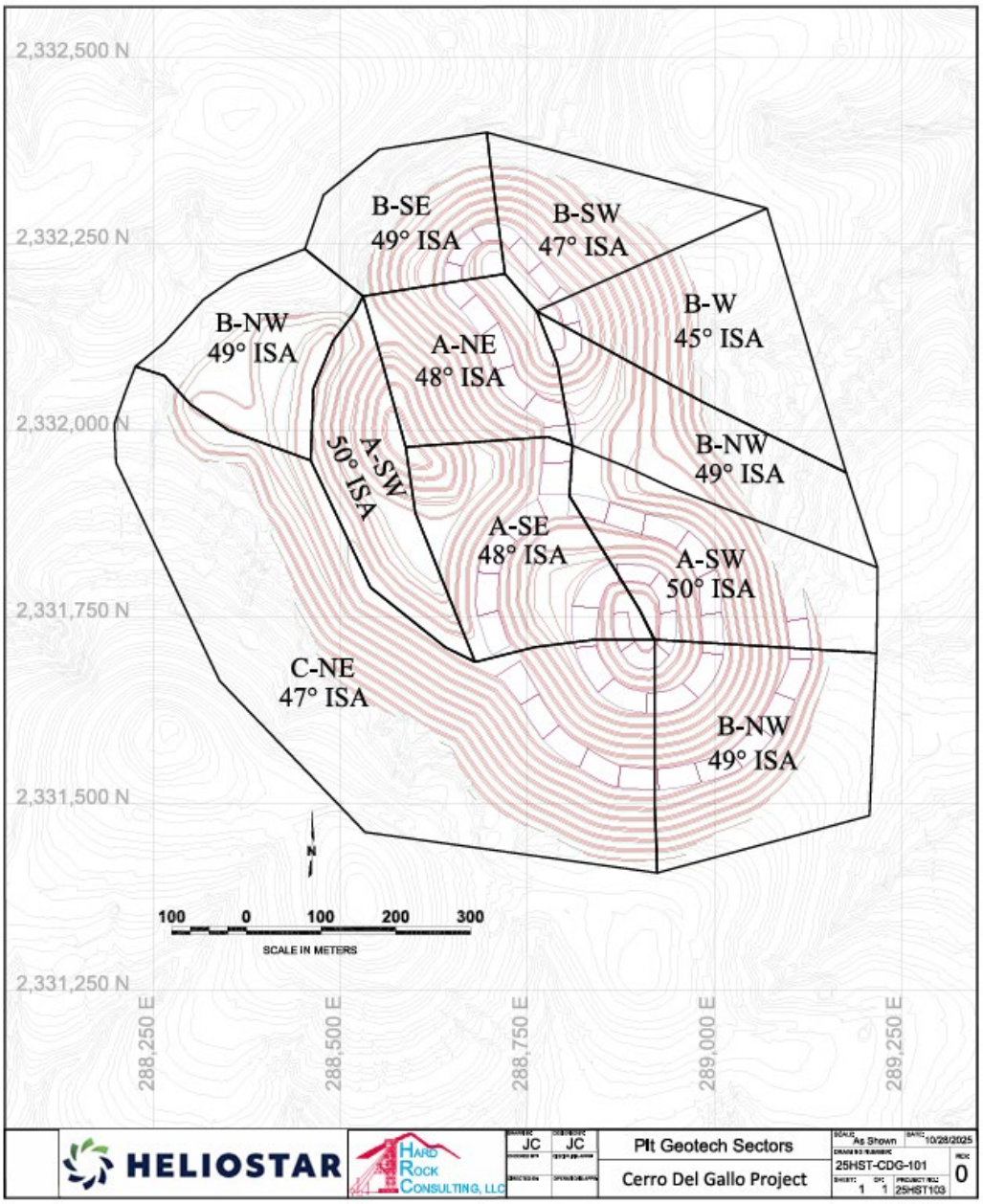
Recommendations for slope parameters were provided by Ken Meyers of the Mines Group Inc. (“Mines Group”) of Reno, Nevada. The pit design is shaped by complex geological and structural conditions. The deposit consists of a barren granodiorite core capped with silica, surrounded by welded ash flow tuff and volcaniclastic sediments primarily composed of shale. Mineralization occurs mainly in a ring around the granodiorite, which results in a configuration of three individual pit bottoms and seven design sectors distributed across three structural domains.

Geotechnical analysis indicates that the rock mass has high unconfined compressive strength and good RQD values, suggesting strong overall conditions. Fracture mapping revealed numerous closely spaced joints, most of which are steeply dipping at angles of 70 degrees or greater, along with some adverse sets between 35 and 65 degrees. Anticipated slope heights of approximately 200 meters contribute to high safety factors from a rock strength perspective. However, slope stability is primarily controlled by rock fabric and backbreak, as well as the potential for multiple bench wedge failures identified through kinematic analysis.

The greatest uncertainty lies in the potential presence of unmapped, adversely oriented fault structures. Continuous subsurface exploration and systematic bench-scale mapping are essential to identify these features early and implement mitigation strategies. Additional characterization of the piezometric surface is also recommended, although current analyses suggest that dewatering measures will effectively manage water-related impacts.

The recommendations as used for pit design are shown in Table 16.1. The maximum inner-ramp angle (“IRA”) recommendations ranged from 45° to 49°. Triple benching to an ultimate bench height of 15 meters is required for all sectors. Figure 16.2 shows the location of each pit slope design sector in relation to the final pit design.

Figure 16.2: Pit Slope Design Sectors



Note: Figure prepared by HRC, 2025. ISA = inter-ramp slope angle.

Table 16.1: Pit Slope Recommendations

Structural Domain	Design Sector	BH (m)	CBW (m)	BFA (º)	ISA (º)
A	NE	15	7.5	68.2	48
A	SE	15	7.5	68.2	48
A	SW	15	7.5	71.3	50
B	SE	15	7.5	69.7	49
B	SW	15	7.5	66.6	47
B	W	15	7.5	63.4	45
B	NW	15	7.5	69.7	49
C	NE	15	7.5	66.6	47

Note: BH = bench height; CBW = catch bench width; BFA = bench face angle; ISA = inter-ramp slope angle.

16.3 Hydrogeological Considerations

The project is located within the Hydrological Region RH-12 “Lerma Santiago”, the RH-12 is made up of 6 Hydrological Sub-regions, and the project is part of the Hydrological Subregion of “La Laja. The climate in this sector is dry, with an average annual rainfall of 586 mm (the heaviest rainfall occurs in summer), with significant evaporation rates.

From the hydrogeological point of view, very little information has been collected. The piezometric conditions within the pit area have not been determined at this time. Conditions are anticipated to be relatively dry in the upper portions of the pit, however, the lower portions of the pit, below about 60 m, is expected to contain water, particularly on the north side of the pit adjacent to a major drainage passing near the pit rim.

The use of in-pit sumps and pumps is planned collect and remove the water from the open pit to permit mining operations. The water from the open pit is anticipated to be required for the ore processing so at this time no treatment or discharge of the water from the open pit is planned. Further studies are required to get a better understating of the amount of water that will be generated from the open pit in the overall project water balance.

16.4 Pit Designs

As discussed in Section 15.2.4, the final pit design was limited to a US$1,650/oz AuEq pit shell due to the currently designed heap leach pad capacity. The pit slopes were designed by geotechnical sector as outlined in Table 16.1. The pit was designed into three phases for the Mineral Reserve evaluation with the first phase targeting the higher-grade ore on the southwest side of the deposit and the second phase targeted a high-

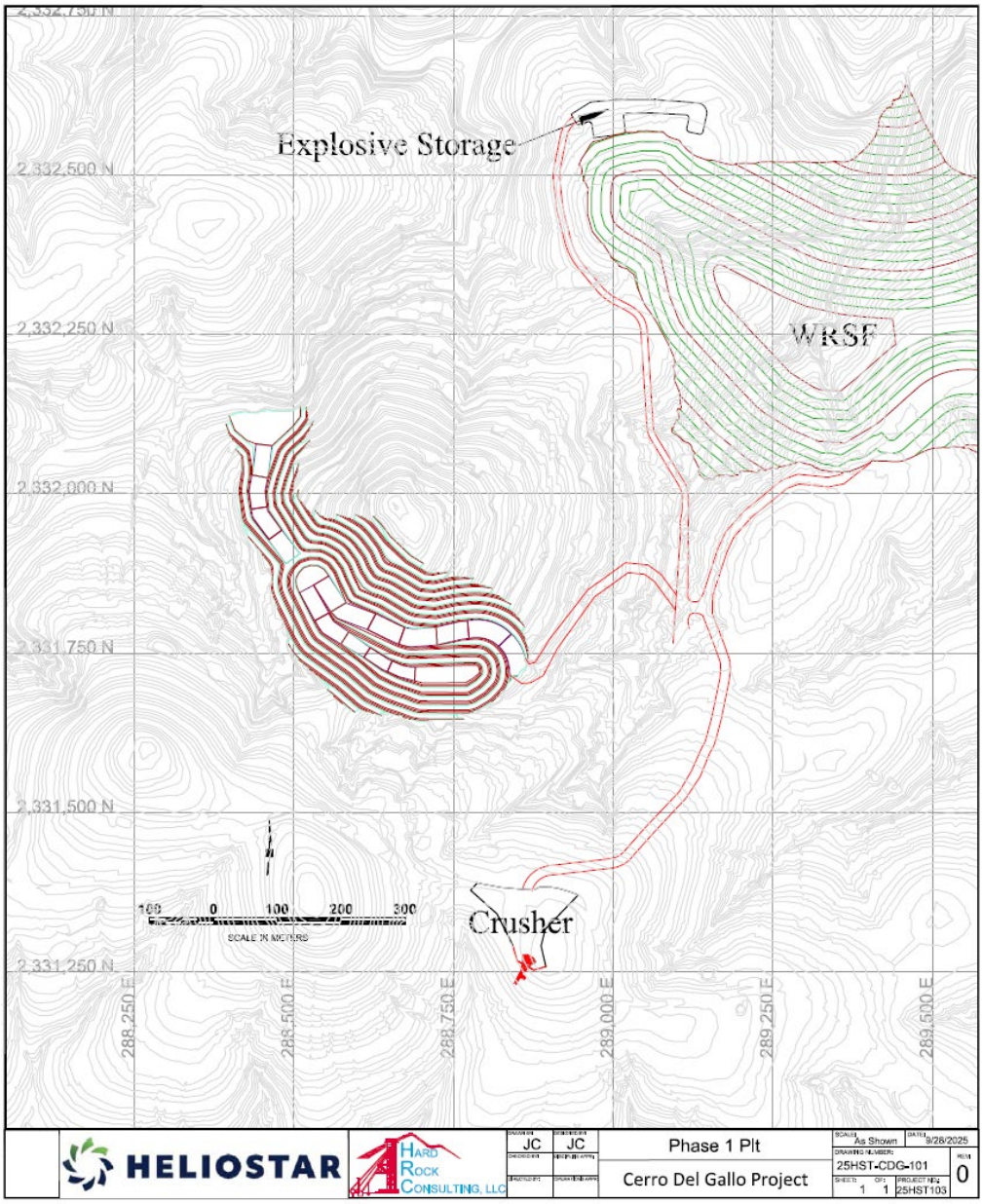
grade zone on the northwest side of the deposit. The pit phase designs are shown in Figure 16.3 through Figure 16.5.

Pit designs were created using five-meter benches for mining. Haul roads are designed at a width of 28 m, which provides a safe truck width (6.7 m wide for Caterpillar (Cat) 777 size truck) to running surface width ratio of 1:3 with an additional 8.6 m for a berm and a drainage ditch. Maximum grade of the haul roads is 10%, except for the lower benches where the grade is increased to 12%, and the ramp width is narrowed to 22 m to minimize excessive waste stripping. The pit design criteria are presented in Table 16.2.

Table 16.2: Pit Design Criteria

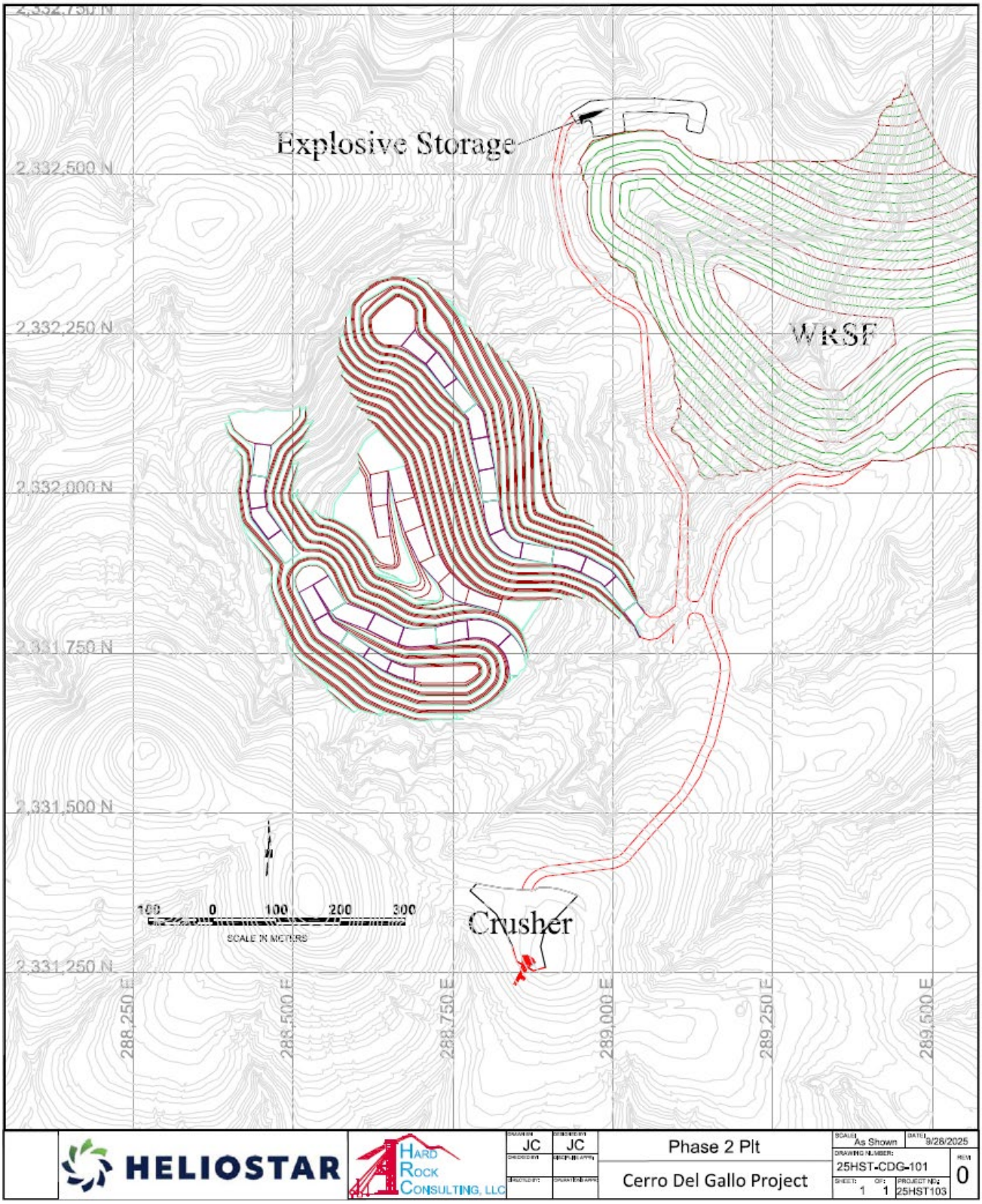
Category	Unit	Value
Ramp widths	m	28
Ramp grade	%	10
Ramp widths pit bottom	m	22
Ramp grade pit bottom	%	12
Mining level heights	m	5

Figure 16.3: CdG Phase 1 Pit



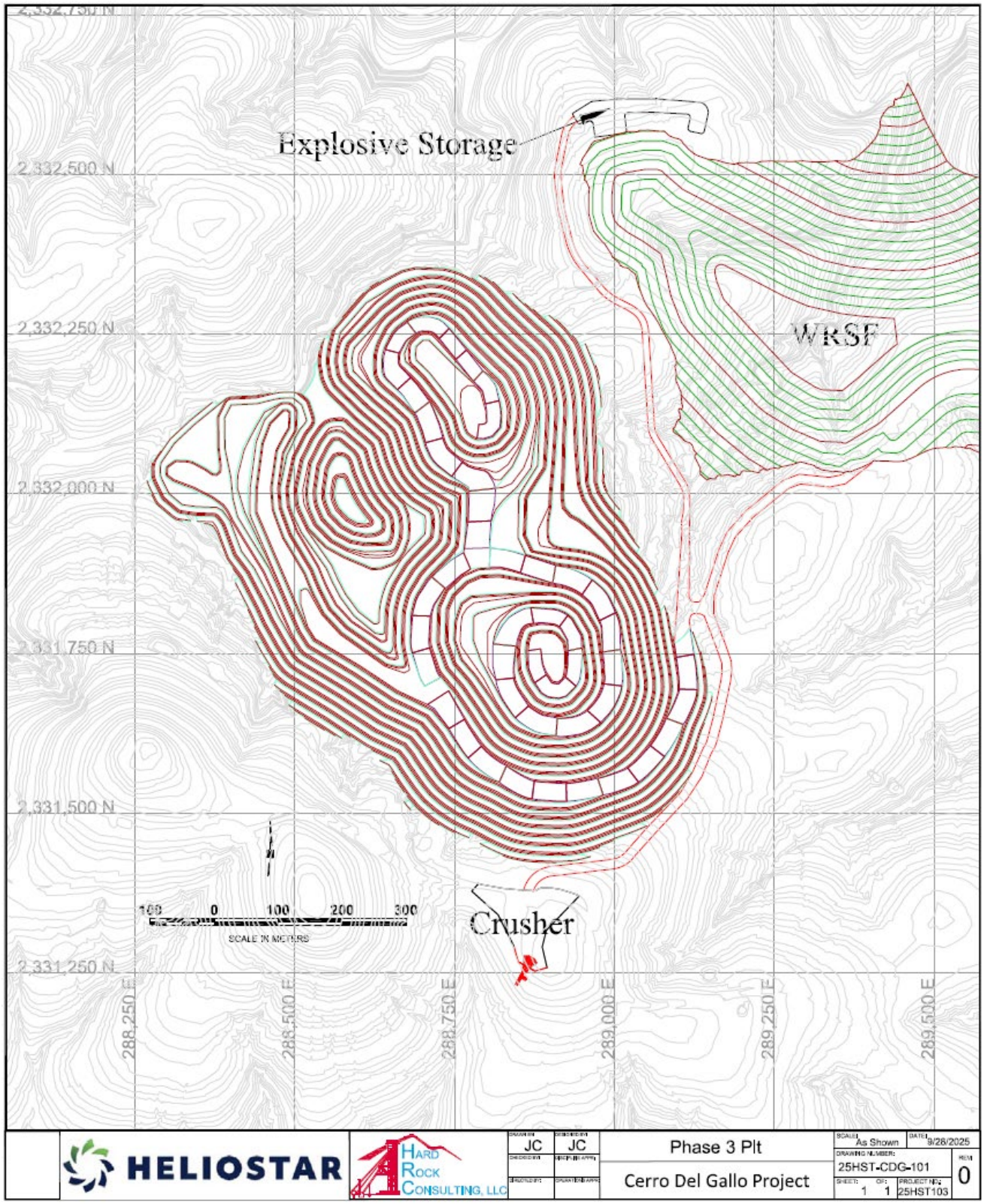
Note: Figure prepared by HRC, 2025

Figure 16.4: CdG Phase 2 Pit



Note: Figure prepared by HRC, 2025

Figure 16.5: CdG Phase 3 Pit (Final Phase)



Note: Figure prepared by HRC, 2025

16.5 Dilution and Mining Losses

The ore zones defined by the Mineral Reserve cutoff are large and continuous with a very favorable strip ratio of 0.32:1 (Waste:Ore). The resource block model used to report resources and reserves also has block sizes of 10 meters (X) by 10 meters (Y) by 5 meters (Z). Due to internal dilution by compositing and the 10-meter by 10-meter by 5-meter block size the QP believes the estimate to be reasonable for mining the deposit with the equipment that has been selected and believes that this represents an appropriate amount of dilution and ore loss for statement of reserves.

16.6 Mine Production Schedule

Production of mineralized material from the open pits are driven by the nominal ore crusher capacity rate of 16,667 t/d, which is equivalent to 6.0 Mt/a, and results in a mine life of approximately 15.3 years.

During the mining of pit ore, the peak mineralized material and waste production is capped at approximately 11.5 Mt/a, with an overall average production rate of 22,775 t/d. The average LOM stripping ratio is estimated to be 0.32:1 with 29.3 Mt of waste and 91.9 Mt of ore. Stripping ratios are very low, however during years 6 and 7 the strip ratios do rise to 0.86 and 0.75 respectively. The total tonnes of ore and waste that will be mined during the life of the Project are summarized in Table 16.3.

Material types were defined to differentiate the material that would be sent to the crusher and the waste facilities. Reserves were categorized using oxide, mixed oxide, mixed sulfide, and fresh material categories. The material types included codes to differentiate Probable material as well as Inferred material. All Inferred material was considered to be waste and sent to the waste storage facility.

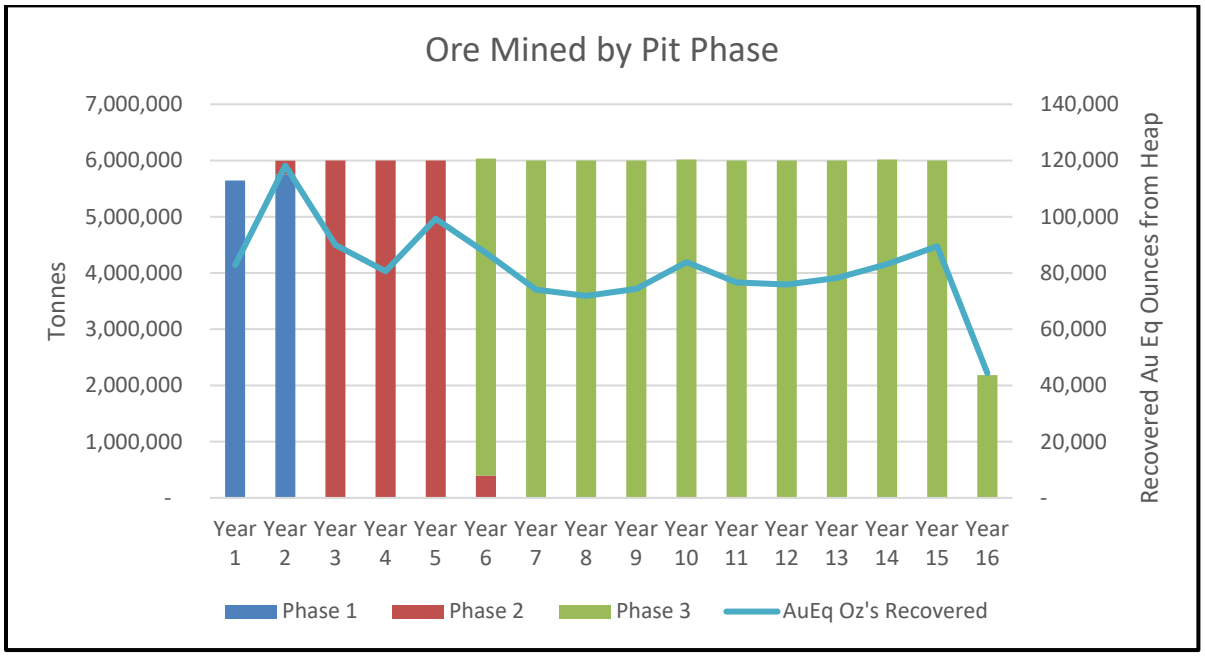
The mine scheduled in phases to provide the highest valued ore to the leach pad in the early years. The annual ore production schedule by pit phase is shown in Figure 16.6. Figure 16.7 shows the annual waste production schedule by pit phase. The amount of waste that has to be stripped rises significantly to 5.2Mt during year 6 for the stripping of the Phase 3 pit. The WRSF designs are of sufficient size for disposal of the waste material defined in the LOM plan.

A small stockpile may be maintained near the crusher so that trucks may dump and return to mining operations in the event of unexpected crusher down time. The balance will be minimized as much as possible during operations. The maximum size on an annual basis is estimated to be 163,000 tonnes.

Table 16.3: Annual Mine Production Schedule Forecast

Item	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	LOM
MINE PRODUCTION																		
Oxide Ore Mined	kt	3,174	1,184	1,573	809	457	1,145	642	130	67	17	0	0	0	0	0	0	9,198
Au grade	g/t	0.64	0.53	0.34	0.21	0.78	0.28	0.28	0.15	0.08	0.09	1.02	0	0	0	0	0	0.46
Ag Garde	g/t	12.94	19.49	18.46	30.24	19.31	18.68	24.86	30.36	24.84	19.76	7.76	0	0	0	0	0	18.46
Cu Grade	%	0.05	0.06	0.07	0.12	0.11	0.12	0.14	0.18	0.2	0.22	0.1	0	0	0	0	0	0.08
Mixed Oxide Ore Mined	kt	131	310	40	47	22	140	386	523	470	462	464	469	366	242	229	110	4,411
Au grade	g/t	0.9	1.19	0.25	0.22	0.62	0.25	0.22	0.24	0.28	0.28	0.28	0.27	0.33	0.58	0.97	0.58	0.42
Ag Garde	g/t	12.74	11.86	13.59	9.56	23.29	20.36	12.58	11.11	11.64	10.45	10.86	9.91	9.93	6.4	5.98	4.44	10.74
Cu Grade	%	0.06	0.06	0.1	0.09	0.13	0.08	0.09	0.09	0.09	0.09	0.09	0.1	0.1	0.11	0.15	0.12	0.09
Mixed Sulfide Ore Mined	kt	2,143	3,695	3,025	2,799	2,920	3,545	3,559	3,038	2,501	2,166	1,879	1,660	1,801	1,941	1,743	346	38,761
Au grade	g/t	0.79	0.89	0.44	0.31	0.63	0.42	0.33	0.32	0.38	0.42	0.45	0.47	0.52	0.58	0.64	0.66	0.5
Ag Garde	g/t	13.06	12.02	17.59	23.95	20.78	15.11	15.61	14.06	13.89	13.32	13.66	12.89	12.56	12.97	13.22	14.37	15.26
Cu Grade	%	0.07	0.07	0.09	0.12	0.1	0.1	0.11	0.11	0.11	0.1	0.09	0.09	0.09	0.09	0.09	0.1	0.1
Sulfide Ore Mined	kt	198	808	1,361	2,345	2,602	1,205	1,413	2,309	2,962	3,372	3,657	3,871	3,834	3,833	4,028	1,728	39,525
Au grade	g/t	0.58	0.73	0.51	0.54	0.63	0.55	0.48	0.47	0.46	0.47	0.48	0.49	0.5	0.53	0.6	0.7	0.53
Ag Garde	g/t	8.57	14.22	15.74	18.83	21.8	14.42	15.35	17.59	17.49	16.88	15.74	14.29	13.11	12.11	10.73	9.45	15
Cu Grade	%	0.05	0.08	0.09	0.13	0.11	0.11	0.11	0.12	0.12	0.13	0.12	0.11	0.11	0.11	0.11	0.13	0.12
Total Ore Mined	***kt***	***5,645***	***5,997***	***6,000***	***6,000***	***6,000***	***6,035***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***2,184***	***91,895***
Au grade	***g/t***	***0.7***	***0.81***	***0.43***	***0.39***	***0.64***	***0.41***	***0.35***	***0.37***	***0.41***	***0.44***	***0.45***	***0.47***	***0.5***	***0.55***	***0.62***	***0.68***	***0.51***
Ag Garde	***g/t***	***12.83***	***13.78***	***17.37***	***22.68***	***21.12***	***15.77***	***16.34***	***15.51***	***15.61***	***15.11***	***14.71***	***13.56***	***12.75***	***12.16***	***11.27***	***9.98***	***15.25***
Cu Grade	***%***	***0.06***	***0.07***	***0.09***	***0.12***	***0.11***	***0.1***	***0.11***	***0.11***	***0.11***	***0.11***	***0.11***	***0.11***	***0.11***	***0.11***	***0.11***	***0.12***	***0.1***
Total Waste	***kt***	***474***	***266***	***1,187***	***926***	***494***	***5,182***	***4,480***	***3,712***	***3,110***	***2,563***	***2,045***	***1,636***	***1,241***	***1,071***	***830***	***87***	***29,305***
Total Mined	***kt***	***6,119***	***6,263***	***7,187***	***6,926***	***6,494***	***11,217***	***10,480***	***9,712***	***9,110***	***8,579***	***8,045***	***7,636***	***7,241***	***7,088***	***6,830***	***2,271***	***121,200***
Strip ratio	ratio	0.08	0.04	0.2	0.15	0.08	0.86	0.75	0.62	0.52	0.43	0.34	0.27	0.21	0.18	0.14	0.04	0.32
Other Tonnes	kt	189	199	180	180	180	305	180	180	180	180	180	180	180	180	180	217	3,070
Total Moved	**kt**	***6,308***	***6,462***	***7,367***	***7,106***	***6,674***	***11,522***	***10,660***	***9,892***	***9,290***	***8,759***	***8,225***	***7,816***	***7,421***	***7,268***	***7,010***	***2,489***	***124,270***
PROCESS PRODUCTION																		
Ore to Heap	**kt**	***5,607***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***2,222***	***91,895***
Au grade	g/t	0.71	0.81	0.43	0.39	0.64	0.41	0.35	0.37	0.41	0.44	0.45	0.47	0.5	0.55	0.62	0.68	0.51
Ag Garde	g/t	12.79	13.8	17.37	22.68	21.12	15.78	16.34	15.51	15.61	15.11	14.71	13.56	12.75	12.16	11.27	10.07	15.25
Cu Grade	%	0.06	0.07	0.09	0.12	0.11	0.1	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.12	0.1

Figure 16.6: Annual Ore Schedule Forecast



Note: Figure prepared by Hard Rock Consulting, 2025.

Figure 16.7: Annual Waste Schedule

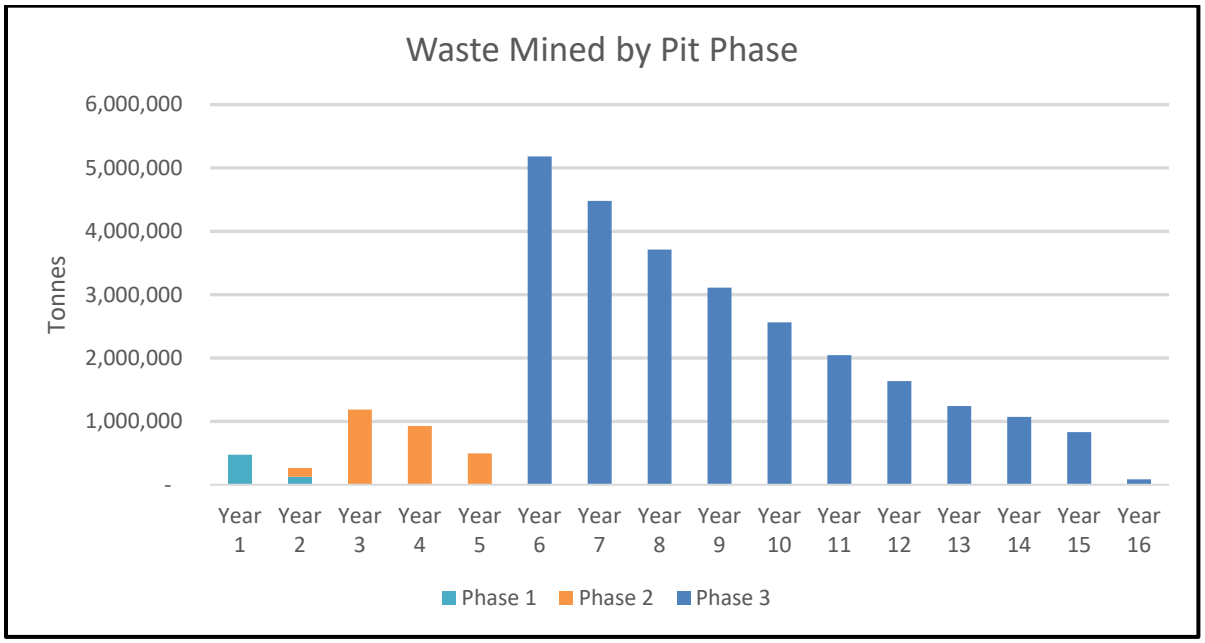


Figure prepared by Hard Rock Consulting, 2025.

16.7 Mine Plan Sequence

The mining of the phase 1 pit will take two years with the stripping of the second pit phase beginning during the last month if year 2. The second phase will last until the first month of year two at with time the third and final phase will begin. The yearly layouts for the mine and WRSF are shown in Figure 16.8 through Figure 16.20. The yearly layouts were used to develop haulage profiles for the LOM plan and also provided to mining contractors for cost proposals.

Figure 16.8: End of Year 1 Pit Topography

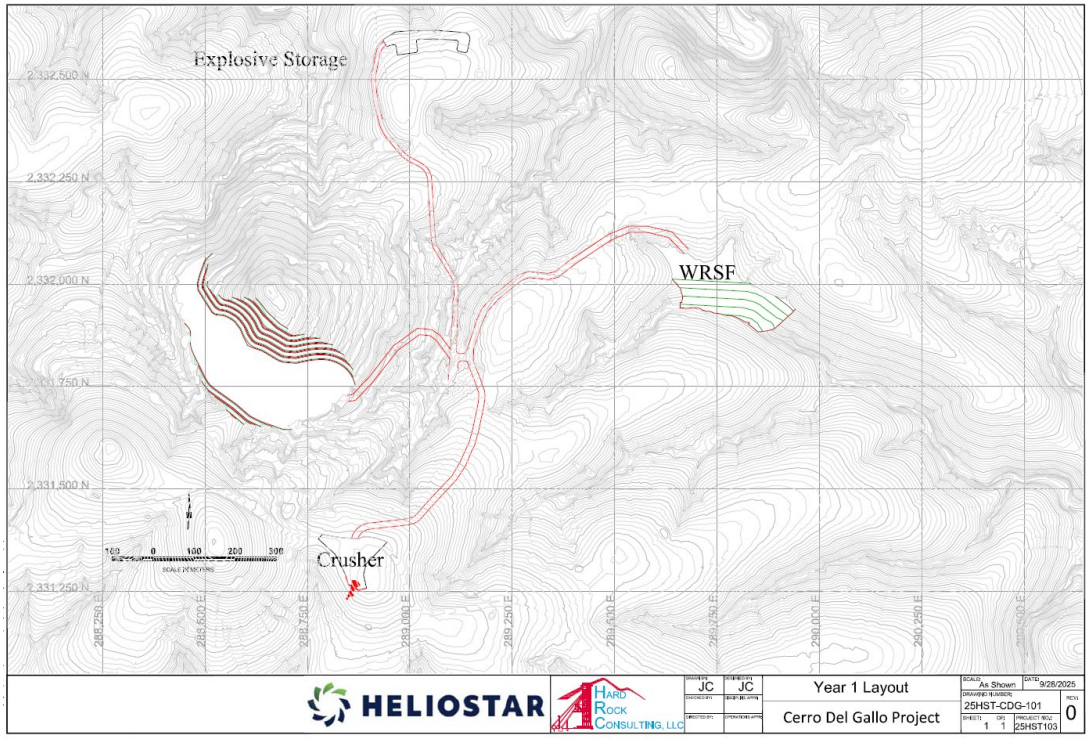


Figure prepared by Hard Rock Consulting, 2025

Figure 16.9: End of Year 2 Pit Topography

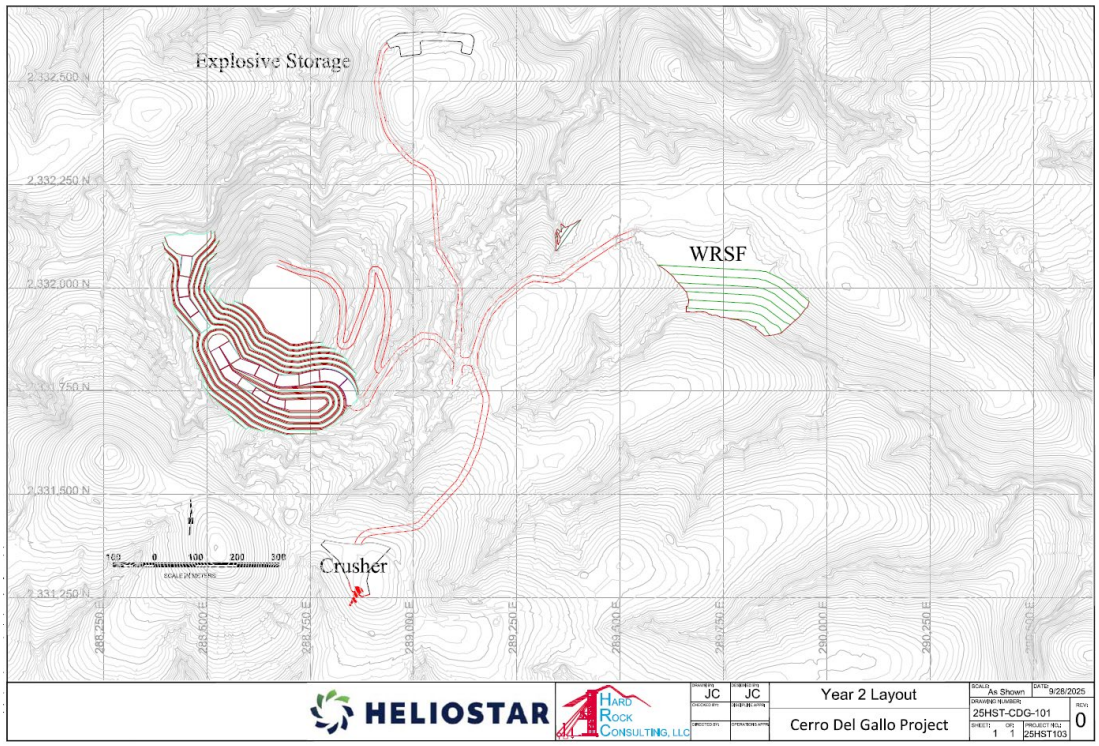


Figure prepared by Hard Rock Consulting, 2025

Figure 16.10: End of Year 3 Pit Topography

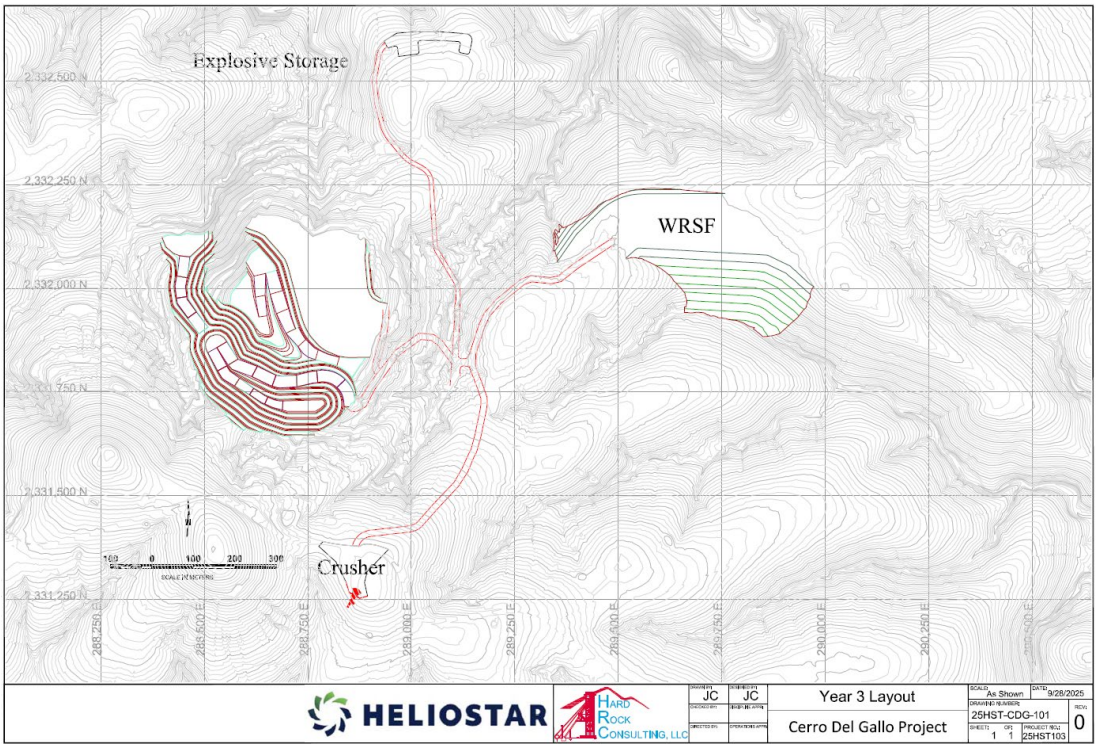


Figure prepared by Hard Rock Consulting, 2025

Figure 16.11: End of Year 4 Pit Topography

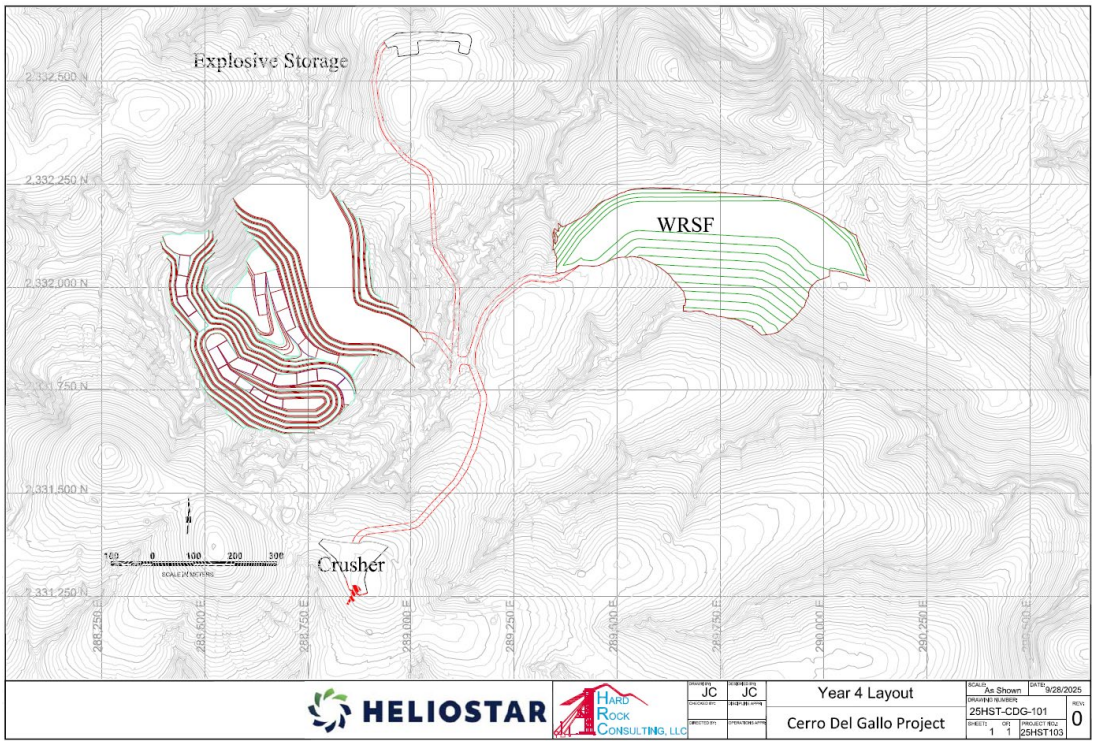


Figure prepared by Hard Rock Consulting, 2025

Figure 16.12: End of Year 5 Pit Topography

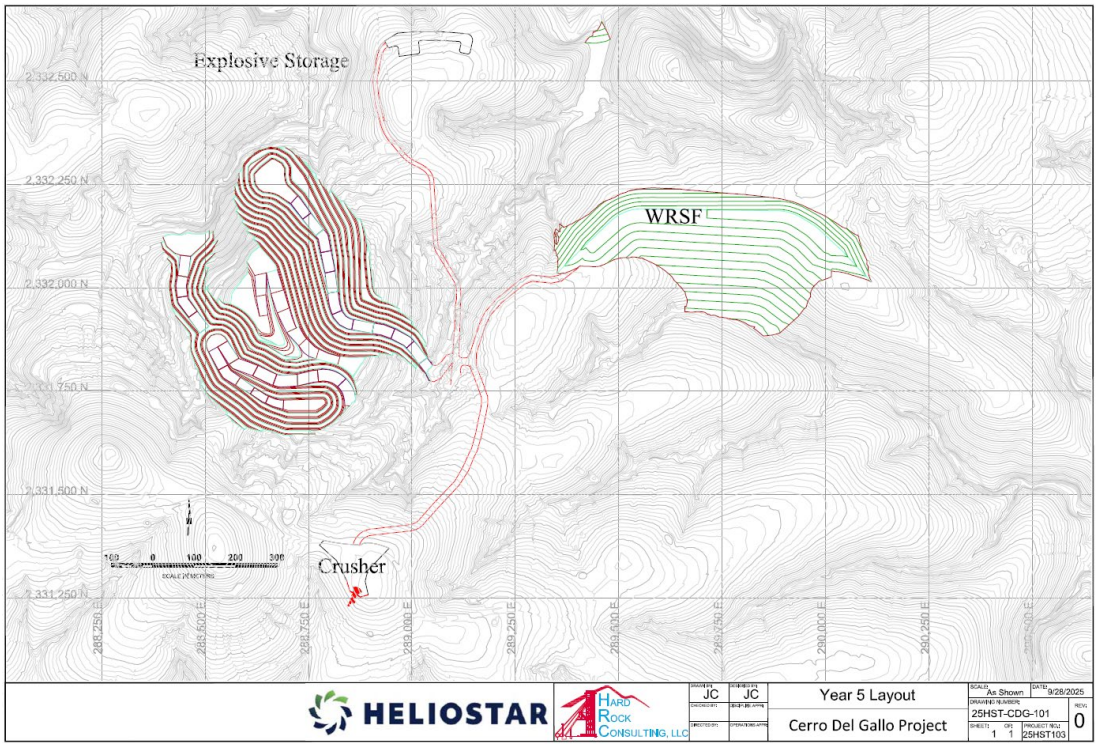


Figure prepared by Hard Rock Consulting, 2025

Figure 16.13: End of Year 6 Pit Topography

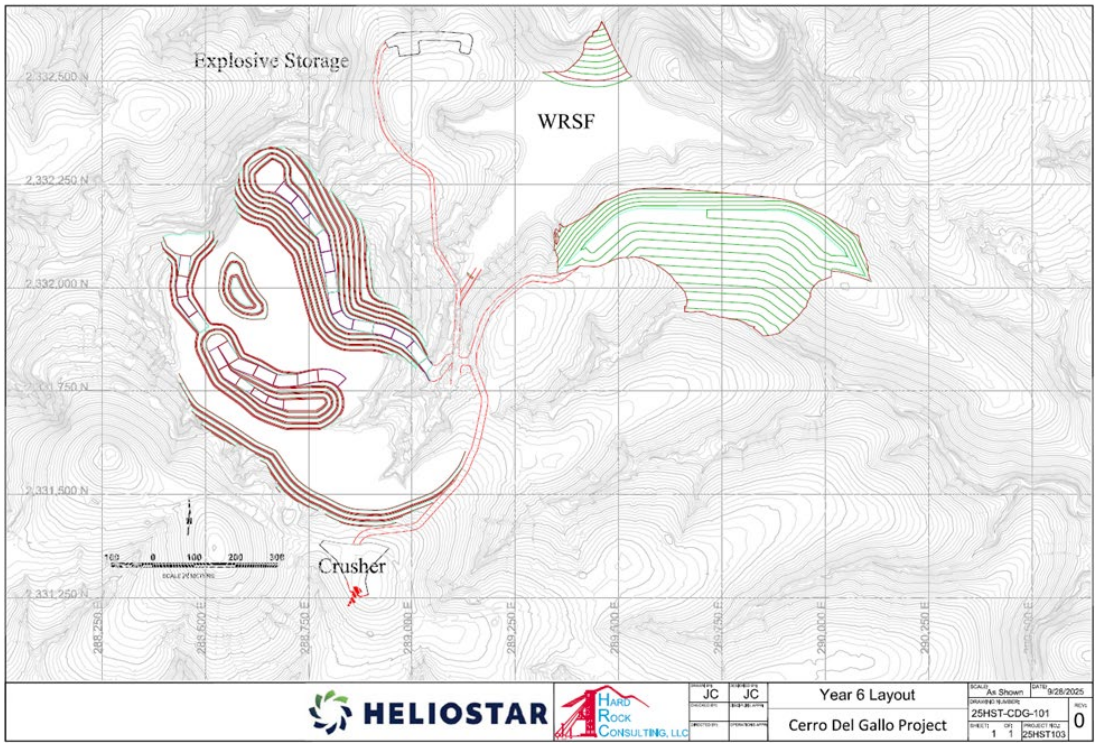


Figure prepared by Hard Rock Consulting, 2025

Figure 16.14: End of Year 7 Pit Topography

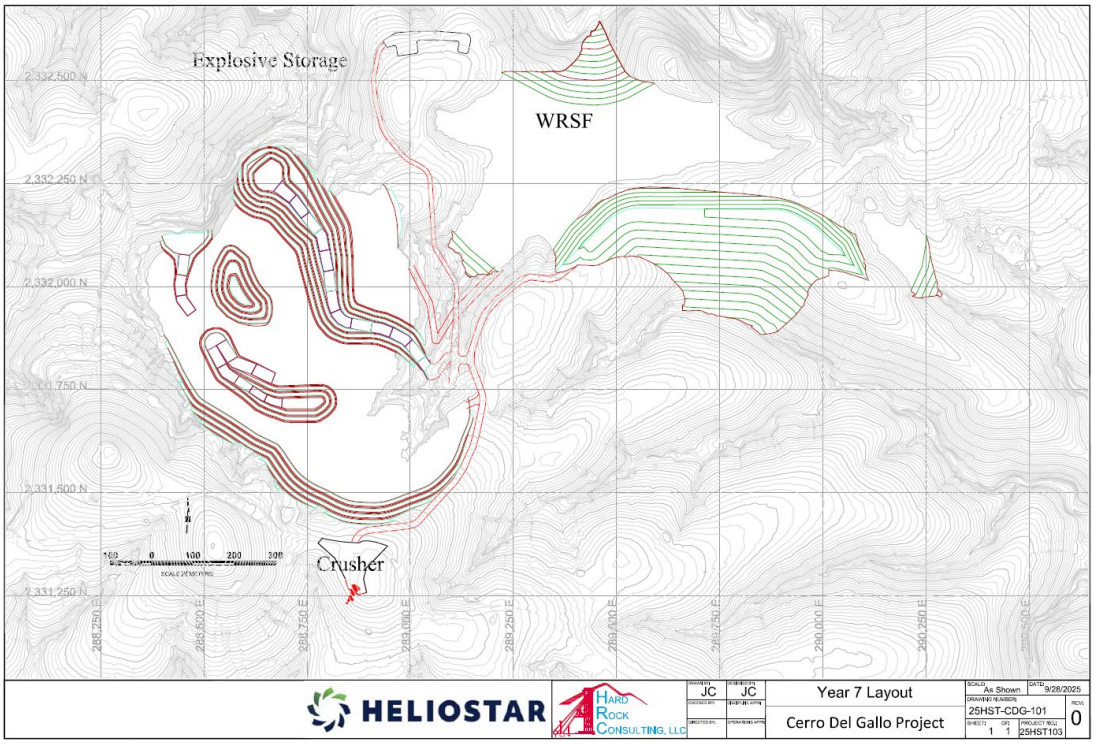


Figure prepared by Hard Rock Consulting, 2025

Figure 16.15: End of Year 8 Pit Topography

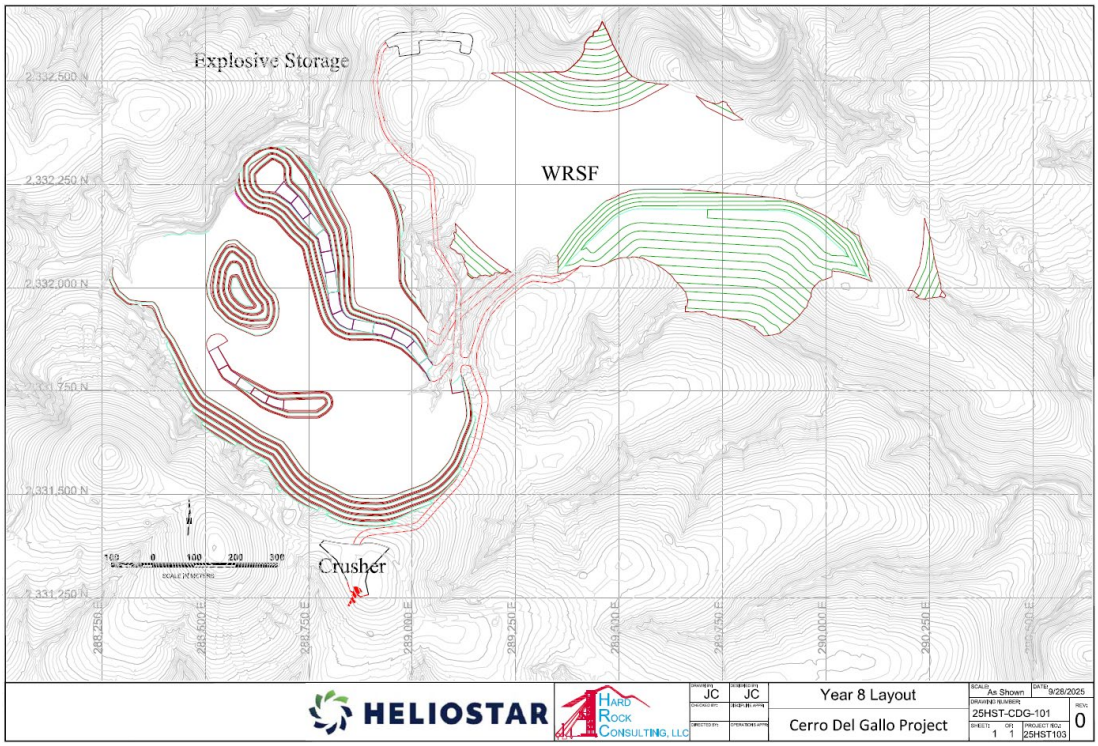


Figure prepared by Hard Rock Consulting, 2025

Figure 16.16: End of Year 9 Pit Topography

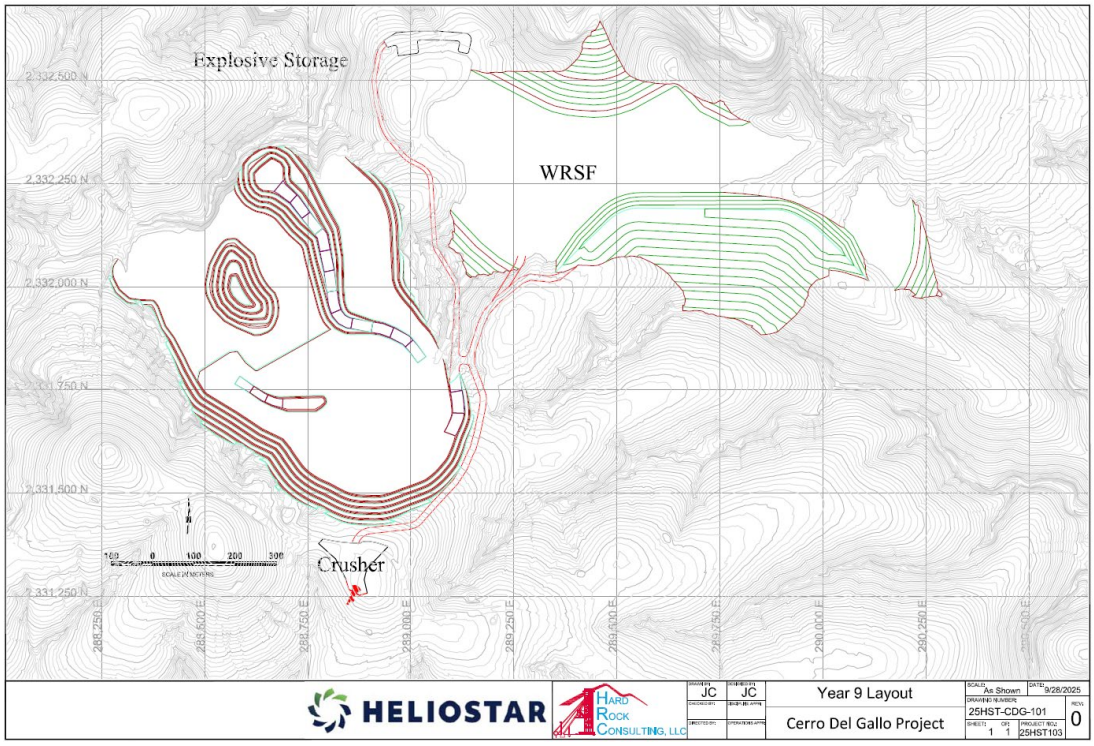


Figure prepared by Hard Rock Consulting, 2025

Figure 16.17: End of Year 10 Pit Topography

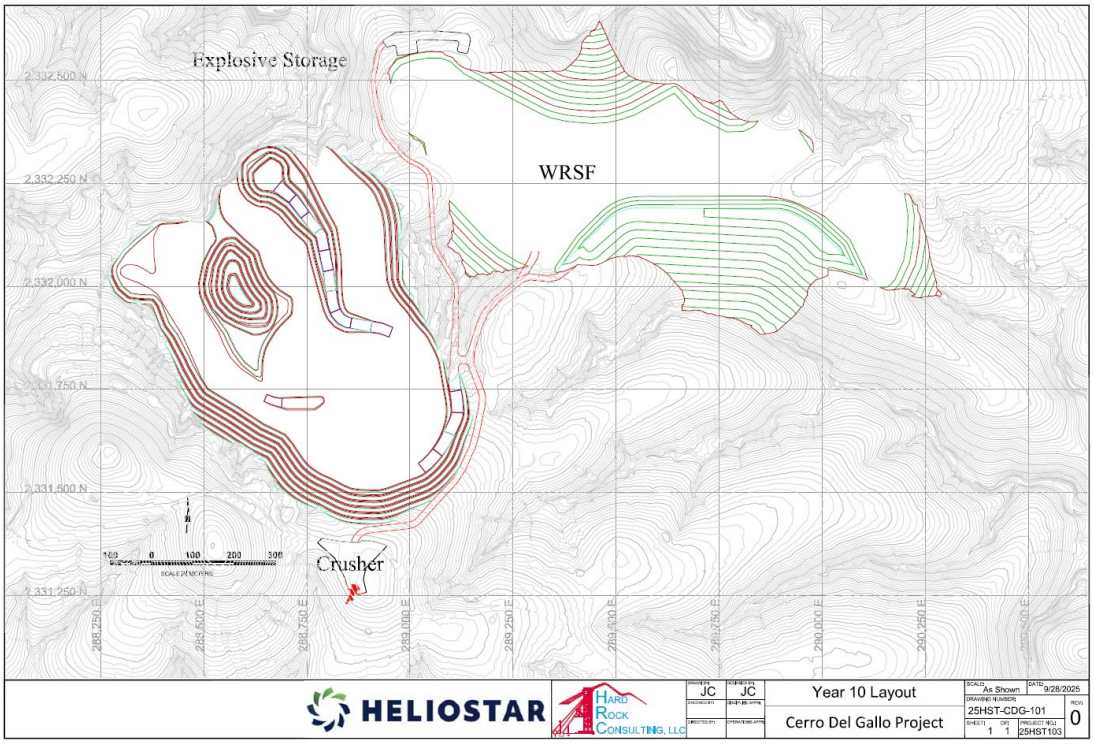


Figure prepared by Hard Rock Consulting, 2025

Figure 16.18: End of Year 12 Pit Topography

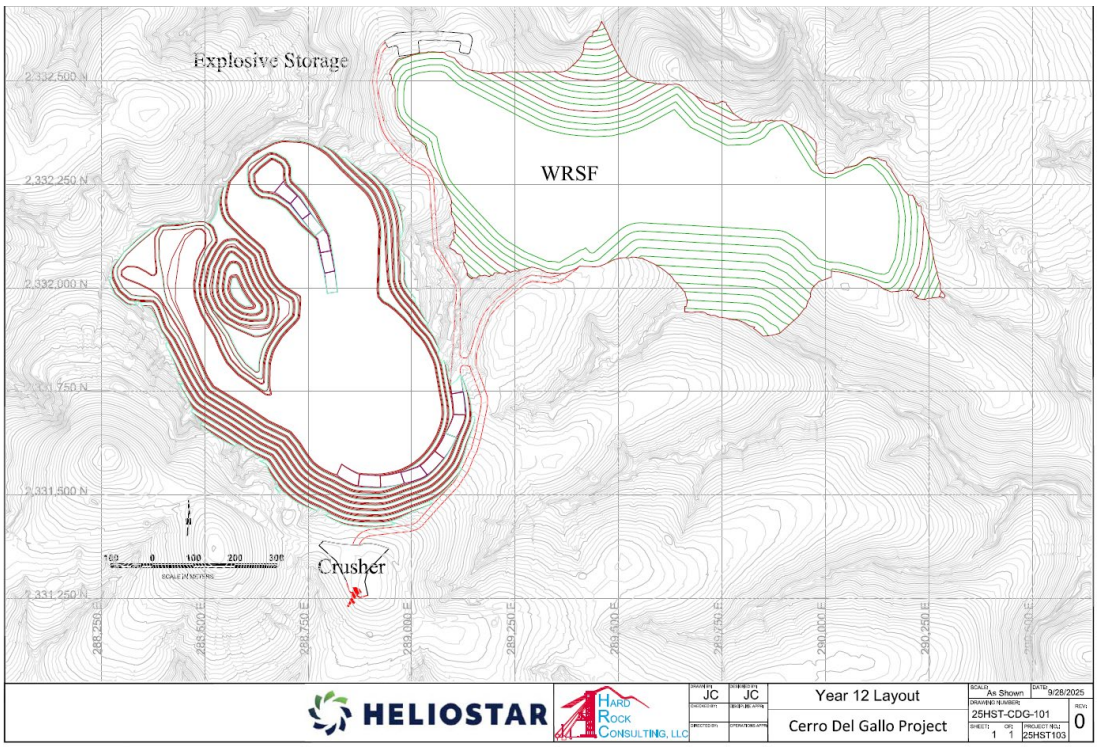


Figure prepared by Hard Rock Consulting, 2025

Figure 16.19: End of Year 15 Pit Topography

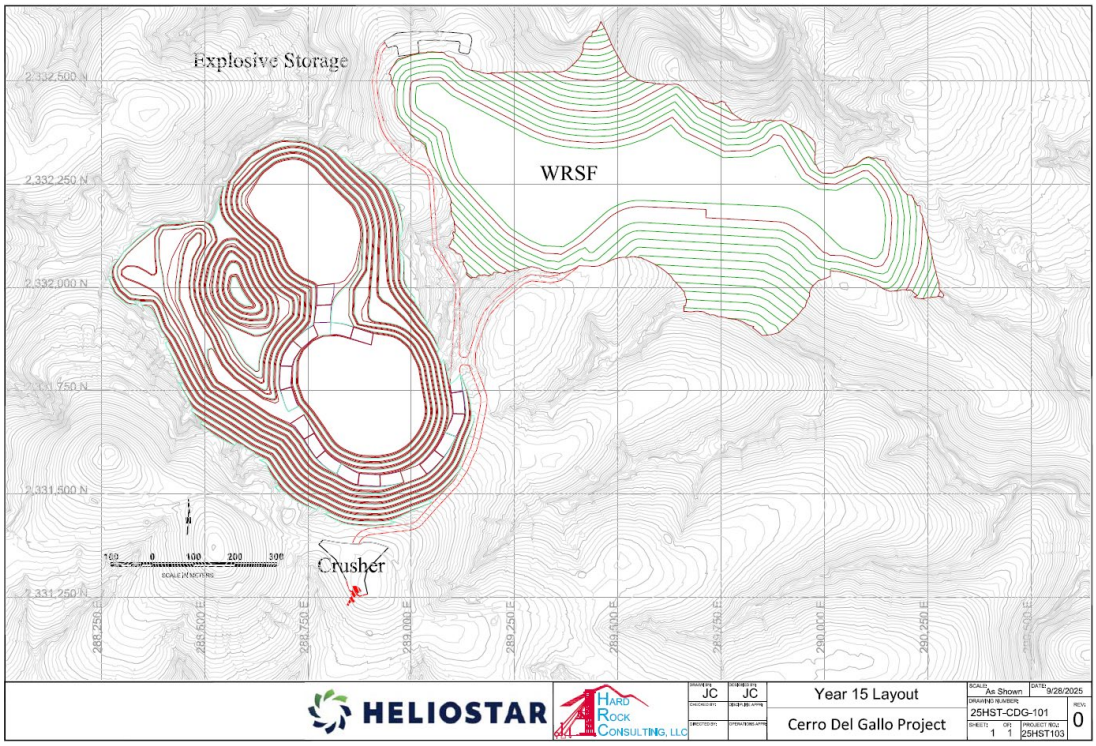


Figure prepared by Hard Rock Consulting, 2025

Figure 16.20: End of Year 17 Pit Topography

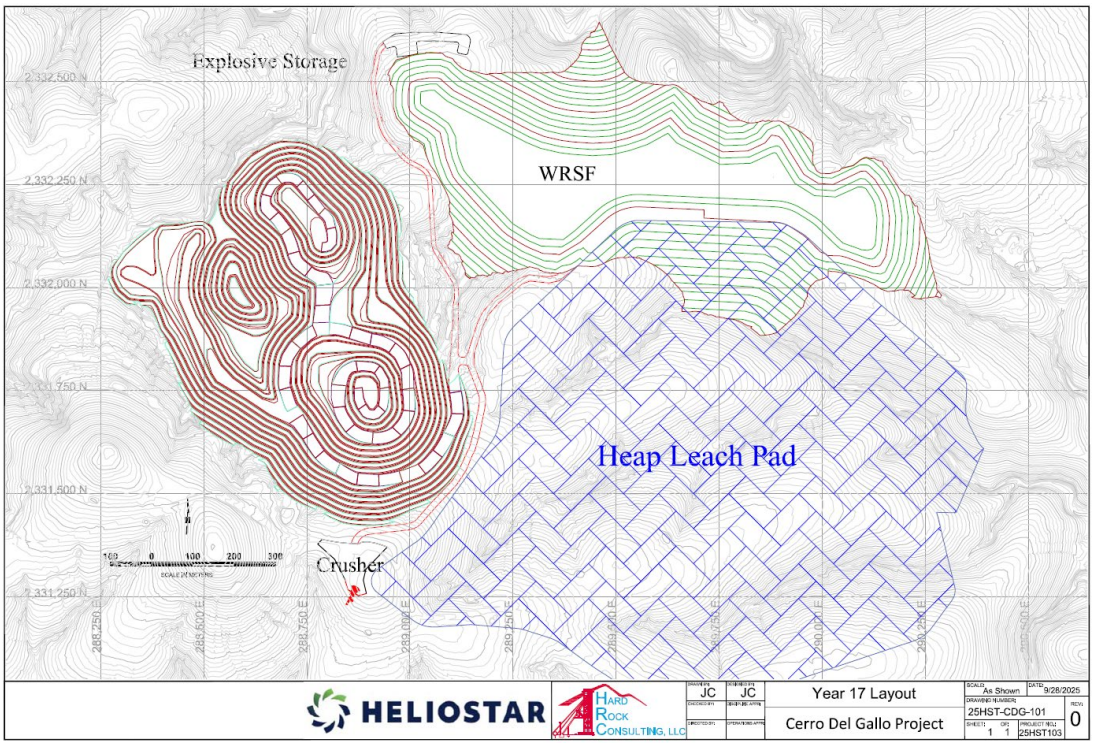


Figure prepared by Hard Rock Consulting, 2025

16.8 Mining Equipment

The majority of the mining equipment is planned to be supplied by a mining contractor. All loading, hauling, drilling, basting and support services are planned to be included within the mining contract. Mine production assumes the use of a 13 cubic meter front-end loader and 91-tonne haul trucks, but final equipment selection will be determined by the contractor.

Productivity estimates for primary mining equipment, including trucks, loaders, and drills, is based on a mining schedule of 24 hours per day with two shifts of 12 hours each. Standby time per shift assumes 0.5 hours for shift startup and shutdown, 0.50 hours for lunch breaks, 0.75 hours of other blasting and other miscellaneous breaks. Based on a 24-hour day, this is a mine schedule efficiency rate of 85%. This is used to determine the potential available operating hours per period which is about 7,483 hours per year. These hours are used along with the number of pieces of equipment and availability to determine available operating hours.

Drilling requirements were based on pattern size, drilling penetration rates, and non-drilling time. Blast patterns were assumed for production, trim-row, and pioneer drilling. The haul profiles are calculated on a monthly basis and form the basis for the truck and loader fleet requirements for the LOM plan shown in Table 16.4. The haul profiles are also used to calculate the required diesel fuel requirements which is assumed to be supplied by the Owner along with the explosives.

Table 16.4: Mine Equipment

Item	Year 1	Year 2	Year 3	Year 4	Year 4	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Production Equipment																
97 t trucks	3	3	3	3	3	5	5	5	5	4	4	4	4	4	5	5
12 m^3 loaders	1	1	1	1	1	2	2	2	2	1	1	1	1	1	1	1
DM45 size drills	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
Pre-shear drills	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Support Equipment																
Cat D8 dozer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Cat D9 dozer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
16 ft grader	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Water truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Source: Hard Rock Consulting, 2025

16.9 Mine Waste Facilities

For the PFS and based on the waste rock quantities and maximum footprint area provided by others, Golder developed a WRD design in its 2019 Study referenced in Section 27 and summarized as follows. Due to the potential for acid waste rock drainage as described in Section 20, three geomembrane lined contact water ponds are designed at the base of the three natural drainages within the area of the WRD. The contact water ponds have been designed to collect seepage water from the Underdrains as well as surface storm water runoff. The ponds have been sized to store the average precipitation events and discharge the surface water from larger storm events. The contact water ponds will provide a location to monitor water quality and pump the water to the HLF, if appropriate. The WRD will be reclaimed progressively during operations to limit the amount of contact water that will need to be managed during operations. The initial waste that is extracted from the pit will be placed below the Phase 1 and 4 leach pad footprints as structural fill and select waste rock, thereafter the waste will be placed in the western drainage of the WRD. After the western drainage of the WRD is full to capacity, the WRD surface will be regraded to an overall slope of 3H:1V that will cover the contact water pond and the surface will receive a cover system to limit the infiltration as needed to meet the environmental compliance criteria. The same process will be conducted for the remaining two drainages that the WRD is sited in. Therefore, typically only one contact water pond will be active at one time during operations. The contact water ponds have been sized to contain (below the spillway) the 1 in 2-year, 24-hour precipitation event falling on the contributing WRD and undisturbed ground areas. The overflow spillways have been designed to convey the 1 in 20-year, 24-hour storm event precipitation event falling on the same areas.

Storm water diversion channels around the perimeter of the WRD have been designed to protect the waste rock surface from erosion. The diversion channels are sized to contain the runoff from upstream basins resulting from the 1 in 100-year, 24-hour storm event. The diversion channels around the WRD are designed to convey this runoff in riprap-lined diversion channels. Sediment control structures are designed in the three drainages downstream of waste to control sediment from runoff conveyed in diversion channels and underdrain flows. Temporary diversion channels will need to be constructed within the WRD footprint to divert non-contact water prior to contact with mine waste. The size and location of the temporary diversion channels will be dependent on the annual development of the WRD.

The waste dump is designed with a capacity of approximately 55.0 million tonnes using a dry waste rock density of 1.9 t/m3. Although the estimated waste rock generated during the life of mine is approximately 2.8 million tonnes more than the WRD capacity, approximately 3.4 million tonnes of waste rock will be used for leach pad construction.

Therefore, the WRD provides some excess capacity if required. Table 16.5 provides a summary of the waste produced during the mine life and the volume of waste used for leach pad construction.

Table 16.5: Waste Placement Volumes

	Total Waste (Mt)	Waste Volume (Mm^3)	To HLF Construction (Mm^3)	To WRD (Mm^3)
Total	57.8	30.4	1.8	28.6

Source: PEA 2020

The WRD will be built in lifts to ensure overall geotechnical stability. The WRD is designed to meet minimum slope stability factor of safety criteria of 1.50 static and 1.05 pseudo static. Golder provided the geotechnical design specifications for the waste dump, which are summarized in Table 16.6.

Table 16.6: Waste Placement Volumes

Parameter	Description
Lift Height	30 meters
Total Dump Height	100 meters
Dump Crest Elevation	2240 meters aMSL
Lift Face Angle	1.5:1 (H:V)
Lift Bench Width	45 meters
Overall Dump Slope	3:1 (H:V) 18.4º

Source: PEA 2020

17 Recovery Methods

17.1 Summary

Test work results have indicated that the CdG ore is amenable to heap leaching for the recovery of gold, silver and copper.

The CdG ore is estimated to contain an average of 0.09% copper based on the mine plan used for this study. A portion of this copper is cyanide soluble and is expected to be extracted in the heap leach circuit. The cyanide soluble copper has an effect on the cyanide consumption. A SART plant that releases cyanide associated with the copper cyanide complex, allowing it to be recycled back to the leach process as free cyanide is included. The resulting copper and silver precipitate will be sold, bringing additional revenue to the project.

The ore will be mined by standard open-pit mining methods, fine crushed using a system incorporating cone and high-pressure grinding roll (HPGR) crushers, agglomerated with cement and conveyor stacked on the heap leach pad in 8-meter lifts.

The heap leach pad was designed by AECOM. The pad will be constructed in four phases and will hold approximately 92 million tonnes. The heap leach pad will have a composite liner consisting of clay/GCL and textured HDPE geomembrane.

Ore will be single-stage leached with a dilute cyanide solution at a high solution application rate for the first 40 days and a lower application rate for the remaining 80 days for a total leach cycle of 120 days. The gold, silver, and copper bearing solution will be collected in the pregnant solution pond and pumped to the SART plant. Pregnant solution will be acidified with sulfuric acid, then copper and silver will be precipitated as sulfides by the addition of sodium hydrosulfide. The precipitate will be thickened and filtered to produce a copper-silver filter cake for shipment to a smelter. The barren solution from the SART plant will be processed in a carbon adsorption-desorption-recovery (ADR) plant to recover gold. The gold will be periodically stripped from the carbon using a desorption process. The gold will be plated on stainless steel cathodes, removed by washing, filtered, dried and then smelted to produce a doré bar.

Engineering and design of the processing plant was undertaken for complete crushing, leaching, and recovery systems. The criteria for the design of the processing circuit are summarized in Table 17.1.

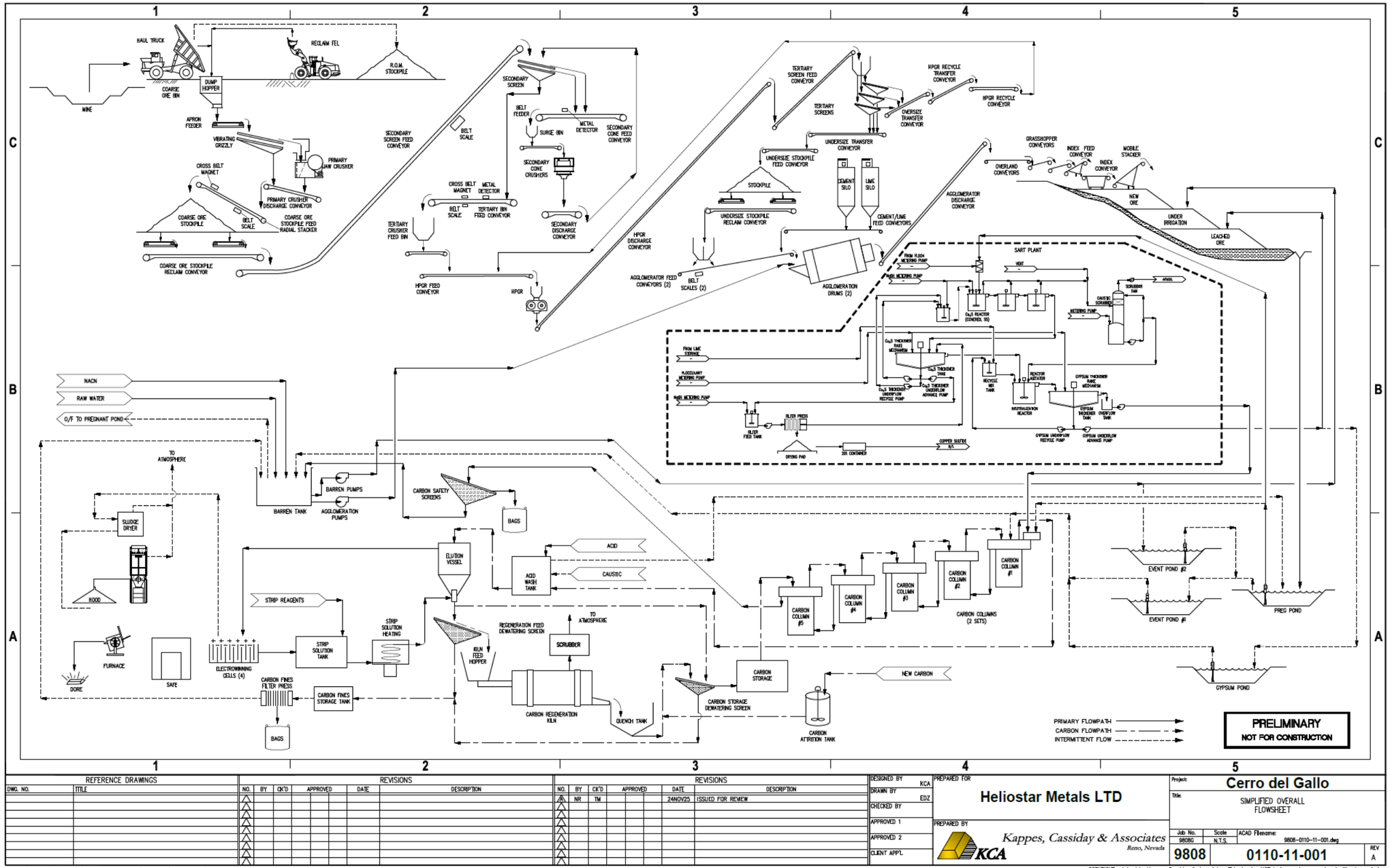
Table 17.1 Cerro del Gallo Process Design Criteria Summary

Parameter	Description
Operating days	360 days per year
Operating time	For the crushing, conveying, agglomeration, and stacking section of the plant, operating time is 75%
Note: All Tonnes Are Dry Tonnes	
Nominal Crushing Capacity	16,667 t/d (696 t/h)
Design Crushing Capacity	22,200 t/d (925 t/h)
Conveyor Design Factor	1.2
Design Conveying Capacity	26,700 t/d (1,110 t/h)
Crushing Product Size, mm	5 (80% passing)
Leach Cycle, days (total)	120
Barren Application Rate	
Phase 1 Leach (40 days), m^3/hr	635 (average) 700 (design)
Phase 2 Leach (80 days), m^3/hr	645 (average) 700 (design)
Total, m^3/hr	1,280 (average) 1,400 (design)
Barren Design Factor	1.1
Barren System Pumping Flow, m^3/hr	1,280 (average) 1,400 (design)
SART & ADR Feed Flow, m^3/hr	1,400 (design)

Source: KCA (2025)

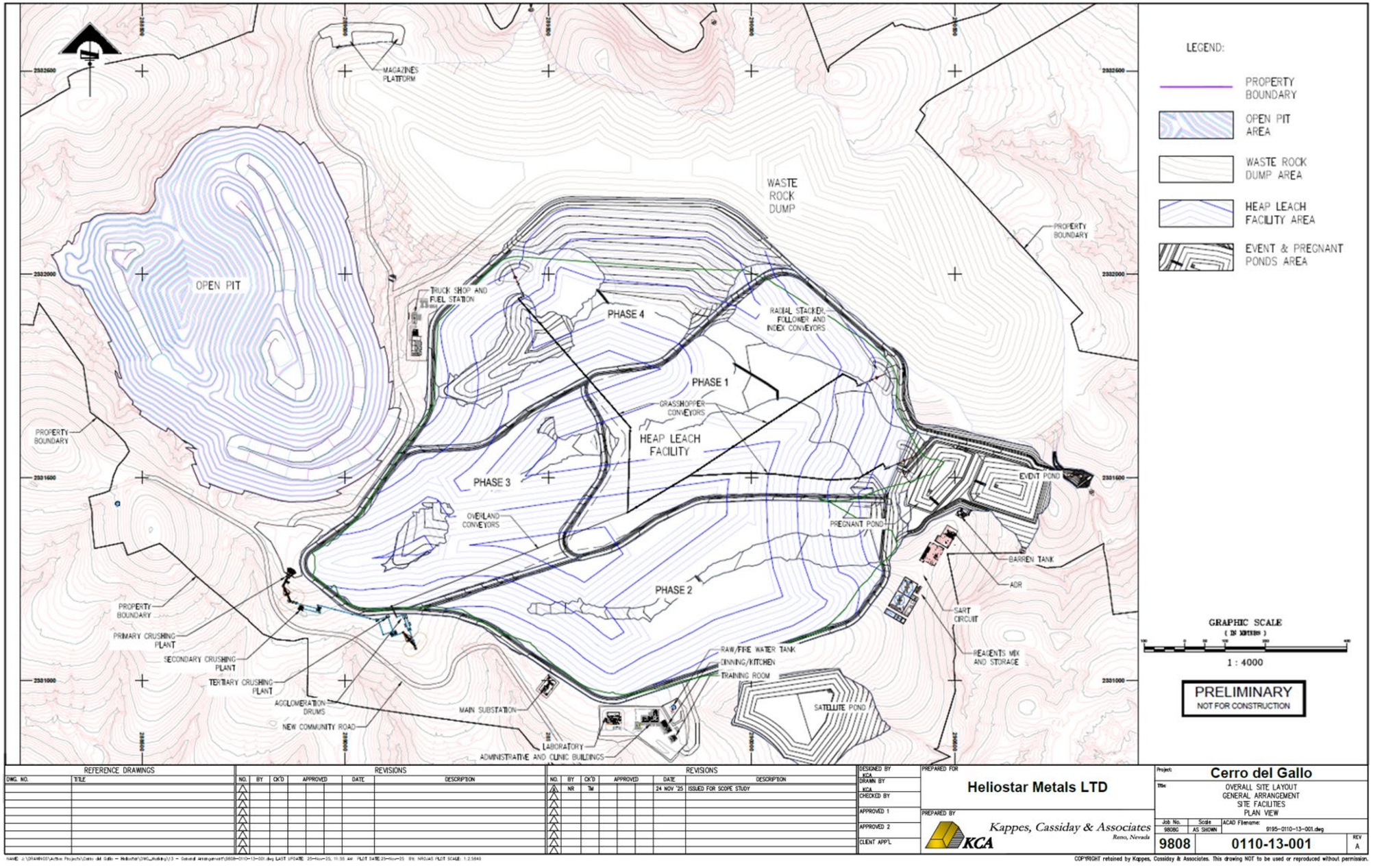
Figure 17.1 and Figure 17.2 present the overall process flowsheet and general arrangement of the mine site.

Figure 17.1: Cerro del Gallo Overall Process Flowsheet



Source: KCA (2025)

Figure 17.2: Cerro del Gallo General Arrangement Drawing



Source: KCA (2025)

17.2 Processing

17.2.1 Crushing

Crushing for the CdG project is accomplished by a three-stage crushing system with an open primary crushing circuit, closed secondary, and tertiary crushing circuits operating seven days per week, 24 hours per day. Run of mine (ROM) material will be delivered and direct dumped, as much as possible, by haul truck from the mine into the 200-tonne ROM feed bin. A permanent rock breaker will be installed to break any oversized material. ROM material from the coarse ore bin will be delivered by a vibrating grizzly feeder. Oversize material is crushed using a primary jaw crusher while the undersize material is combined with the primary jaw product on a primary crusher discharge conveyor. The primary jaw crusher is operated in open circuit and is designed to crush the vibrating grizzly oversize to 80% passing 145 mm. The primary crushing products is stockpiled by a coarse ore stockpile feed stacker conveyor.

Material from the primary crushed stockpile is reclaimed using reclaim feeders and is conveyed to the secondary screen feed conveyor. The secondary crushing circuit includes two double deck vibrating screens and two cone crushers. The secondary crushing circuit is operated in closed circuit with a product size of 80% passing 36 mm.

Primary crushed ore is combined with the secondary cone product and is fed to the secondary screen. The secondary screen oversize is transferred to the secondary cone crusher surge bin by conveyors and is fed to the secondary cone by a belt feeder. The secondary cone crusher discharge recycles back to the secondary screen.

Secondary screen undersize material is conveyed to the secondary product storage bin, reclaimed using belt feeders and is transferred to the tertiary crushing circuit. The tertiary crushing circuit consists of an HPGR crusher operated in closed circuit with a fine screening plant. The design for the final crushed product is 80% passing 4 to 6 mm.

Material from the secondary product storage bin is transferred to the HPGR crusher. The product from the HPGR is conveyed to a bin. The bin has multiple discharge points onto three triple-deck vibrating tertiary screens. The tertiary screen oversize is transferred to the HPGR recycle conveyor and recirculated to the HPGR feed conveyor. The tertiary screen undersize is stockpiled by the fine ore stacking conveyor.

Material from the fine ore stockpile is reclaimed using reclaim feeders and conveyed to the agglomeration system by the fine ore reclaim conveyor. The fine ore reclaim conveyor discharges to a splitting chute to feed two parallel agglomeration feed conveyors. Cement is added to the ore on the agglomeration feed conveyors from the cement silos,

depending on the material type. The cement addition rate is controlled by a weightometer mounted on the individual agglomeration feed conveyors. The agglomeration feed conveyors feed two parallel agglomeration drums where barren process solution is added and cement is blended in. The agglomeration drums discharge onto the agglomeration discharge conveyor and the material is transported to the stacking system.

17.2.2 Heap Conveying and Stacking

The heap leach will be constructed in eight-meter lifts using a mobile conveyor stacking system. The Phase 1 leach pad conveying and stacking system will consist of an overland conveyor, 23 mobile grasshopper conveyors, and index feed conveyor, a horizontal index conveyor and a radial stacker. The overland conveyor transfers the material from the agglomeration discharge conveyor to the mobile grasshopper conveyors, which feed the conveyor stacking system. As the radial stacker progresses, the system is periodically stopped to add or remove grasshopper conveyors as needed. Phases 2 through 4 will increase the leach area without any additional equipment required.

Stacked material will consist of crushed and agglomerated ore. Once a lift has finished leaching, and is sufficiently drained, a new lift can be stacked over the top of the old lift. The old lift will be cross-ripped with a dozer prior to stacking the new lift to break up any compacted ore and to redistribute material that may have been winnowed by the irrigation solution or rainfall. Stacked lifts will progress in a stair-step manner.

17.2.3 Heap Leaching

Following stacking, the material is irrigated with a dilute sodium cyanide barren leach solution and the resulting gold, silver and copper bearing solutions are collected into the pregnant solution pond. The CdG Project has been designed as a single pass system with no intermediate solution used for heap application. The heap will be irrigated using a drip-tube irrigation system for solution application. PVC pipes are used to distribute the solution to the drip-tubes on top of the heap. Antiscale agent is added to the suction of the barren and pregnant solution pumps to reduce the potential for scaling problems within the system.

The total leach cycle of 120 days has been designed for the heap leach system, which is based upon metallurgical test work completed to-date. The leach cycle is operated in two stages. The first stage of the leach cycle is 40 days and leach solutions will be applied to the ore at a nominal application rate of 12 L/h/m^2. The second stage of the leach cycle has a duration of 80 days and the leach solutions will be applied to the ore at a nominal application rate of 6 L/h/m^2. The leach solutions will have an approximate cyanide concentration of 1500 ppm when applied to the heap.

Two horizontal centrifugal pumps, operating in parallel at the barren tank will be used for the barren solution application to the heap.

The barren pumps will be mounted next to the barren tank along with process solution pumps and an agglomeration solution pump along the side of the tank. High-strength sodium cyanide solution and an antiscale agent will be added to the suction side of the barren leach solution pumps by metering pumps. The combined nominal flow to the heap is 1,280 m^3/h.

Gold, silver and copper bearing solutions draining from the leach pad are collected by a network of perforated drainage pipes that are directed to the pregnant solution pond. Pregnant solution is pumped from the pregnant solution pond by submersible pump to the SART circuit.

17.2.4 Heap Leach Facility Design

Ore from the CdG deposit will be processed by heap leaching. A single heap leach facility has been designed for the site by AECOM. The Heap Leach Facility (HLF) has an ore capacity of approximately 92 million tonnes (Mt) using a dry ore density of 1.6 tonne/m^3 (t/m^3). The HLF design is described in detail in the Golder's report “Cerro Del Gallo PFS – Heap Leach Facility Design” (Golder 2019).

The Pre-Feasibility design of the leach pad meets or exceed North American standards, such as the Nevada Administration Code 445A.434 for the lining system design and Mexican Standard NOM-155-SEMERNAT-2007 for the stability of the HLF, which are intended to lessen the risk of environmental impact to the local soils, surface water, and ground water. North American construction standards are intended to mitigate environmental impacts to surface and subsurface water sources. Actual standards used in subsequent stages should be carefully considered and implemented to ensure that environmental impacts are mitigated to the extent required under prevailing laws, regulations and international standards.

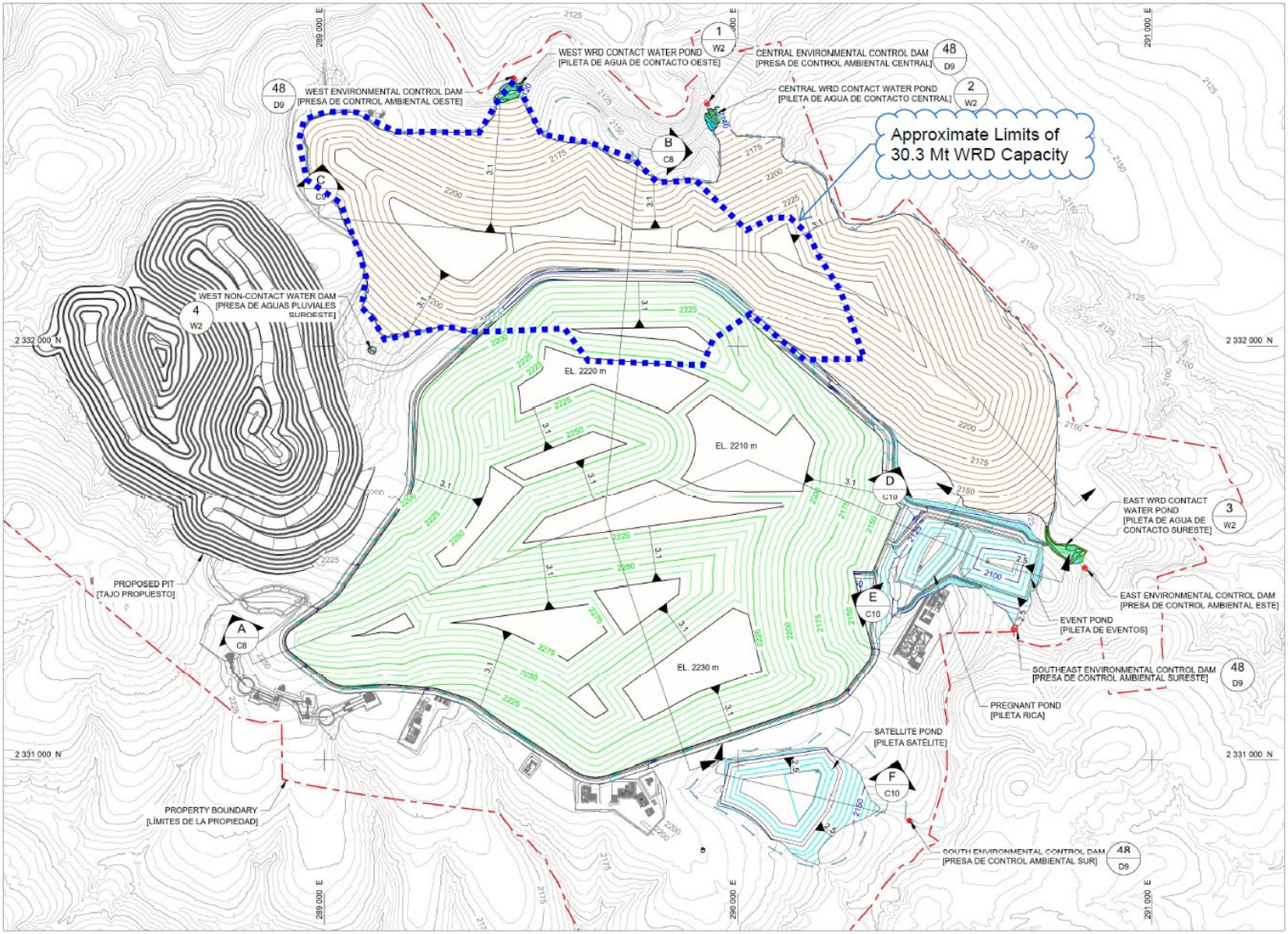
The 139.8-hectare HLF has a maximum heap height of 80 meters. Ore is designed to be stacked at a rate of 16,667 tonnes per day (tpd). Once production crushers are operational, ore will be crushed and agglomerated, then placed on the leach pad using portable conveyors feeding a conveyor-stacker. AECOM was informed that the ore is intended to be stacked in approximately 8-meter lifts with benches provided between lifts to create an average overall ore slope of 3H:1V (horizontal to vertical), which is necessary to provide geotechnical stability and help reduce grading during reclamation.

The foundation of the HLF consists of an underdrain system within the natural drainages to capture spring water and transport it to the toe of the process pond fill. Fill material needed for perimeter roads, process ponds, and leach pad foundations are intended to be sourced from waste rock mined from the pit. The leach pad is a geomembrane lined pad that is divided into four separate construction phases. Each phase is constructed with a solution collection system that drains by gravity to the Pregnant Pond at the toe of the Phase 1 leach pad. During upset conditions, the Pregnant Pond overflows into an Event Pond located downstream of the Pregnant Pond. During wetter than average climate conditions described in Section 5, excess seasonal solution accumulation will be pumped to a Satellite Pond located within the drainage south of the Phase 2 leach pad expansion.

The HLF is designed in four phases providing a total lined leach pad surface area of approximately 1.2 million square meters, as shown in **Error! Reference source not found.**.

Phase 1 consists of constructing the eastern portion of the leach pad, perimeter access road, underdrain system, pad geomembrane lining system, leak detection system, solution collection system, permanent and temporary stormwater diversion facilities, the Pregnant Pond and the Event Pond. Phase 2 consists of constructing the southern portion of the leach pad, perimeter access road, underdrain system, pad geomembrane liner system, leak detection system, solution collection system, and the Satellite Pond. Phase 3 consists of constructing the western portion of the leach pad, perimeter access road, underdrain system, pad geomembrane liner system, leak detection system, and solution collection system. Phase 4 consists of constructing the northern portion of the leach pad, perimeter access road, underdrain system, pad geomembrane liner system, leak detection system, and solution collection system.

Figure 17.3: HLF and WRD Layout



Source: AECOM (2025)

This is a summary of materials used to construct the leach pad according to the standards described above. The leach pad is designed with a grading plan to meet minimum slope stability factor of safety criteria of 1.50 static and 1.05 pseudo-static and to promote positive gravity solution flow in the solution collection system above the leach pad liner. In general, this consists of localized grading along the lower portions of the pad to achieve a minimum design grade of 2%. Grading also includes general shaping of the leach pad site to provide smooth surfaces with local slopes no steeper than 2.5H:1V in preparation for liner system placement. The existing incised ravines that pass through the leach pad site are designed with an underdrain system extending beneath the leach pad liner and ponds consisting of a non-woven geotextile surrounding a perforated 300-mm diameter corrugated polyethylene (CPE) pipe bedded in drain gravel. The underdrain system is designed to capture seepage through the overlying leach pad structural fill as well as natural springs beneath the foundation. The ravine drains discharge into the Underdrain Monitoring Sump at the base of the leach pad and ponds. Phase 1 is located at the convergence of two natural drainages that make up the Phase 3 and 4 leach pads, therefore a temporary collection system is designed upstream of Phase 1, at the low spot of each natural drainage. The collection system ties into a solid wall 450 mm diameter HDPE DR11 pipe that discharges at the same location as the underdrains.

The leach pad is designed with a composite liner system consisting of (from top to bottom):

- A 700-mm thick drainage layer overliner containing a network of solution collection pipes described below;
- A 1.5-mm thick, single-sided textured, high density polyethylene (HDPE) geomembrane;
- A 300-mm thick compacted soil liner bedding with a permeability of no greater than 1 x 10^{-5} centimeters per second (cm/sec) or a geosynthetic clay liner (GCL) with a permeability of no greater than 5 x 10^{-9} cm/sec where allowed according to the stability evaluation/criteria; and
- Prepared subgrade.

The Pregnant and Event Ponds utilize a similar composite lining system as the HLF with an additional secondary 1.5 mm HDPE geomembrane and geonet layers above the soil bedding layer. These additional layers provide a synthetic dual-containment and leak detection system. The Satellite Pond is not expected to receive any fluids during average climate conditions and was therefore designed with a single geomembrane lined system without the secondary geomembrane and geonet layers.

Material suitable for use as soil liner bedding appears to be sparse within the limits and immediate proximity of the HLF and WRD. Soil liner bedding may be either mine waste

that meets the material requirements for soil liner bedding or it may need to be imported from local off-site sources.

The drainage layer overliner material placed above the leach pad geomembrane is a free-draining crushed durable gravel with a minimum permeability of 1 x 10^{-1} cm/sec. The overliner material is placed on lined leach pad slopes flatter than 10%. Lined slopes steeper than 10% rely on the ore to convey the solution into the solution collection pipes. The first lift of ore placed on the geomembrane liner needs to meet maximum particle size requirements and have a minimum permeability of 1 x 10^{-3} cm/sec. The minimum permeability requirement of the overliner is designed to prevent the maximum head on the liner exceeding 0.7 m. A small portable crusher operated by a contractor is planned to manufacture the overliner material by crushing and processing durable mine waste rock that has been mined from the mine pit, or durable rock developed through on-site excavation within the footprint of the leach pad or process facilities.

The leach pad is divided into solution collection cells using geomembrane-lined cell separation berms. During leaching of the ore, solution is collected above the composite liner system by a network of perforated collection pipes within the drainage layer overliner material. The perforated solution collection piping network consists of 100 to 450 mm diameter N-12 corrugated polyethylene (CPE). The cell outlet pipes from each cell's low spot consists of a 600 mm diameter HDPE solid wall pipe that conveys the leachate to the Pregnant Pond located at the down-gradient end of the leach pad. The pipe type and size are selected based on the expected amount of leachate solution and the expected maximum ore height that the pipe will experience.

Solution will be applied to each lift of ore placed on the leach pad at a weighted average rate of 8 liters per hour per square meter (L/hr/m^2) that consists of two leach cycles as described by KCA in Section 17.1. The first cycle is designed to be applied to the heap at 12 L/hr/m^2 for 40 days and the second cycle is designed to be applied to the heap at 6 L/hr/m^2 for 80 days. The leachate solution is planned to be pumped and applied to the ore at a maximum total volumetric flowrate of 1,400 m^3/hr. Given this solution application rate and the minimum permeability of the overliner, the collection pipe size and spacing at the base of the heap have been designed to maintain a maximum 700 mm hydraulic head on the leach pad liner system

During operations, a gypsum slurry will be produced as a by-product of the SART operations. To manage the gypsum slurry, a lined Gypsum Pond has been designed in the upper reaches of the eastern drainage of the WRD, as shown on Figure 17.3. Due to the potentially high concentration of cyanide in the gypsum slurry, the pond will consist of the same double liner system that has been designed for the Pregnant and Event Ponds.

The Gypsum Pond is designed with a capacity to contain 2 years of gypsum produced. After two years of pumping gypsum slurry into the pond from the SART plant, the pond will either be capped with a single geomembrane liner system, or the crest will be raised using downstream construction to increase the storage capacity. During the first 2 years of operation the gypsum slurry volume and geotechnical properties will be monitored to explore other disposal and storage options.

The natural topography within the leach pad ranges from about 2 percent to 50 percent grade. Substantial local grading is required for constructability and geotechnical stability including:

- A toe fill is required at the downhill end (eastern toe) of the leach pad to meet the minimum geotechnical factors-of-safety. The toe fill is designed with a maximum 2 percent grade sloping towards the west to promote drainage of the solution collection system above the geomembrane liner.
- A fill bench is required on the ridge between Phase 1 and 2, near the toe of the heap.

The leach pad is designed to have a maximum internal slope of no steeper than 2.5H:1V so typical construction equipment can be safely employed without additional controls.

Storm water diversion channels are sized to contain the runoff from upstream basins resulting from the 1 in 100-year, 24-hour storm event that is a typical industry standard. The diversion channels around the HLF and process ponds are designed to convey this runoff in riprap-lined diversion channels. Sediment control structures are designed in drainages downstream of the facility to control sediment from runoff conveyed in diversion channels and underdrain flows.

17.2.5 Process Water Balance

A pre-feasibility level evaluation of process water management for the HLF and WRD was developed. This evaluation included development of a deterministic water balance that accounts for inflows such as rain and leach solution, outflows such as evaporation and consumptive loss due to ore and waste rock wetting.

To estimate inflow and outflow water requirements, the following criteria were given by KCA unless otherwise stated:

- Average as-mined moisture content and specific moisture retention of the ore and waste rock;
- Nominal solution application rate to the leach pad is 8 l/hr/m²;

- Average and maximum solution flow rate to the leach pad is 1,280 m^3/hr and 1,400 m^3/hr, respectively;
- Solution will be applied with drip irrigation emitters;
- MTS has estimated the total waste rock produced during mining activities is approximately 33.3 Mt. The WRD will store 30.3 Mt of waste using a dry waste rock density of 1.9 tonnes/m^3. The remaining waste rock produced during mining (approximately 3 Mt) will be used for the construction of mine facilities, including HLF foundation;

To estimate inflow and outflow water requirements, the following criteria were assumed:

- Barren tank contains a negligible fluid storage capacity and was ignored in the water balance calculations;
- The WRD will be reclaimed progressively with a robust cover system immediately after each section of the dump is stacked to its ultimate configuration, moving west to east;
- Temporary diversion channels are designed upslope of each WRD lift to limit the amount of runoff from undisturbed areas that report to the contact water ponds.
- The HLF process and event ponds were sized using the ultimate configuration of the leach pad, which occurs from years 6 through 16 of operations.
- Phased leach pad areas;
- Waste Rock Dump areas by year;

17.2.5.1 Solution Storage

The HLF is designed to be a zero-discharge facility during average and wet annual climate conditions. The HLF utilizes a Pregnant, Event, and Satellite ponds to collect and store solution. The process ponds are designed to contain the leachate solution and stormwater runoff from the heap during the 1 in 100 wet year climate scenario when the Phase 4 leach pad is in operation.

The HLF process ponds include provisions to accommodate the volume storage requirements from the following combined design conditions, below 0.6 meter of freeboard across all three ponds:

17.2.5.1.1 Pregnant Pond

- Operational volume: 36 hr working volume at the average solution application rate of 1,280 m^3/hr;
- Capacity for pump shutdown: 12 hr drain down at the maximum solution application rate of 1,400 m^3/hr;

17.2.5.1.2 Combined Pregnant and Event Ponds

- Operational volume: 36 hr working volume at the average solution application rate of 1,280 m^3/hr;
- Capacity for pump shutdown: 12 hr drain down at the maximum solution application rate of 1,400 m^3/hr;
- Maximum accumulation of water in the ponds resulting from the annual average precipitation;
- Runoff from the design storm event (1 in 100, 24 hr storm depth of 111.8 mm);

17.2.5.1.3 Combined Pregnant, Event and Satellite Ponds

- Operational volume: 36 hr working volume at the average solution application rate of 1,280 m^3/hr;
- Capacity for pump shutdown: 12 hr drain down at the maximum solution application rate of 1,400 m^3/hr;
- Maximum accumulation of water in the ponds resulting from the 1 in 100 wet year precipitation.

Table 17.2 presents the required storage volume of the process ponds. The approximate volume of the process pond based on the storage criteria listed above. Table 17.3 presents the capacity of the ponds below freeboard.

Table 17.2 Total Required Storage Volumes

Criteria	Pregnant Pond (m^3)	Combined Pregnant and Event Ponds (m^3)	Combined Pregnant, Event, and Satellite Ponds (m^3)
36-hr Operating Volume	46,080	46,080	46,080
12-hr Draindown Volume	16,800	16,800	16,800
100-yr, 24-hr Storm Volume	—	94,200	—
Average Annual Fluid Accumulation Volume	—	52,832	—
1 in 100 Wet Year Annual Fluid Accumulation Volume	—	—	550,428
Total Scenario Storage Volume Required Below Freeboard	**62,880**	**209,912**	**613,308**

Source: AECOM (2025)

Table 17.3 Designed Pond Storage Volumes

Criteria	Pond Storage Volume Below Freeboard (m³)		
	Pregnant Pond	Event Pond	Satellite Pond
Individual Pond Storage	86,200	160,800	366,800
Total Combined Storage	613,800		

Notes: 1. Pond storage capacities shown use a freeboard depth of 0.6m
Source: AECOM (2025)

17.2.5.2 Make-up Water Requirements

Heliostar requested AECOM to review the suitability of the climate data used in previous PFS. AECOM concluded that the information remains valid and reliable, making it appropriate for hydrologic modeling, water resource management, and infrastructure planning, as outlined in AECOM's Memo "Cerro del Gallo -Climate Phase 1" (AECOM 2025).

17.2.5.2.1 Average Climate Conditions

During average climate conditions, the make-up water requirements are reduced with each subsequent phased expansion. Table 17.4 presents the approximate make-up water flow rates required for Phases 1 through 4 during average climate condition.

Table 17.4 Make-up Water During Average Climate Conditions

Month	Phase 1 (m³/hr)	Phase 2 (m³/hr)	Phase 3 (m³/hr)	Phase 4 (m³/hr)
January	88.7	84.5	82.2	79.1
February	97.9	94.3	92.3	89.6
March	111.5	109.2	107.9	106.1
April	115.3	112.2	110.5	108.2
May	105.3	95.9	90.7	83.7
June	70.8	42.3	26.5	5.2
July	49.9	11.7	0.0	0.0
August	59.5	29.3	1.4	0.0
September	57.0	27.6	0.0	2.6
October	82.9	72.6	55.4	2.6
November	91.0	88.0	86.4	84.2
December	88.8	86.1	84.7	82.7
Average	**84.9**	**71.1**	**61.5**	**53.7**
Minimum	**49.9**	**11.7**	**0.0**	**0.0**
Maximum	**115.3**	**112.2**	**110.5**	**108.2**

Source: AECOM (2025)

17.2.5.2.2 1 in 100 Dry Year Climate Conditions

During the 1 in 100 dry year climate conditions, the make-up water requirements remain constant throughout operations. Make-up water requirements typically go down for each phased expansion due to the increased lined area that captures additional precipitation

runoff that can be used in the process circuit. Since the 1 in 100 dry year annual precipitation is 0.0 mm, there is no additional runoff captured with the phased expansions. Table 17.5 presents the approximate make-up water flow rates required for Phases 1 through 4 during the 1 in 100 dry year climate condition.

Table 17.5 Make-up Water During the 1 in 100 Dry Year Climate Conditions

Month	Phase 1 through 4 (m^3/hr)
January	114.5
February	125.0
March	139.7
April	145.9
May	144.8
June	136.8
July	128.6
August	125.7
September	121.2
October	118.4
November	115.1
December	111.8
Average	**127.3**
Minimum	**111.8**
Maximum	**145.9**

Source: AECOM (2025)

Excess Water Treatment During Reclamation

After the leach pad has been stacked to capacity or ore stacking has ceased, the heap will be rinsed in an effort to remove unwanted constituents from the ore, which may include metals and cyanide. The ore on the leach pad will be rinsed for a period of time that is necessary to bring the solution to an acceptable discharge quality. During this period, the drain down of rinsing solution and precipitation that drains down through the heap would exceed the capacity of the ponds during average and wet climate conditions. Therefore, a water treatment plant capable of removing 127,000 m^3 and 620,000 m^3 of annual excess water from the process ponds during the average and 1 in 100 wet year climate, respectively, has been incorporated into the process fluid system during this period.

17.2.6 SART

Pregnant solution will be treated in a SART plant for removal of copper and silver prior to entering the ADR plant for recovery of gold. Copper precipitation in the SART circuit also includes silver, as both are expected to precipitate together.

17.2.6.1 Copper Precipitation

Copper precipitations operations include acidification of pregnant solution, precipitation of copper with sodium hydrosulfide in three agitated tanks, thickening copper precipitate,

recycling thickener underflow solids to the precipitation tanks, neutralizing acidified thickener underflow prior to filtration and filtration. Filter cake will be conveyed to one of two drying pads with four days of capacity. The filter cake will be dried with air and construction heaters, if required. The dried copper precipitate will be sized to pass a 5 mm screen and loaded in plastic lined 20-ton containers or 1-tonne bulk bags for shipment.

17.2.6.1.1 Pregnant Solution Acidification

Concentrated sulfuric acid, 98 wt.%, will be mixed to 30 wt.% in a dilution tank. Pregnant solution will flow through an in-line mixer and be combined with 30 wt.% sulfuric acid to acidify the incoming pregnant solution to pH 4.0-4.5. The current design is for 1,400 m^3/hr of pregnant solution (nominal flow of 1,280 m^3/hr).

17.2.6.1.2 Copper Recycle Mix Tank

Thickener U/F recycle slurry, 15-43 m^3/hr at 10-25 wt% solids, is combined with fresh sodium hydrosulfide solution in the Copper Recycle Mix Tank before entering the first precipitation reactor. Sodium hydrosulfide, 25 wt%., is added to the Copper Recycle Mix Tank to condition the sulfide surface before entering the precipitation tank. The recycle mix tank will have a residence time of 2 minutes, diameter 1.7 meters and total height 4.2 meters, and constructed of 304 SS. The tank is agitated with dual A200/A510 impellers, 5.5 kW motor with shaft seal.

17.2.6.1.3 Copper Precipitation Tanks

The acidified pregnant solution is combined with recycled, conditioned copper precipitate in the first of three agitated precipitation tanks. The solution overflows from the third precipitation tank into the Copper Sulfide Thickener. The precipitation tanks are sized for 1451 m^3/hr at 1.1 wt% solids, with a residence time of 5 minutes per tank for a total of 15 minutes. The tank diameter is 5.7 meters and total height 7.2 meters and constructed of 304 SS. The tank is agitated with dual A510E impellers, 15 kW motor with shaft seal.

17.2.6.1.4 Copper Thickener

Solution overflowing the third precipitation tank will be combined with flocculent in the Copper Sulfide Thickener. Copper Sulfide Thickener overflow solution will gravity flow to the Neutralization Reactor. Copper Sulfide Thickener underflow will be recycled to the Copper Rapid Mix Tank, and advanced to the Filter Feed Tank.

The Copper Sulfide Thickener, 25 meters diameter x ~2.6 meters high, is high-rate thickener constructed of 304 SS and is covered. The thickener rake mechanism has an 11 kW motor. The design pressure will be for -500 mm H_20 to 700 mm H_20. The slurry is flocculated with 5 g/m^3 flocculent and thickened to an underflow percent solid from 10-25% solids. The design thickener rise rate is 3.0 m/hr.

Flocculent will be delivered dry and mixed in a standard mixing system and stored at a concentration of 0.5%. Flocculent will be diluted at the feed well to 0.02 wt.%.

17.2.6.1.5 Copper Precipitate Filtration

The Copper Thickener underflow will advance to the Filter Feed Tank and be filtered in the Copper Filters. The wet filter cake is conveyed to drying pads, dry material sizing, and conveyed to containers for bulk shipment to a smelter.

17.2.6.1.6 Copper Filter Feed Tank

Copper Thickener underflow slurry advancing at 2.0-5.6 m^3/hr at 10-25 wt.% solids will be stored in Copper Filter Feed Tank. The slurry at pH 4.0-4.5 will be neutralized with caustic solution 20 wt%, 0.027 m^3/hr to pH 7-8. The Copper Filter Feed Tank provides a residence time of 12 hours. The tank diameter is 4.2 meters, total height 5.2 meters, and constructed of 304 SS. The tank is agitated with a single A510E impeller, 19 kW motor with shaft seal.

17.2.6.1.7 Copper Filtration

Copper precipitate filtration cycles will operate manually every 4-8 hours. The batch cycle will be initiated by the operator, and the filter will be filled at a rate of 36 m^3/hr. The filling cycle will be followed by filtrate removal with a core blow. The filter cake be subjected to a blow cycle, and will discharge with a cake moisture of 40 wt.%. A specific filtration rate of 33 kg/m^2/hr, 15.9 mm per recessed plate, 30 mm cake thickness, and 10-16 bar operating pressure will give a filter with a total chamber volume of 3.5 m^3. Two filters will be installed. Cake blow will be designed for 2 minutes at 1 m^3/m^2-min. Filtrate will be returned to the Neutralization Tank at a rate of 18 m^3/hr.

17.2.6.1.8 Filter Cake Conveying

Copper filter cake will be produced at a rate of 0.728 tph as a 40 wt% cake, 0.296 tph dry cake based on 90% copper precipitation and 95% silver precipitation. The filter cake may assay up to 61.6 wt% copper and 1.174% silver and may be contaminated with gypsum and sediments that may lower the metal grades.

Copper filter cake batches will be conveyed away from the filter press over a 15-30 minute period at a rate of 6.1 wet tph to one of two drying pads. One drying pad will provide 4 days of residence time. Filter cake moisture will be reduced from 40 wt% to 15-20 wt%. The drying pads will be partially covered, with fans to blow air over the filter cake. Two construction type propane heaters rated for 1 million BTU/hr/each will provide heat for drying during winter nights. The dried filter cake will be sized to 100% minus 5 mm with a roll crusher/lump breaker and vibrating screen. The screen underflow product will be conveyed at a rate of 2 tph to 20-ton containers. The blended copper concentrate will be sampled from the conveyors as they fill the containers for concentrate settlement purposes. Two 20-ton containers will be filled every four days.

17.2.6.1.9 Caustic Scrubber Systems

The Copper Recycle Mix Tank, Copper Precipitation Tanks, Copper Sulfide Thickener, Copper Filter Feed Tank, NaSH Storage Area sumps and Copper Area sumps will be ventilated to a caustic scrubbing system. The caustic scrubbing system will consist of two scrubbers, both packed towers with integral pumps, instrumentation and fan on UPS and emergency power. All motors will have VFD drives. The scrubbing system will remove hydrogen sulfide and hydrogen cyanide from the vent gases with 10-20 wt% caustic solution.

The scrubbing system design allows for a normal operating case and an emergency operating case. The normal operating case will treat 6,800 Nm^3/hr (4,000 scfm) with assumed HCN and H_2S concentrations of 100 ppmv. The scrubber efficiency of 99.8% will discharge 0.2 ppmv HCN and H_2S.

The emergency scrubber system will operate with caustic solution circulating to the top of the packing and back to the pump, but not through the packing. Continuous hydrogen cyanide and hydrogen sulfide monitors will divert Normal Scrubber Discharge gas to the emergency scrubber when a high concentration of either gas is detected and simultaneously open the valve to distribute solution to the emergency column packing. The emergency scrubber is designed to treat a burst of gas at 17,100 Nm^3/hr (10,000 scfm), 76,800 ppmv H_2S, and 61,300 ppmv HCN for 5 minutes. The scrubber will discharge 10 ppmv HCN and 15 ppmv H_2S.

The quantity and concentrations of gas for the emergency release case are based on complete acidification of one precipitation tank as a batch process, with release of all reactive gases. The gas burst duration was based on review of plant operating data in a metal sulfide leach process with concentrated sulfuric acid that generated large pulses of hydrogen sulfide gas.

The Normal Operation Scrubber is 1.0 meters diameter and 6.4 meters high with 13.6 m^3/hr circulation rate with a 3 minutes solution hold-up, and is constructed of FRP.

The Emergency Scrubber is 1.0 meters diameter and 9.9 meters high with 17.0 m^3/hr circulation rate with 8 minutes of solution hold-up, and is constructed of FRP. A single extension stack from the scrubbing system will vent gas from both scrubbers 20 meters above grade.

17.2.6.1.10 On-Line Analysis

In order to minimize operator exposure to process streams containing HCN and H_2S, timely respond to fluctuations in copper concentrations caused by normal operations and

sudden rainfall, and reduce over and under feeding of reagents, an on-line analyzer is provided.

On-line analysis of Cu, Zn, Cd, Ag, pH, redox and sulfuric acid will be determined from five streams, the pregnant solution, the acidified pregnant solution, and the discharge from each precipitation tank. The system includes stream sampling with a multiplexer, primary and secondary sample filtration, XRF analysis of Cu, Zn, Cd, and Ag, and analysis of pH, redox potential, and sulfuric acid with an automatic titrator.

17.2.6.1.11 Area Sump Pumps and Hydrogen Peroxide System

Sump pumps in the sodium hydrosulfide area, and all acidic solution areas will be vented to the caustic scrubber system. The ventilated sumps will minimize accumulation of hydrogen sulfide gas in the sump areas. Hydrogen peroxide, 10 wt%, solution will be provided to sump areas and sample points in a ring-main type system to destroy hydrogen cyanide and hydrogen sulfide during upset process conditions.

17.2.6.2 Solution Neutralization

Solution from the copper precipitate thickener will overflow by gravity to the Neutralization Tank. The acidified thickener overflow will be neutralized with slaked lime and recycled gypsum thickener underflow slurry. Slurry from the Neutralization Tank will discharge to the Gypsum Thickener. Gypsum thickener overflow solution will gravity flow to the ADR plant. Gypsum thickener underflow will be recycled and advanced to a storage pond the first year of operation and unused areas of the heap leach pad for the life of the mine.

17.2.6.2.1 Recycle Gypsum Mix Tank

Recycle gypsum thickener underflow solids will be conditioned with slaked lime in the recycle mix tank to simulate a high-density sludge (HDS) process, and achieve higher underflow solids densities than typically generated by direct neutralization, which generates a low-density sludge.

The recycle gypsum thickener underflow, 10.6 m^3/hr, 25 wt% solids will be mixed with slaked lime, 5.8 m^3/hr, 20 wt% solids in the Gypsum Thickener Recycle Mix Tank. The carbon steel/rubber lined tank will provide 5 minutes retention time and have a diameter of 1.5 meters and 2.0 meters total height. The tank will be agitated with a single A510E mixer with 3.7 kW installed power.

17.2.6.2.2 Neutralization Tank

The acidified pregnant solution from the copper thickener, 1,407 m^3/hr, is combined with recycled conditioned gypsum solids in the Gypsum Neutralization Tank, 16.4 m^3/hr, caustic scrubber discharge, 0.1 m^3/hr, and copper filter filtrate, 1.2 m^3/hr. The solution overflows the Neutralization Tank through and upcomer to the Gypsum Thickener at a rate of 1425 m^3/hr.

The Neutralization Tank is sized for a residence time of ~5 minutes. The tank diameter is 5.7 meters and total height 7.2 meters, is constructed of carbon steel, and is rubber lined. The tank is agitated with a single A510E impellers, 3.75 kW motor.

17.2.6.2.3 Gypsum Thickener

Solution overflowing the Neutralization Tank will be combined with flocculent in the Gypsum Thickener. Gypsum Thickener overflow solution will gravity flow to the ADR plant. Gypsum Thickener underflow will be recycled to the Recycle Gypsum Mix Tank, and advanced to a holding pond the first year of operation, and to unused areas of the heap leach pad thereafter.

The Gypsum Thickener, 28.3 meters diameter x ~2.6 meters high is high-rate thickener constructed of carbon steel. The thickener rake mechanism has a 15 kW motor. The slurry is flocculated with 5 g/m^3 flocculent and thickened to an underflow percent solid from 10-25% solids. The design thickener rise rate is 2.25 m/hr.

Flocculent will be delivered dry and mixed in a standard mixing system and stored at a concentration of 0.5%. Flocculent will be diluted at the feed well to 0.02 wt.%.

17.2.7 Metal Recovery

The recovery plant is designed to recover gold and any silver not recovered by the SART plant by an adsorption-desorption-recovery (ADR) process. Precious metals in the heap leach pregnant solution will be adsorbed on to activated carbon in the carbon adsorption circuit (adsorption). Loaded carbon from the carbon adsorption circuit is then desorbed in a high-temperature elution process coupled to an electrowinning circuit (desorption), followed by drying and smelting of the resulting sludge to produce doré bullion (recovery). Prior to elution, each batch of carbon will be acid washed to remove any scale and other inorganic contaminants that might inhibit gold adsorption on carbon.

17.2.7.1 Adsorption

The adsorption section of the ADR will consist of a single train of carbon columns consisting of five cascade type open-top up-flow carbon adsorption columns. Each of the carbon columns will have a capacity of 10 tonnes of activated carbon.

Pregnant solution is pumped to the carbon adsorption columns by submersible pumps in the pregnant solution pond. Antiscale agent is added at the pump suctions to prevent scaling of the carbon that can affect carbon loading. Barren solution exiting the last carbon columns flow through a screen to separate and capture any floating carbon from the solution.

Adsorption of gold from the pregnant solution is a continuous process. Periodically, the carbon contained in the lead column in the series becomes loaded with gold and is transferred to the acid wash and desorption circuit as a batch using carbon pumps. On average, approximately 5 tonnes of carbon per day are expected to be loaded and treated. The carbon columns hold a total of 10 tonnes of carbon so holding tank with retain 5 tonnes of loaded carbon while the remaining 5 tonnes are being stripped.

Carbon in the remaining columns is then advanced, one at a time, and a batch of new (or stripped/regenerated) carbon is transferred into the final empty column from the unloaded carbon storage tank.

Generally, the stripping of carbon will occur about 7 times each week with each strip lasting 12 to 16 hours.

17.2.7.2 Carbon Acid Wash

Acid washing consists of circulating a dilute acid solution through the bed of carbon to dissolve and remove scale from the carbon. Acid washing is performed on a batch basis.

After carbon is transferred into the acid wash column, but before any acid is introduced, fresh water is circulated through the bed of carbon to remove any entrained caustic cyanide solution. This rinse solution is pumped to a waste collection pipe with the acid wash circulation pump where it is transferred to the barren tank. A dilute acid solution is then prepared in the mix tank, and circulation is established between the acid wash vessel and the acid mix tank. Concentrated acid is injected into the recycle stream to achieve and maintain a pH ranging from 1.0 to 2.0. Completion of the cycle is indicated when the pH stabilizes around 2.0 without acid addition for a minimum of one full hour of circulation.

After acid washing has been completed, the acid wash pump will transfer spent acid solution from the acid mix tank and wash vessel either to the acid recovery tank or directly to the waste collection pipe. The carbon is then rinsed with raw water followed by rinsing with dilute caustic solution to neutralize any residual acid. Total time required for acid washing a ten-tonne batch of carbon is four to six hours. After acid washing is complete, a carbon transfer pump will transfer the carbon to the desorption section.

17.2.7.3 Desorption

A Zadra pressure elution, hot caustic desorption circuit has been selected for the CdG Project. This type of circuit requires 24 hours or less to complete a cycle and, for this reason, each strip batch is sized for ten tonnes of carbon. Each desorption cycle requires the transfer of a ten tonne batch from the acid wash circuit to the strip vessel.

The desorption circuit is sized to elute, or “strip,” the gold from a ten-tonne batch of carbon into pregnant eluate solution. During the elution cycle, gold is continuously extracted by electrowinning from the pregnant eluate concurrently with desorption. A complete desorption cycle will require approximately 18 hours.

After a batch of carbon has been transferred to the elution vessel, barren strip solution (eluant) containing sodium hydroxide and sodium cyanide is pumped through the heat recovery and primary heat exchangers, and introduced to the elution vessel at a temperature of 135°C and a nominal operating pressure of approximately 340 kPa (50 psig).

Under normal operating conditions, barren eluant solution from the solution storage tank will pass through the heat recovery exchanger to be preheated by hot pregnant eluant leaving the elution column. The barren eluant solution then passes through the primary heat exchanger to raise the temperature up to 149 °C using pressurized hot water from the boiler system.

The elution column contains internal stainless steel inlet screens to hold carbon in the column and to distribute incoming stripping solution evenly in the column. Pregnant eluant solution leaving the elution column passes through external stainless steel screens before passing the cooling heat exchanger to reduce the eluate temperature to about 75°C (to prevent boiling). The cooled pregnant eluate solution is sent to the electrowinning cells.

After desorption is complete, half of the stripped carbon is pumped to carbon reactivation dewatering screens to remove water and carbon fines, and transferred to carbon regeneration. The other half of the carbon is screened to remove fines and transferred to the carbon storage tank.

17.2.7.4 Electrowinning and Refining

The electrowinning circuit is operated in series with the elution circuit. Solution is pumped continuously from the barren eluant tank through the elution vessel, then through the electrowinning cells, and back to the barren eluant tank in a continuous closed loop process.

The gold-laden solution exiting the elution column is filtered to trap any carbon escaping from the column; passes through the heat recovery exchanger and the cooling exchanger to reduce the solution temperature to 75ºC and flows to the electrowinning circuit.

Gold is won from the eluant in the electrowinning cells using stainless steel cathodes and a current density of approximately 50 amperes per square meter of anode surface.

Caustic soda (sodium hydroxide) in the eluate solution acts as an electrolyte to encourage free flow of electrons and promote the precious metal winning from solution. To keep the electrical resistance of the solution low during desorption and the electrowinning cycle, make-up caustic soda must sometimes be added to the barren eluant tank. Barren eluate solution leaving the electrolytic cells discharges to the E-cell discharge pump box where it is pumped back to the eluate storage tank for recycle through the elution column.

Periodically, all or part of the barren eluant is dumped to the barren tank and new solution is added to the eluate storage tank. Typically, about one-third of the barren eluant is discarded after each elution or strip cycle. Sodium hydroxide and sodium cyanide are added as required from the reagent handling systems to the barren eluant tank during fresh solution make-up.

The precious metal-laden cathodes in the electrolytic cells are removed about once or twice per week and processed to produce the final doré product. Loaded cathodes are transferred to a cathode wash box where precipitated precious metals are removed from the cathodes with a pressure washer. The resulting sludge is pumped to a plate-and-frame filter press to remove water and the filter cake is loaded into an electric dryer to remove moisture from the filter cake.

After drying, the gold sludge will be mixed with fluxes and smelted in an electric furnace to produce doré bullion.

Periodically, slag produced from the smelting operation is re-smelted on a batch basis to recover residual metal values, or will be crushed and manually added to the heap leach pad.

A hood collects the furnace fumes which will pass through a bag house to remove particulates, then through an induced draft fan. The system will be designed to remove over 99.5% of the particulates present in the exhaust fumes.

17.2.7.5 Carbon Handling and Regeneration

Thermal regeneration consists of drying the carbon thoroughly and heating it to approximately 750ºC for ten minutes. It is expected that thermal reactivation will be performed after every elution cycle to maintain carbon activity levels.

The ten-tonne carbon batch to be thermally reactivated is dewatered on a static screen, transferred to the regeneration kiln feed hopper and fed to the regeneration kiln by a screw feeder. Hot, regenerated carbon leaving the kiln falls into a water-filled quench tank for cooling and storage. Carbon in the carbon quench tank is pumped to a vibrating

screen; screen oversize is sent to the carbon storage tank and the screen undersize is collected in in the carbon fines tank, where periodically the carbon fines are dewatered using a filter press and stored in bulk bags. Ultimately, quenched regenerated carbon is pumped to the adsorption circuit dewatering screen to remove any fines and the coarse carbon is added to the adsorption circuit.

New carbon is first added to the carbon conditioning tank which is equipped with an agitator and is used for attritioning new carbon. After attritioning, the new carbon is transferred to the unloaded carbon tank from which it is transferred to the adsorption circuit by a carbon transfer pump.

17.2.8 Reagents

17.2.8.1 Cyanide

Sodium Cyanide is delivered as briquettes in 1,000 kg bulk bags or in ISO containers and is stored in a covered storage area with approximately 30 days of storage. Dual systems are installed to either mix the briquettes in an agitated tank or utilize a solid liquid system that pumps water through the ISO containers. Sodium cyanide is used to leach the gold and silver from the ore on the heap.

17.2.8.2 Cement

Cement is delivered in bulk truckloads. Cement storage is in two 150-tonne silos with an estimated cement consumption in the range of 84 to 334 tonnes per day depending on sample type, with a LOM average of 166 tonnes per day.

Cement from the silos is metered directly onto the agglomeration feed conveyors through variable speed feeders based on weightometer measurements. The cement silos will be equipped with bin activators and dust collectors.

17.2.8.3 Slaked Lime

Pebble lime will be delivered and stored in a 150-tonne lime silo. The lime will be conveyed from the silo to a lime slaker system. Slaked lime will be stored in an agitated tank with 12 hours residence time at a solids density of 20 wt%. Slaked lime will be pumped to the Gypsum Thickener Mix Tank at a rate of 5.8 m^3/hr. The Lime Storage Tank has a diameter 5.5 meters and total height 5.8 meters. The tank is agitated with dual A510E impellers with 5.6 kW installed power.

17.2.8.4 Sodium Hydroxide (Caustic)

Sodium hydroxide will arrive as 50% solution in 10,000-liter containers. The caustic will be diluted to 20 wt% for storage. Storage will take place at the SART plant in a stainless tank, 2.7 meters diameter and 3.1 meters high. Caustic will be distributed from the SART plant to the ADR in a 55-gallon drum, or similar sized day tank.

17.2.8.5 Concentrated Sulfuric Acid

Concentrated sulfuric acid, 98 wt%, will arrive by truck in 20-ton batches. The truck will be unloaded into a single carbon steel tank 5.7 meters diameter, 5.7 meters high. The tank provides storage capacity for 3 days.

17.2.8.6 Sodium Hydrosulfide (NaHS)

Sodium hydrosulfide will arrive in tanker truck at a 40 wt.% solution. The tanker truck will be unloaded into the Sodium Hydrosulfide Dilution Tank. The tanker contents will be sampled and diluted to 25 wt% for storage. The dilution tank is, 3.4 meters diameter and 4.0 meters high and is heat traced and insulated. The storage tank is 5.0 meters diameter and 5.4 meters high and is heat traced and insulated. The diluted storage tank provides 9 days of storage.

17.2.8.7 Flocculant

Flocculant will arrive as a dry powder in 25 kg bags. Flocculant will be mixed in a flocculant mixing system, and transferred to a storage tank as 0.5 wt% solution. The flocculant storage tank provides 16 hours residence time and is 4.3 meters diameter and 4.8 meters high.

17.2.8.8 Antiscale Agent

Antiscale Agent will be received in drums or plastic tote containers. Antiscale agent will be added by metering pumps at the barren solution and pregnant solution pump suction inlets. Antiscale agents will be used to prevent carbonate scaling in pumps, piping and on the carbon.

17.2.8.9 Hydrogen Peroxide

Hydrogen peroxide, 10 wt.%, will be delivered to the SART plant in 10,000-liter containers and transferred into 304 SS storage tank. The design rate is for 100 L/hr, or 2.4 m^3/day. The storage tank will be 2.5 meters diameter and 3.5 meters high and provide 12 m^3 of storage, 5-days of storage. Hydrogen peroxide will be fed by pumps to sump areas and sample points in the SART plant area via a ring-main to destroy hydrogen cyanide and hydrogen sulfide as required.

18 Project Infrastructure

Existing infrastructure for the CdG includes a site office, dirt and gravel roads, and limited power line throughout the Project site. Internet and cellular communications are currently available, though these systems will need to be expanded for operations.

18.1 Roads

Access to the Project site is by the paved Mexican Highway 110, and a secondary dirt road that goes to San Antón de las Minas community. A private road will enter into the mine property approximately 2 km before reaching the San Antón de las Minas community. This road will provide access to the administration offices, mine, process plant and other Project facilities. After the intersection of the private and community road, approximately 1.5 kilometers of the community road will be re-located a short distance to the south of the existing road.

18.1.1 Site Roads

Internal site roads are established to serve as mine haul roads, service roads and in-plant roads which connect the facilities for access purposes.

18.1.1.1 Haul Roads

Haul roads have been designed for hauling ore and waste from the mine pit to either the crusher or to the waste rock disposal dumps. Haul roads are designed with a maximum percent grade of 10% and a 25-meter width. Maximum speed on these roads should be no more than 40 km/h.

Haul roads will be constructed by the mining contractor.

18.1.1.2 Service Roads

The site service roads are connected to the site access road and are used to join the site facilities. All service roads are designed at a maximum percent grade of 12.5% and a width of 8 meters. The combined service roads join the following areas:

- Administrative area;
- Primary crushing;
- Secondary and tertiary crushing;
- Leach pad;
- SART plant;
- ADR plant.

18.1.1.3 In-Plant Roads

In-plant roads are required around the crushing area and are primarily for maintenance purposes. In-plant roads are designed at a maximum grade of 12.5% and a width of 6 meters for one-way access and 12 meters for two-way access.

18.2 Power Supply and Distribution

Power supply to the CdG Project will initially be generated on site using two each 2,500 KW diesel generator units operating, with an additional unit on standby. Power will be generated at 4,160 V, 3 phase, 60 Hz and stepped up to 13.8 kV by a transformer for site distribution. The generator system has been sized to meet the average power demand.

It is assumed that in Year 2 of operations, power supply at 115KV will be available by connecting to the national grid and power generation at site will no longer be needed. Site power will be distributed using overhead power lines at 13.5 kV with the main substation be located near the largest power consumption area which will be the HPGR area. Power from the main substation will be stepped down to 13.8 kV and connected to the site distribution power line. Two of the temporary generators and their associated fuel tanks will remain at the project to be utilized as emergency power backup for the process plants.

18.2.1 Estimated Power Consumption

The estimated project power consumption is presented in Table 18.1. Required power for the operations and the facilities is approximately 6.9 MW on average for LOM, and 6.4 MW average excluding the first and last year of operation (6.4 MW and 2.5 MW power consumption for first and last years, respectively).

Table 18.1 Project Electrical Power Consumption – LOM Average

Area	Attached kW	Annual kWh Consumed	kWh/t Ore
Area 001 - Site & Utilities General	248	677,160	0.11
Area 005 - Power Generation & Site Distribution	15	58,320	0.01
Area 010 - Primary Crushing	425	2,066,581	0.34
Area 011 - Secondary Crushing	1,048	5,091,069	0.85
Area 012 - Tertiary Crushing	4,359	21,185,834	3.53
Area 020 - Agglomeration	494	2,396,278	0.40
Area 030 - Stacking System	1,002	3,440,544	0.57
Area 031 - Heap Solution Handling	2,121	12,537,704	2.09
Area 035 - SART Plant	703	4,069,329	0.68
Area 040 - Recovery	303	993,872	0.17
Area 050 - Refining	393	1,364,181	0.23
Area 080 - Reagents	336	1,512,759	0.25
Area 090 - Water Distribution System	576	2,415,533	0.40
Area 100 – Laboratory	293	1,423,980	0.24
Total	**12,316**	**59,233,144**	**9.87**

Source: KCA (2025)

18.3 Water Supply and Distribution

The Project obtained eight water concessions under Argonaut, however five of the concessions expired in November 2024 due to lack of payments. The Company has discussed with CONAGUA the possibility of reactivating the six expired concessions, and is waiting to determine whether additional concessions are needed meet the project's water demand before taking any actions. The total concession volume currently permitted and active is 640,000 m^3/year. The water concession information is provided in Table 18.2.

Table 18.2 Water Rights and Volumes

Concessionaire	Title Number	Granted Volume m^3
San Antón de las Minas S.A. de C.V.	08GUA154831/12FMDL14	130,000
San Antón de las Minas S.A. de C.V.	08GUA154832/12FMDL14	130,000
San Antón de las Minas S.A. de C.V.	08GUA154833/12FMDL14	140,000
San Antón de las Minas S.A. de C.V.	08GUA123064/12FMDL16	300,000
San Antón de las Minas S.A. de C.V.	08GUA155423/12FMDL16	200,000
San Antón de las Minas S.A. de C.V.	08GUA110394/12FMDL16	140,000
San Antón de las Minas S.A. de C.V.	08GUA155418/12FMDL16	250,000
San Antón de las Minas S.A. de C.V.	08GUA155432/12FMDL16	150,000
		1,440,000

Source: Heliostar (2025)

The project is expected to consume an annual average of 628,000 – 900,000 m3 per year, depending on the amount of leach pad in the circuit. San Antón currently has two wells that tests indicate will provide approximately half of the maximum project demand. Additional wells will be drilled to provide the remainder of the estimated maximum demand of 55 L/s.

As discussed in further detail in Section 24.2, there are two water wells that belong to the CdG Project. The first water well is located about 4 km from the property boundary. It is completely closed, protected, and ready to install the pump and pipes. The power supply will be through the CFE power line that passes nearby. The second well is 7.5 km from the property boundary. This well is piped and already connected to the CFE power line.

Flow tests on these two wells resulted in sustainable flows of 12 L/s and 16 L/s, respectively. There is a plan to drill more wells to reach the design, short term water requirement of 55 L/s. A current study was conducted to find the zones and targets to find water on the average of 25-30 L/s.

The project will require water supply for the following uses:

- Mining operations for dust control, drilling, etc.;
- Crushing for dust control;
- Makeup water for the heap leach pad;
- Process plant and laboratory;
- Modular offices and other site facilities.

18.3.1 Process Water

AECOM determined operational water requirements in conjunction with climatic conditions, as discussed in Section 17. The process water balance considers the water consumed by the Project and the water collected from precipitation events on the Project components in addition to seasonal evaporation.

Solution from the heap leach pad will drain to the Pregnant Pond, where it will be pumped through the processing facility to recover precious metals and then pumped back to the leach pad in a continuous cycle. The Event Pond will be located adjacent to the pregnant solution pond to allow containment of excess process solution during precipitation events which will add additional water to the contained system. A Satellite Pond will also be included where excess water can be transferred for containment. The Pregnant, Satellite and Event Ponds were designed with a combined capacity to contain normal process volumes and contingent storage capacity for a total of 613,800 m^3 below the freeboard level.

Process water requirements are first met by pumping collected waters from the Event and Satellite Ponds; after that resource is exhausted, make-up requirements will be met by well water.

18.3.2 Raw Water

Raw water for the project will be pumped directly from the water wells to raw water tanks. There are two raw water tanks included in the project; one tank located next to the administration area and a secondary water tank located near the crushing area. A pumping system will be included at the primary raw water tank to pump water directly into the secondary water tank so during normal operation, the water wells will only need to pump into the primary water tank.

Water from the storage tanks will gravity flow to be utilized in the process facilities and for domestic uses.

18.3.3 Potable Water

Potable water will be bottled and delivered to the project site.

18.3.4 Fire Water

The raw water tank located near the administration area will be dual-purpose tank, a portion of this tank will be designated for fire water use.

18.4 Project Buildings

The project facilities will be supplied in the form of a modular office, shipping containers for warehouse storage, the refinery will be masonry walls with a structural steel roof, the motor control centers (MCC) are assumed to be modified shipping containers and prefabricated steel buildings will be used for the administration building, laboratory, guard house, clinic, dining room, training room.

18.5 Explosives Storage

Facilities for the proper storage and safekeeping of explosives are included. These facilities will be designed and located in compliance with Mexican regulations.

Within a dedicated area fenced for explosives, there will be one ventilated silo and two CMU brick explosive magazines. The silo is designated for Ammonium Nitrate storage. The explosive storage silo will have a capacity of approximately 65 tons of ammonium nitrate.

A 200 m^2 CMU brick powder magazine will be used to store low explosive products, such as ANFO, emulsion packaging, boosters and detonation cord.

A smaller 20 m^2 magazine will be used to store the detonators and will have at least 300 meters separation from the larger magazine and silo.
Approximate distances from the explosives storage to notable infrastructure are as follows:

- 600 meters north of the heap leach boundary;
- 1,100 meters north of the primary crusher;
- 200 meters north of the LOM waste rock dump limit;
- 2,200 meters north of the admin building.

All of the above distances exceed the minimum safety distance requirements of the explosive regulations established by Secretaría de la Defensa Nacional (SEDENA) plus a natural barrier between some of the facilities.

18.6 Security

Access to the project will be limited by perimeter fencing around the entire site. A guardhouse at the primary entry point to the project will serve as a security check point that will be manned 24 hours per day, 7 days a week for identification control, random checks, drug and alcohol monitoring and vehicle check-in/out. A security contractor will be used for general site security and protection of mine assets.

18.7 Waste Disposal

18.7.1 Sewage

Wastewater and sewage will be handled by subsurface local septic tanks and centralized leach-fields.

18.7.2 Solid Waste

Special wastes such as waste oil, glycol coolant, solvent fluids, used oil filters, used batteries, and contaminated fuel, will be handled, stored, transported, and disposed of in accordance with appropriate Hazardous Waste Regulations. A certified transport and disposal company will collect all waste to transport offsite for final disposal.

A fenced temporary storage facility for hazardous waste will be included. A roofed storage area will be designated for used batteries, used lubricants, coolant and other miscellaneous fluids, and used tires.

A site for temporary storage of recyclable materials will be established. Such items as scrap metal, tires, glass, recyclable plastics and drink containers will be separated, containerized as appropriate, and temporarily stored until sufficient volumes are available for shipment to a recycling point.

Non-recyclable and non-hazardous waste will be managed with a dedicated local company and waste sent to the Dolores Hildalgo municipal landfill. There currently exists waste disposal service in the San Antón de las Minas community on a weekly basis. The project will contract waste disposal services to the current established infrastructure.

A location on the mine site will be designated as an outdoor storage or 'boneyard' area for placement of items that are not yet ready for disposal, but which may still be of use for spare parts. These items are likely to include equipment parts, vehicles, and pieces of equipment, and metal components. As much of this material as possible will be utilized during the mine life. Materials remaining in the boneyard at the end of mine life will either be shipped off site for salvage value, recycled, or disposed of in the landfill if they meet the criteria for disposal at that location.

19 Market Studies and Contracts

No market studies for gold were completed and no contracts are in place in support of this Technical Report. Gold production can generally be sold to any of a number of financial institutions or refining houses and therefore no market studies are required.

Multiple potential buyers were contacted for potential purchase of the SART precipitate. One company provided payment terms for the SART precipitate. Based on this response, the terms used for this study are as follows:

- Metal Payments
  - Copper – 90% of the concentrate content
  - Silver – 90% of the concentrate content
  - Gold – 90% of the concentrate content
- Deductions
  - Treatment Charge – US$210.00 per dry tonne of concentrate.
  - Copper Refining Charge – US$0.08 per pound of the payable copper.
  - Silver Refining Charge – US$0.45 per troy ounce of the payable silver.
  - Gold Refining Charge – US$6.00 per troy ounce of the payable gold.

The base case financial model for the CdG Project utilizes a gold price of US$2,300/oz, silver price of US$27.00/oz, and copper price of US$ 4.00/lb.

This study assumes that mining operations will be conducted by contractors working under the supervision of the chief mining engineer. The required contracts are:

- A general mining contractor;
- A blasting agent/high explosives manufacturer that will also be responsible for delivering the blasting products to the site, loading the blast holes and detonating the blasts;
- A specialty drilling contractor to drill small diameter holes for pre-splitting final pit walls and drilling holes for slope reinforcement if the general mining contractor cannot perform these tasks. (Quotations for these services have been received and were used to estimate costs for the study, but no contracts are currently in place.)

20 Environmental Studies, Permitting and Social Impact

The available information on environmental, permitting and social or community factors related to the Cerro del Gallo Project (Project) is discussed in this section. Data relevant to this section were primarily provided by the company San Antón de las Minas, S.A. de C.V., now a wholly-owned subsidiary of Heliostar Metals (Company), via the Project data room.

20.1 Introduction

The Project is located in a historic mining state and extraction activities from a nearby underground mine has led to a local community that developed around the mine. Multiple environmental baseline studies and permitting efforts were conducted by previous owners; however, much of the original information has not been located by the Company. The Company will be reviewing the available information to determine whether a new permitting effort with all new baseline studies will be initiated or whether there is available information that could be used as part of the permitting effort. The Company personnel include an environmental permitting specialist and a community relations specialist who are supporting the Project.

20.2 Environmental Permitting Requirements and Status

The Mexican Constitution contains provisions for the regulation of natural resources in Article 27, which is regulated by the Mexican Mining Law for mining activities such as exploration, mining, and processing activities.

On May 8, 2023, the Mexican Government enacted a decree amending several provisions of the Mining Law, the Law on National Waters, the Law on Ecological Equilibrium and Environmental Protection and the General Law for the Prevention and Integral Management of Waste, which became effective on May 9, 2023 (the Decree). This Decree amends the Mexican mining and water laws, including: (i) the duration of the mining concession titles, (ii) the process to obtain new mining concessions (through a public tender), (iii) imposing conditions on water use and availability for the mining concessions, (iv) the elimination of “free land and first applicant” scheme; (iv) new social and environmental requirements in order to obtain and keep mining concessions, (v) the authorization by the Mexican Ministry of Economy of any mining concession’s transfer, (vi) new penalties and cancellation of mining concessions grounds due to non-compliance

with the applicable laws, (vii) the automatic dismissal of any application for new concessions, and (viii) new financial instruments or collateral that should be provided to guarantee the preventive, mitigation and compensation plans resulting from the social impact assessments, among other amendments.

Over 500 constitutional challenges, known as “amparos”, have been filed against the new law. The challenges include arguments that the reforms violate due process and impose burdensome requirements on mining companies, and there was a lack of debate and transparency in the Senate during the passage of the reforms. Additional implementing regulations associated with the mining law reforms were expected to be issued within 180 days (that is, early November 2023), and that timeframe was later updated to be June 2025 per information shared by the Extractive Activities Coordination Unit of the Ministry of Economy; however, none had been issued as at the Report effective date.

20.2.1 Environmental Permitting Regulations

The Secretariat of Environment and Natural Resources (Secretaría de Medio Ambiente y Recursos Naturales, or SEMARNAT) is the main regulatory body in charge of enacting and enforcing environmental regulations throughout Mexico, including the issuance of environmental permits. SEMARNAT comprises multiple autonomous agencies with administrative, technical, and advisory functions, which are summarized in

Table 20.1

Table 20.3. An additional agency, the National Institute of Anthropology and History (Instituto Nacional de Antropología e Historia or INAH) has responsibility for the preservation, research, and dissemination of the country's archaeological, anthropological, historical, and paleontological heritage.

Table 20.1 Overview of SEMARNAT Agencies

SEMARNAT Unit	Function
National Water Commission (Comisión Nacional del Agua, or CONAGUA)	Responsible for the management of national water, including issuing water concessions, water extraction permits (both surface water and groundwater), and wastewater discharge permits.
National Forestry Commission (Comisión Nacional Forestal, or CONAFOR)	Mandate is to develop, support, and promote the conservation and restoration of Mexico's forests.
Attorney General for Environmental Protection (Procuraduría Federal de Protección al Ambiente, or PROFEPA)	Monitors compliance with environmental regulations and responsible for the enforcement of environmental law.
National Commission for Natural Protected Areas (Comisión Nacional de Areas Naturales Protegidas, or CONANP)	Oversees the management and protection of 192 protected areas throughout Mexico.
The Safety, Energy and Environment Agency (Agencia de Seguridad, Energía y Ambiental, or ASEA)	Regulates and oversees industrial safety and environmental protection, and integrated waste management specifically with respect to hydrocarbon-related activities.
National Institute of Ecology and Climate Change (Instituto Nacional de Ecologia y Cambio Climatico or INECC)	Responsible for the coordination of technological and scientific research and development with a focus on environmental protection and conservation. This institute provides technical and scientific support to SEMARNAT for the instrumentation of the national environmental policy, so to promote and diffusion of criteria, methods and technologies for environmental conservation and sustainable use of natural resources. It also evaluates compliance of the goals and actions of the Climate Change National Strategy.
General Directorate of Environmental Impact and Risk (Subsecretaría de Gestión para la Protección Ambiental con la Dirección General de Impacto y Riesgo Ambiental, or DGIRA)	Responsible for issuing environmental permits and authorizations.

The primary environmental law is the General Law on Ecological Equilibrium and Environmental Protection (Ley General de Equilibrio Ecológico y Protección al Ambiente or LGEEPA), which provides a general legal framework for environmental legislation. Key related Federal statutes include:

- General Law on Sustainable Forest Development (Ley General de Desarrollo Forestal Sustentable)
- National Water Law (Ley de Aguas Nacionales)
- General Law on Wildlife (Ley General de Vida Silvestre)
- General Law on Climate Change (Ley General de Cambio Climático)

- General Law on the Prevention and Comprehensive Management of Waste (Ley General para la Prevención y Gestión Integral de los Residuos)
- General Law of Environmental Responsibility (Ley General de Responsabilidad Ambiental)

SEMARNAT regulates permitting or licenses under the regulations and standards derived from LGEEPA, divided into the following main topics:

- Hazardous materials and wastes: registration of generators, management plans, authorization to manage hazardous waste, contaminated soil remediation, import/export permits, environmental risk assessments and approval of accident prevention plans
- Forest management: authorizations, notices, reports, inscriptions and records regarding timber and non-timber forest exploitation, commercial forest plantations, collection of forest biological resources, phytosanitary certificates, land use change in forest land, forest product transportation, storage and transformation centers of forest products, forestry technical services and national forest register
- Wildlife: Convention on International Trade in Endangered Species of Wild Fauna and Flora (CITES) certificates for import and export, management units for wildlife conservation, extractive and non-extractive usage, authorizations, licenses for hunting, animal specimen register, scientific collections and wildlife conservation
- Air: authorizations and procedures for operation and environmental compliance, as well as alternative methodologies for air care and quality improvement
- Environmental impact and risk: environmental impact evaluation is a management instrument that guarantees, when approved, the sustainable development of investment projects, establishing measures to protect the environment and for rational use of natural resources
- Maritime and terrestrial: permit procedures for this zone are the instruments to give the rights to use and exploit beaches, federal zones and land gained to the sea, guaranteeing the organized and sustainable protection, conservation and exploitation for integral development of these zones

SEMARNAT oversees the Official Mexican Standards (Normas Oficiales Mexicanas, or NOMs), which are mandatory technical regulations that establish the rules, specifications, and/or requirements. Key NOMs relevant to the mining operations are listed in

Table 20.2.

Table 20.2 List of Key Official Mexican Standards Applicable to Cerro del Gallo Mining Operations

NOM	Description
NOM-001-SEMARNAT-2021	Wastewater discharges
NOM-003-CONAGUA-1996	Water extraction and well construction
NOM-011-CNA-2000	Water conservation and evaluation of water availability
NOM-035-SEMARNAT-1993	Methodology to measure total suspended particles in air
NOM-043-SEMARNAT-1993	Maximum permissible limits of solid particles from fixed source emissions
NOM-045- SEMARNAT-1996	Maximum permissible limits for opacity of exhaust from vehicles
NOM-052-SEMARNAT-2005	Identification, classification and lists of hazardous waste
NOM-054-SEMARNAT-1993	Procedure to determine hazardous waste segregation
NOM-059-SEMARNAT-2010	Flora and fauna protection, including at-risk species
NOM-080-SEMARNAT-1994	Maximum permissible limits for noise from vehicle emissions
NOM-081-SEMARNAT-1996	Noise emissions
NOM-083-SEMARNAT-2003	Urban solid waste management
NOM-087-SEMARNAT-1995	Medical (biological and infectious) hazardous waste management requirements
NOM-120-SEMARNAT-2011	Environmental protection specifications for mining exploration activities
NOM-138-SEMARNAT/SS-2003	Hazardous waste management requirements
NOM-147-SEMARNAT/SSA-2004	Soil metal contamination management and remediation
NOM-155-SEMARNAT-2007	Environmental protection specifications for gold and silver heap leach facilities
NOM-157-SEMARNAT-2009	Mine waste management plans
NOM-161-SEMARNAT-2011	Special handling waste and management plans

20.2.2 Water Resources

Water resources are regulated under the National Water Law, December 1, 1992 and its regulation, January 12, 1994 (amended by decree, December 4, 1997) and overseen by CONAGUA, the national water authority. Its core mission is to ensure the sustainable management of Mexico's water resources, both surface water and groundwater. In Mexico, ecological criteria for water quality are set forth in the agreement issued by SEMARNAT (Criterios Ecológicos de Calidad del Agua CE-CCA-001/89, or Ecological Criteria for Water Quality, dated December 2, 1989). These criteria are used to classify bodies of water for suitable uses including drinking water supply, recreational activities, agricultural irrigation, livestock use, aquacultural use and for the development and

preservation of aquatic life. The quality standards listed in the regulation indicate the maximum acceptable concentrations of chemical parameters and are used to establish wastewater effluent limits.

In Mexico the water quality standards are linked to usage (that is, as a wastewater discharge or for drinking water). Discharge limits for wastewater have been established according to the receiving waters (NOM-001-SEMARNAT-2021). No groundwater environmental criteria have been established for naturally-occurring groundwater. Permissible limits for potable water parameters have been established in NOM-127-SSA1-2021.

20.2.3 Ecological Resources

In 2000, the National Commission of Natural Protected Areas (CONANP) (formerly CONABIO, the National Commission for Knowledge and Use of Biodiversity) was created as a decentralized entity of SEMARNAT. As of November 2001, 127 land and marine Natural Protected Areas had been proclaimed, including biosphere reserves, national parks, national monuments, flora and fauna reserves, and natural resource reserves.

Ecological resources are protected under the Ley General de Vida Silvestre (General Wildlife Law). NOM-059-ECOL-2000 specifies protection of native flora and fauna of Mexico. It also includes conservation policy, measures and actions, and a generalized methodology to determine the risk category of a species.

Other applicable laws and regulations include: the Forest Law, December 22, 1992, amended November 31, 2001, and the Forest Law Regulation, September 25, 1998.

20.2.4 Expropriations and Land Negotiations

Use and exploitation of goods and land expropriation of “ejido” and communal properties are subject to the provisions of agrarian laws. Ejido land is granted by the Mexican government to a group of people for collective use and cultivation. The following government agencies coordinate surface land management:

- Secretariat for Agrarian Development; Territorial and Urban (SEDATU) is in charge of promoting land ownership legal compliance, especially in rural areas. SEDATU oversees making public policies to access justice and agrarian development.
- National Agrarian Registry controls land ownership of ejidos and communities (communal land owners). This agency oversees all the legal procedures regarding land ownership legalization, issuing of land titles and certificates, regulation of land authorities (ejidos, communities), registration and validation of any process regarding land ownership and ejido member succession lists.

- Agrarian Prosecutor Agency is a social service institution that serves to protect the rights of agrarian individuals. Its services include legal counselling for possession conciliation or legal representation.

20.2.5 Free Trade Agreement

Canada, the United States and Mexico participate in a free trade agreement. The initial agreement, the North American Free Trade Agreement or NAFTA, was replaced in 2020 by the United States-Mexico-Canada Agreement (USMCA). NAFTA addressed the issue of environmental protection, but each country was responsible for establishing its own environmental rules and regulations. The USMCA retains many of NAFTA's provisions but introduces new rules and updates in areas such as digital trade, intellectual property and labor rights.

20.2.6 Permitting Process

Environmental permitting in Mexico requires four primary documents to be reviewed and approved by SEMARNAT:

- Manifestación de Impacto Ambiental (MIA): Mexican Environmental Impact Assessment, including MIA Modifications for any changes to project planning and operations. MIAs describe potential environmental and social impacts that may occur in all stages of the operation as well as the measures to prevent, control, mitigate or compensate for these impacts.
- Estudio Técnico Justificativo (ETJ): technical justification study for the Change in Land Use (Cambio de Uso de Suelo or CUS), that allows the land use to be changed to mining.
- Estudio de Riesgo Ambiental: environmental risk assessment.
- Programa para la Prevención de Accidentes (PPA): accident prevention plan.

Federal environmental licenses (Licencia Ambiental Unica, or LAUs) are issued when the project is approved. The LAU sets out the acceptable limits for air emissions, hazardous waste, and water impacts, as well as the environmental impact and risk of the proposed operation. Figure 20.1 summarizes the environmental permitting process for the authorization of mining operations in Mexico.

Figure 20.1: Overview of Environmental Permitting Process for Mining Operations in Mexico



1 SEMARNAT: Mexican Secretariat of Environment and Natural Resources
2 INAH: Mexican National Institute of Anthropology and History
3 SEDENA: Mexican Secretariat of National Defense
4 DTE: Document that integrates the MIA, ER, and ETJ application in a single submittal, as an optional procedure

Source: Figure prepared by Stantec.

Depending on the level of risk and scope, the MIA and CUS may be evaluated separately in Mexico City, by the state delegation of SEMARNAT in the state where the project is located, or by filing in a unified technical document for land use change (DTU), which is evaluated exclusively at SEMARNAT's main office in Mexico City.

According to the LGEEPA, this evaluating process can take up to 60 business days to be completed but it can be extended for an additional 60-day period at the discretion of DGIRA based on alleged project complexity. The DTU evaluation can also be paused once during the compulsory period if clarifications and/or additional information are requested by one of the government agencies reviewing the document. The key permits required for mine construction, and the legal permitting timeframes, are presented in

Table 20.3, and the key permits for operation and closure are presented in

Table 20.4.

Table 20.3 Key Environmental Permits Required for Mine Construction

Mining Stage	Required formality		Agency	Response time	Comments
CONSTRUCTION	OPTION 1	Environmental Impact Manifest (MIA)	SEMARNAT	3-6 months	Baseline studies should be conducted to support the MIA. A comprehensive environmental manifest shall be prepared and submitted to SEMARNAT for evaluation and authorization.
		Land Use Change Study (ETJ)	SEMARNAT	2-3 months	A detailed forestry inventory and a technical study shall be prepared and submitted to SEMARNAT for evaluation and authorization.
		Risk Analysis Study (ER)	SEMARNAT	3-6 months	A risk analysis shall be prepared and submitted and will be evaluated together with the MIA, when high risk substances such as cyanide is used in the process.
	OPTION 2	Integrated Technical Report (Documento Técnico Unificado or DTU)	SEMARNAT	3-6 months	A comprehensive technical document that integrates information of the MIA, ER and ETJ should be prepared and submitted to SEMARNAT for evaluation and authorization.
	Land Use/construction License		Municipality	1 month	An application letter shall be submitted to the municipal authorities to obtain the authorization letter.
	Permit for disposal of non- hazardous residues		Municipality	1 month	An application letter needs to be submitted to the municipal authorities, specifying the expected type and amount of non-hazardous waste from the mine construction and operation.
	Explosive handling		SEDENA, Municipality and State Government of Sonora	3 months	An application letter shall be submitted to SEDENA. Also an endorsement letter shall be obtained from the State Government and the Municipality.
	Cultural and archeological clearance		INAH	1 to 8 months	A request letter should be submitted to INAH to have the project checked for cultural and archaeological sites. A survey will be done by INAH personnel and, if there is some cultural or

				archeological interest, a rescue and documentation program will be performed.
	Water use concessions	CONAGUA	3 months	Water extraction requires a water use concession from CONAGUA, which indicates a total extraction volume permitted annually. CONAGUA will be notified once the water usage begins. The volume of water to be used in the mining activities should be measured and paid.

Source: KCA (2020)

Table 20.4 Permits Required for Mine Operation and Closure

Mining Stage	Required formality	Agency	Response time	Comments
OPERATION	Water discharge permit	CONAGUA	3 months	An application needs to be filed with CONAGUA with estimated annual volume and the quality of the discharge. This may include the sanitary service water discharge or any other water discharge to septic tanks or the environment.
	Operation license	SEMARNAT	2 to 4 months	Requires an inventory of air emission sources, water discharges and solid wastes.
	Accident prevention plan	SEMARNAT	None	Based on the risk analysis, it is necessary to establish a plan and procedures to prevent and respond to emergencies and accidental events. SEMARNAT will register this plan.
	Mining residues management plan	SEMARNAT	None	Need to prepare this plan according to NOM-157-SEMARNAT-2009. SEMARNAT will register this plan.
	Hazardous waste generator registry	SEMARNAT	None	Requirement to keep records of any hazardous waste movement at the mine facilities and deliveries to an authorized external company.
CLOSURE	Closure and reclamation plan	SEMARNAT	Not specified	Need to submit a comprehensive closure and reclamation plan, typically 6 months before mine closure.

20.2.7 Environmental Permitting History

The Project has not received any environmental permits. A previous owner had advanced the mineral development project to the point of submitting applications for environmental authorization, however, the permitting authority requested additional information to be submitted due to deficiencies found in the application. A history of permit submissions made by previous site developers and the subsequent denials is summarized below. The QP notes that the permitting documentation prepared by the previous owner has only been partially available for review; the Company is in the process of attempting to locate historic studies, environmental permitting documents and correspondence from the environmental authority.

20.2.7.1 Historic Permitting

San Antón de las Minas, S.A. de C.V., a subsidiary of Argonaut, submitted a permit application in April 2019 for a Unified Procedure for Change of Forest Land Use (DTU) under the medium-grade regional modality B (DTU-BR and ERA) for approval to use the land associated with the Project for the purposes of mining. The submission included legal, technical, and environmental documentation, such as proof of land ownership and risk assessment study. In December 2019 SEMARNAT denied authorization of the permit. The reasons for denial were inconsistencies and deficiencies in the environmental impact and forestry land-use change documentation, including unclear Project descriptions, mismatched surface area data, inadequate environmental studies, and insufficient mitigation plans. Multiple government agencies, including local and state authorities, issued unfavorable opinions, citing risks to biodiversity, water resources, and local livelihoods. The resolution for denial concluded that the Project would violate several environmental laws and regulations. However, San Antón de las Minas, S.A. de C.V. could resubmit a revised application in the future.

In August 2019, San Antón de las Minas, S.A. de C.V. submitted a request to withdraw a previously submitted regional Environmental Impact Manifest (MIA) for the Project. The request was based on preliminary exploration results indicating that one area lacked mining potential and would be repurposed, while another area with viable potential would be retained in the environmental impact footprint. Additionally, new metallurgical testing was required, which could significantly alter the Project scope and its associated environmental impacts. As a result, San Antón de las Minas, S.A. de C.V. requested that the original MIA evaluation be withdrawn.

In June 2020, San Antón de las Minas, S.A. de C.V., submitted a revised request for approval of a DTU. It is the QP's understanding that no response was received from SEMARNAT.

In November 2020, a new Regional MIA (MIA-R) was submitted, and then in 2021 SEMARNAT denied the MIA-R and the Environmental Risk study (ER). The denials were based on lack of critical technical information, including detailed environmental studies, risk assessments, and compliance with land-use and ecological restoration policies. SEMARNAT stated that the applicant failed to provide sufficient data on biodiversity, hydrology, and mitigation strategies, leading to concerns about potential impacts on water, soil, air, and local ecosystems. The Project was deemed incompatible with regional planning instruments and faced social opposition. As a result, SEMARNAT concluded that the Project did not meet legal and environmental standards and formally denied its approval, though the applicant retained the right to appeal or resubmit with the required information. It is the QP's understanding that San Antón de las Minas, S.A. de C.V. did

not resubmit the application, and that SEMARNAT would consider this application as closed.

The permitting history is shown in Table 20.5, based on the available information provided to the QP.

Table 20.5 Environmental Permitting History

Permit type and identification	Submittal Request	Date	Comments
Environmental Impact Manifest (MIA), 09/DL-0270/08/18	Request to withdraw the MIA	Application submittal date is unknown. Request to withdraw made on August 20, 2018.	One area lacks mining potential and will be repurposed; the other shows potential and will be retained
Unified Procedure for Change of Forest Land Use (DTU-BR) and Environmental Risk Study (ERA) SGPA/DGIRA/DG/09968	Surface area of 370.32 ha	Applications for DTU-BR and ERA submitted on April 26, 2019. Additional information submitted August 29, 2019 and application denied on December 17, 2019.	Denied due to inconsistencies and deficiencies in environmental and land-use documentation
Unified Procedure for Change of Forest Land Use (DTU-BR)	Requested approval for a DTU under a medium-grade regional modality B	Application submitted June 5, 2020. No response received from SEMARNAT.	Request for CDG project involving high-risk activities with supporting legal, technical, and environmental documents
Environmental Impact Manifest Regional Modality (MIA-R) and Environmental Risk Study (ER) SGPA/DGIRA/DG/04483	Open pit gold, silver, and copper mine. Surface area of 385.26 ha	Application submitted November 12, 2020. Authorization denied on November 14, 2021.	Project denied due to missing technical data, environmental risks, and non-compliance with regulations. Argonaut did not respond with an additional information submittal.

The Company has developed a new permitting strategy and schedule for its project development, as discussed in the following section.

20.2.7.2 Current Environmental Permitting

Argonaut via its Mexican subsidiary, contracted with Herrera and Sato, a consulting company based in Mexico City, to carry out environmental baseline studies during the period of 2021, 2022 and 2023, resulting in the partial preparation of a new regional MIA and risk assessment evaluation (MIA-R and ERA). The QP notes that this work was carried out prior to the 2023 Mining Law Reform, and that Argonaut Gold via its subsidiary San Anton de Las Minas filed an injunction against these reforms; however, the suspension has not had a final ruling, so it is not known whether new requirements will be applied to the work carried out for the MIA-R and ERA.

The surface drainage system adjacent to the proposed mine facilities will need to be evaluated to determine if it is classified as a federal surface water zone per CONAGUA requirements. Should the drainage be a federal zone, any disturbance to the zone will need to be permitted. The Company has contracted a consultant, SIIMA, to conduct a study to delineate the federal zone. The work is anticipated to be completed by the end of October 2025. If a federal zone is identified, then the mine design would include a diversion channel for the drainage. The Company plans to restart the permitting process after the SIIMA study has been completed.

The Company plans to review the MIA-R information collected to-date and subsequently decide whether the data is sufficient to prepare a new application, or whether new or additional information will be needed for environmental permitting. The Company has assumed a permitting schedule starting in early 2026 and requires 24 months for data acquisition, preparing the MIA application, submitting the application, responding to one additional data request from SEMARNAT, and receiving the authorization

20.3 Water Concessions

The Project obtained eight water concessions under Argonaut, however five of the concessions expired in November 2024 due to lack of payments. The Company has discussed with CONAGUA the possibility of reactivating the six expired concessions, and is waiting to determine whether additional concessions are needed meet the project's water demand before taking any actions. The total concession volume currently permitted and active is 640,000 m^3/year. The water concession information is provided in

Table 20.6.

Table 20.6 San Antón de las Minas Water Concessions

Concession Identifier	Start Date (YYYY-MM-DD)	Expiration Date (YYYY-MM-DD)	Status	Location		Extraction Well Associated with Concession	Permitted Extraction Volume
				Latitude	Longitude		m^3/year
08GUA110394/12FMDL16	2018-12-04	2028-12-04	Active	21°04'02.0000"	100°59'27.2000"	SAM 1	140,000
08GUA123064/12FMDL12	2019-12-17	2029-12-17	Active	21°04'02.0000"	100°59'27.2000"	SAM 1	300,000
08GUA154831/12FMDL14	2014-11-02	2024-11-02	Lapsed	21°04'02.0000"	100°59'27.2000"	SAM 1	130,000
08GUA154832/12FMDL14	2014-11-02	2024-11-02	Lapsed	21°04'07.2000"	100°59'15.0000"	None	130,000
08GUA154833/12FMDL14	2014-11-02	2024-11-02	Lapsed	21°05'14.8000"	101°00'00.0000"	None	140,000
08GUA155418/12FMDL16	2013-06-05	2023-06-05	Lapsed	21°04'02.0000"	100°59'27.2000"	SAM 1	250,000
08GUA155423/12FMDL16	2018-08-13	2026-08-13	Active	21°04'02.0000"	100°59'27.2000"	SAM 1	200,000
08GUA155432/12FMDL16	2013-12-04	2023-12-04	Lapsed	21°04'02.0000"	100°59'27.2000"	SAM 1	150,000

20.4 Baseline and Supporting Environmental Studies

Environmental baseline data were collected for multiple seasons to support an initial environmental impact assessment application that was submitted in 2019, and additional data were collected in 2021, 2022 and 2023. The most recent baseline environmental information available (Herrera y Sato, 2024) has been used to understand the environmental setting, which is described below. The environmental baseline studies covered fauna, flora, water, and air quality.

The Project is not within any designated environmental protection areas.

20.4.1 Fauna

Field surveys, transects, and existing literature were used to estimate faunal diversity in the Project area that would require an environmental permit to change the land use from forest to mining. The fauna diversity results indicated that birds were the most diverse group of species, followed by mammals. Amphibians and reptiles had the lowest species counts.

A total of 12 mammal species from nine families were recorded (Didelphidae, *Leporidae, Sciuridae, Heteromydae, Felidae, Canidae, Mephitidae, Procyonidae,and Cervidae)*. Due

to the elusive nature of mammals, indirect observations based on tracks and droppings were most frequently noted. Camera traps captured three species: raccoon (*Procyon lotor*), coyote (*Canis latrans*), and gray fox (*Urocyon cinereoargenteus*). Direct sightings included black-tailed jackrabbit (*Lepus californicus*) and rock squirrel (*Otospermophilus variegatus*). The most frequently detected species was raccoon, with tracks found in 19 out of 21 transects, followed by signs of coyote in 12 transects.

The bird survey recorded 43 species from 21 families and 8 orders, representing 16% of the potential species for the area and 12% of those known in the state of Guanajuato. The most represented families were Passerellidae (sparrows) and Tyrannidae (flycatchers), each with six species. Most birds were resident species (33), while 10 were winter migrants. Four species were identified as semi-endemic to Mexico, meaning their entire population winters in the country. No bird species were listed under NOM-059-SEMARNAT-2010, which lists protected species in Mexico. The most abundant species was the American white pelican (*Pelecanus erythrorhynchos*), accounting for 19% of sightings, followed by the bushtit (*Psaltriparus minimus)*, lark sparrow (*Chondestes grammacus*), house finch (*Haemorhous mexicanus*), and Cassin's kingbird (*Tyrannus vociferans)*. There are no permanent water bodies at the Project, and no fish species were noted during the baseline studies.

20.4.2 Flora

In the Project area and the larger zone of direct influence, a total of 151 plant species were identified, spanning 40 botanical families. This reflects a high level of biodiversity, indicating a rich and varied ecosystem. Vegetation was categorized into four main strata that each contributes uniquely to the ecological structure and function of the region: arboreal (trees), arbustive (shrubs), cacti and agaves, and herbaceous plants. A limited number of species within the cacti and agave group are listed under NOM-059-SEMARNAT-2010, Mexico's official list of protected species, highlighting the importance of conservation efforts in this area.

20.4.3 Arboreal (Trees) Stratum

The tree layer recorded 10 species and a total of 652 individual trees. The most abundant species were Palo dulce (*Eysenhardtia polystachya*), Encino manzano (*Quercus eduardi*), Roble (*Quercus resinosa*), and Encino quiebra hacha (*Quercus rugosa*), which together comprised the majority of the 652 trees documented. This layer plays a crucial role in providing canopy cover and habitat for other organisms. Notably, none of the tree species are currently listed under the NOM-059-SEMARNAT-2010 protection status.

20.4.4 Arbustive (Shrubs) Stratum

This layer was the most abundant in terms of individual plants, with 46 species and 6,671 individuals recorded. Dominant species include Ocotillo (*Dodonaea viscosa*) and Palo dulce (*Eysenhardtia polystachya*), both showing high population numbers. It is noted that Palo dulce is listed as both a tree and a shrub based on its height at the time of the field survey. Despite the high abundance and diversity, no shrub species were found to be under any conservation status according to NOM-059-SEMARNAT-2010, suggesting a relatively stable shrub community.

20.4.5 Cacti and Agave Stratum

Within this stratum, 13 species and 365 individuals were documented. The most common species was Tepenexcomite (*Echinofossulocactus pentacanthus*), which is a small, globe-shaped cactus, with 151 individuals. Three species in this group are designated as protected species: *Mammillaria rettigiana* (Special Protection), *Mammillaria zephyranthoides* (Threatened), and *Dasylirion acrotrichum* (Threatened). This highlights the ecological sensitivity of this stratum and the need for targeted conservation measures.

20.4.6 Herbaceous Plants Stratum

The herbaceous stratum was the most diverse, with 83 species recorded. The species with the highest ground coverage was Pasto rosado (*Melinis repens*), indicating its dominance as a ground cover. Despite the exhibited diversity, none of the herbaceous species are listed under NOM-059-SEMARNAT-2010, suggesting that this stratum is not currently at risk but still plays a vital role in soil stabilization and ecosystem productivity.

20.5 Soils

In the Project area, Leptsol was the only soil type present. Also known in other classifications as Litosols and Rendzinas, these soils are very shallow, stony, and poorly developed, often containing a high concentration of calcareous material. They are found across all climate types—dry, temperate, and humid—and are particularly common in mountainous regions and shallow limestone plains.

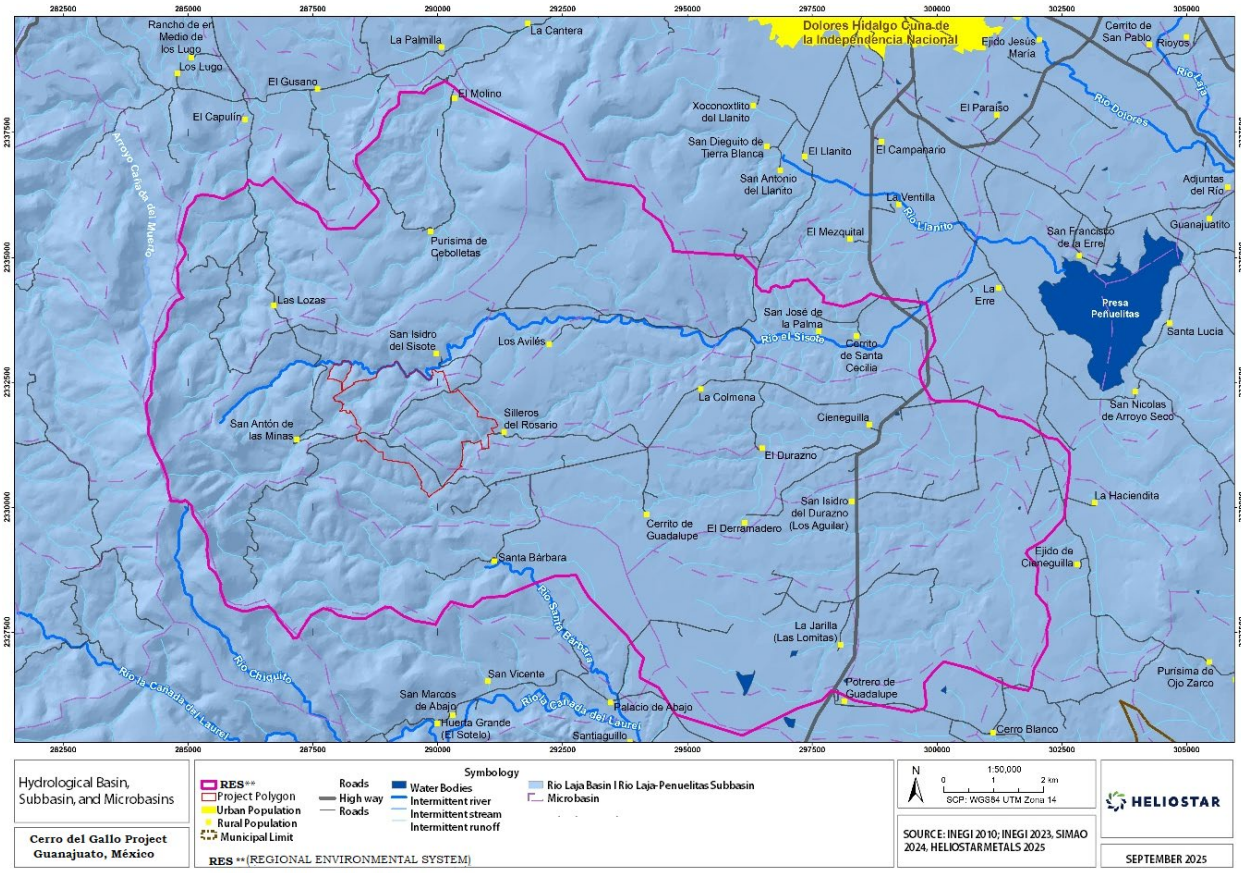
The QP notes that the presence of calcareous material may be beneficial in closure planning.

20.6 Hydrologic and Hydrogeologic Setting

The Project is in the hydrologic basin “Rio Laja” and the subbasin Rio Laja-Peñuelitas. “Cuenca Alta del Rio La Laja”. There are no perennial streams or water bodies within the

Project. The Peñuelitas Reservoir, located southeast of the town of Dolores Hidalgo and about 13 kilometers (km) from the Project, is the closest surface water body (Figure 20.2). The dam has a storage capacity of 14 cubic hectometers (hm^3) and is primarily used for irrigation.

Figure 20.2: Surface Hydrology



The Project is in the aquifer “Cuenca Alta del Rio Laja”, which covers a surface of 3,000 km. The aquifer is described as a granular body formed by alluvial deposits. Regionally, it is unconfined or semi-confined. CONAGUA has classified the aquifer as over-exploited with a deficient of 61M cubic meters (CONAGUA, 2024).

No local hydrogeologic studies have been made. Site personnel have noted that the historic underground mine has flooded, but the depth-to-water has not been determined.

20.7 Surface Water and Groundwater Quality

Surface waters in the Project area occur as ephemeral streams with water flow only during storm events and primarily during the rainy season. Local residents build small retaining ponds along drainages as sources of water for livestock and agriculture.

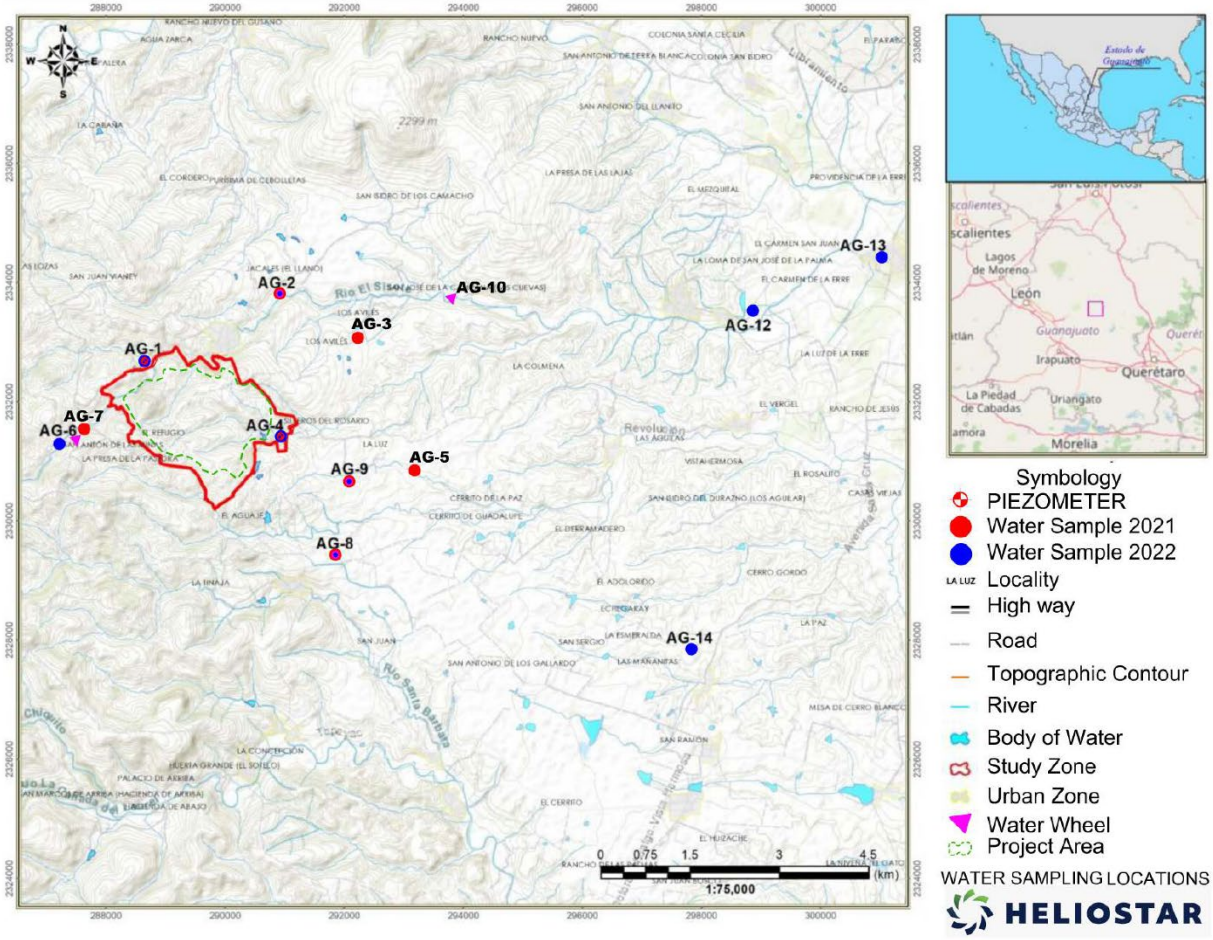
As part of the Project environmental baseline studies a water study was carried out by Siima, contracted by Argonaut, during 2021, 2022, and 2023. The findings were presented in the report titled “Hydrogeological and Baseline Study for Cerro del Gallo” (Estudio Hidrogeológico y de Línea Base para Minera Cerro del Gallo) (Siima, 2023).

Water quality samples were collected from monitor wells, potable water wells for the different communities, and “norias” and some retention ponds (Figure 20.3). A noria typically refers to an agricultural well and could be hand-dug. The report did not include descriptions or photographs of the sampling points.

Three sampling events were conducted during different seasons during the period of 2021 through 2023 to assess both surface and groundwater:

- Nov 2021 – Rainy Season: nine water samples (three wells, two norias, four streams)
- June 2022 – Dry Season: nine samples (six wells, two piezometers, one noria)
- Oct 2023 – Rainy Season: nine samples (three wells, two piezometers, two norias, two streams)

Figure 20.3: Water Sampling Locations



Source: Heliostar, 2025.

Each sampling event included field measurements of temperature, pH, and electrical conductivity, plus laboratory analysis for parameters such as major ions (bicarbonates, calcium, magnesium, sodium, sulfates, chlorides) and trace elements (arsenic, lead, iron, zinc, copper), plus nitrates, fluorides, total dissolved solids (TDS), hardness, and E. coli. In the report, the surface water and groundwater results were compared to the limits set by Mexican standard NOM-127-SSA1-2021 for drinking water. The QP notes that this standard is more stringent than the parameters in NOM-001, which regulates discharges to surface water bodies.

Surface water quality samples results indicated that most samples were within the limits set by Mexican standard NOM-127-SSA1-2021 for drinking water, except for TDS, hardness and E. coli. Seasonal variation was observed, with higher concentrations of some parameters (hardness, sulfate, TDS, electrical conductivity, magnesium, and chlorides) during the dry season, likely due to reduced water flow and increased mineral

contact. During the first and second sampling events (2021 and 2022), all surface water samples were within the permissible limits established by NOM-127-SSA1-2021. During the 2023 sampling event, the San Antón stream sample exceeded the limits for iron and E. coli, and the San Isidro stream sample exceeded the limit for E. coli.

Groundwater quality sample results included exceedances of the Mexican drinking water permissible limits for arsenic in one sample (AG-8) and TDS in two samples (AG-2 and AG-9) in 2021; no exceedances in 2022; and in 2023 one exceedance for hardness (AG-6), three exceedances for iron (AG-6, AG-1, AG-4 and AG-2), one exceedance for lead and zinc (AG-2) and two exceedances for E. coli (Ag-08 and water well San Jose).

The presence of certain elements in the water samples from the Project area may be influenced by both natural and anthropogenic factors. Elevated levels of iron and arsenic are likely linked to the natural geological composition of the region, particularly given the mineralization in the local area. These elements are commonly found in mineralized rock formations and can leach into groundwater through natural geochemical processes. In contrast, the exceedances of lead and zinc observed in sampling point AG-2 during 2023 suggest a case of localized contamination, potentially from previous mining activities or infrastructure. Additionally, the detection of E. coli in surface water and shallow wells is most likely due to uncontrolled animal or human waste, especially during the rainy season, when livestock grazing and inadequate waste management can lead to microbial contamination of surface water sources. E. coli detected in a water well is indicative of an improper well design or poor construction that allowed surface water contamination to migrate to groundwater.

20.8 Air Quality

Air quality monitoring for total suspended particulates per NOM-035-SEMARNAT-1993 and particulates per NOM-025-SSA1-2021 was carried out by a certified laboratory (Herrera y Sato, 2024). NOM-035 applies to particles up to 50 micrometers (µm) in diameter, whereas NOM-025 measures particle sizes of less than less or equal to 2.5 µm in diameter, or PM2.5, and less than or equal to 10 µm in diameter (PM10).

Prior to the 2023 sampling for PM2.5 suspended particles, a site survey was carried out to identify potential pollutant sources, adjacent land uses, and prevailing wind directions and intensities determine placement of monitoring equipment. The selected sampling locations were:

- Point 1: North of El Sisote town – potential location for waste rock dumps
- Point 2: West of San Antón de las Minas town – potential site for the open pit
- Point 3: East of Silleros del Rosario town – potential site for the process plant

- Point 4: South of access road to San Antón de las Minas

The 2023 testing results for total suspended particulates, PM2.5 particulates and PM10 particulates were all within regulatory limits.
The QP notes that the results provide a background level based on one event and do not provide background information based on a variety of weather conditions.

20.9 Mining Wastes

A geochemistry study was conducted to evaluate the environmental stability of waste rock and leached ore (SRK Consulting, 2016). The program focused on determining the potential for generation of acid rock drainage (ARD) and metal leaching (ML) based on static and kinetic testing laboratory test methods:

- Waste rock analysis per NOM-157-SEMARNAT-2009. This regulation requires analyzing each sample (dry base) for 10 elements (antimony, arsenic, barium, beryllium, cadmium, chrome, mercury, silver, lead and selenium). If the total concentration of any of these elements is above the NOM-157 permissible limits, a mobility procedure test (that is, a meteoric water mobility test) is also required. Any waste produced during the mining process, such as waste rock, must also be analyzed for acid generation potential using the acid-base accounting (ABA) test.
- Leached ore analysis per NOM-155-SEMARNAT-2007. The regulation for heap leach facilities requires the meteoric water mobility test to characterize spent ore.
- Hazardous waste characterization per NOM-052-SEMARNAT-2005. This regulation establishes hazard characteristics (corrosivity, reactivity, explosiveness, flammability, toxicity, and biological-infectious potential), the identification procedure, and the listings of hazardous wastes in Mexico.

The geochemistry program used samples and/or data generated at the Project under various previous owners, including Kings Minerals, Primero Mining and Argonaut Gold.

20.9.1 Geochemistry Characterization Program

The geochemistry program was conducted in two phases by SRK Consulting during 2013 and 2016. During Phase I, 50 samples (and five duplicates) were selected from exploration drill cores, based on anticipated waste rock volumes and spatial distribution (SRK Consulting, 2016). In addition to the Phase I samples, historic data were provided by the Company for 12 samples (six ore and six waste rock) from a geochemistry program carried out by Kings Minerals in 2011 for hazardous waste classification, which included short-term leaching potential and ABA analyses.

Based on the Phase I program results, a subsequent Phase II geochemical program was developed and implemented to generate kinetic data for waste rock lithologies. Ten samples (plus one field duplicate) were analyzed using the static methods listed above and simultaneously submitted for 40-week humidity cell tests (HCT) to develop data on the onset of acidic conditions, leachate chemistry and depletion rates of acid generating and neutralizing minerals. The Phase II program also included:

- Static analysis of four samples that are residue from the metallurgical leach test to determine if spent ore will be a hazardous waste according to Mexican regulations.
- Static and kinetic analyses of nine mineral samples representing ore to determine their acid generation potential during heap leaching. This information was requested by the metallurgical team, as acid generation during heap leaching will consume the base required to maintain the cyanide leaching solution within operational parameters.
- Static analysis of eight weathered surface samples of tuff and felsic rock to gain an understanding of the final products of waste rock oxidation and potential impact to waste rock drainage chemistry.
- Static analysis of nine shallow samples and one composite sample of near surface waste rock.
- In June 2016, additional environmental samples were reanalyzed for calcium and sulfur assay data as surrogates for neutralization and acid generating potential.

20.9.2 Results

Results from the Phase I and II geochemical testing of the San Anton waste rock indicate that most samples from key lithologies (tuff, felsic intrusive, and silicified intrusive) are potentially acid generating (PAG) or fall into an uncertain category, with about 30% being non-acid generating due to low sulfide content. The static test results indicate that 70% of the waste rock will be classified as a hazardous waste and requires special handling based on Mexican regulations for mining waste.

Phase II static and kinetic analyses of the major waste rock lithologies (felsic and tuff) provided further evidence that the waste rock and leached ore samples could be acid generating. Samples from the upper 40 meters of the deposit tend to have lower acid generating potential due to weathering of the sulfides. The neutralization potential (NP) would be depleted in one to 17 years. After the NP is depleted, the pH could decrease, and the rate of acid generation and metal leaching could increase.

Whole rock metals analysis indicated that the total concentrations of antimony, arsenic, cadmium, chromium, lead, mercury, selenium, and silver in the waste rock are significantly elevated compared to global crustal averages for low-calcium granite.

Results of the leachate tests and total metals concentrations are indicative of a strong potential for metal leaching.

Results of the short-term leachate tests indicate that only one sample exceeded the non-hazardous permissible limits for mining wastes (NOM-157-SEMARNAT-2009) for selenium (5.63 mg/L); however, the result was attributed to lab error. Two of the Kings Minerals samples from 2011 exceeded the NOM-157-SEMARNAT-2009 for cadmium; however, the quality of these data were considered to be unknown and no other sample exceeded the cadmium limits. The geochemistry study concluded that waste rock is not a hazardous waste based on short term leaching tests, although the majority of the waste rock is PAG or uncertain. Metals leaching from waste rock may be problematic. For example, lead was leached at a concentration rate of 1.39 mg/L in a felsic intrusive sample; the permissible limit for discharge to the environment is 0.2 mg/L. The study also indicated that the results from weathered samples of felsic intrusive and tuff indicated that the unoxidized samples are PAG and have the potential to leach metals; however, weathered samples appear to have undergone complete oxidation of pyrite (the primary source of acid generation) and the weathered samples were non-PAG. To mitigate PAG, the addition of 281,000 to 468,500 tonnes of limestone—sourced from a nearby deposit in the Esperanza Formation—was recommended to manage the WRSF. Blending calcite into the waste rock could neutralize acid generation and reduce metal leaching.

Static analysis of nine composite samples of spent ore confirmed that all samples are potentially acid generating and capable of leaching metals, requiring classification and management as hazardous waste under NOM-052-SEMARNAT-2009.

Kinetic testing of the nine composite ore samples from Phase I revealed that most ore types have acid-generating potential, although not all became acidic during the 40-week test period. The NP depletes within a few months to two years, indicating that acid generation could pose challenges for environmental protection features of the design during operations and closure. The metals leaching trends in general started high and then decreased gradually over time or stabilized. The concentrations of some metals were elevated, in particular arsenic, cadmium, and selenium, and could be problematic for discharge to the environment.

The QP notes that additional geochemical testing is recommended to refine and confirm the geochemistry study results and to develop mitigation measures for operational and closure purposes.

20.10 Environmental Management Requirements and Plans

20.10.1 Mining Wastes Management

The operating mine is planned to generate waste rock from the open pit mining operations and spent ore from the heap leaching operations. A detailed monitoring plan will need to be developed during final design of the HLF and WRSF to assure that these facilities are developed, operated, and monitored in accordance with the intent of the design and to ensure environmental protection. In addition, Mexican regulations will require the mining waste management plan to be submitted and approved by the environmental authority.

With respect to the HLF and WRSF, the monitoring plan will include:

- Monitoring of solution flows into and out of the process ponds, and monitoring of pond elevations.
- Monitoring the Pregnant and Event Ponds leak detection systems to monitor that fluids are not passing through the pond's containment.
- Monitoring the leach pad leak detection system to monitor that fluids are not passing through the primary pad containment.
- Monitoring the quality of surface water and groundwater around the HLF and WRSF to confirm that surrounding waters are not being impacted by mining and processing activities.
- Monitoring of the seepage from the WRSF that is recirculated to the process.
- Routine sampling of the HLF and WRSF, as required by conditions set in the environmental permit.

Other monitoring that may be determined as critical during the final design phase of the Project, which may include settlement monitoring of deep, select waste rock fills or movement monitoring of critical slopes.

20.10.2 Hazardous and Non-Hazardous Waste Management

Special wastes, such as waste oil, glycol coolant, solvent fluids, used oil filters, used batteries, and contaminated fuel, will be handled, stored, transported, and disposed of in accordance with appropriate hazardous waste regulations. The Mexican environmental authority will require that a plan that describes how the hazardous and regulated wastes are managed be prepared and authorized. The preliminary management scenario includes the following:

- A fenced temporary storage facility for hazardous waste will be constructed. A roofed storage area will be designated for used batteries, used lubricants, coolant and other miscellaneous fluids, and used tires. A certified transport and disposal company will collect all waste to transport offsite for final disposal.

- A site for temporary storage of recyclable materials will be established. Such items as scrap metal, tires, glass, recyclable plastics and drink containers will be separated, containerized as appropriate, and temporarily stored until sufficient volumes are available for shipment to a recycling point.
- Non-recyclable and non-hazardous waste will be managed with a dedicated local company and waste will be sent to the Dolores Hidalgo municipal landfill. A domestic waste removal service in the San Antón de las Minas community currently exists on a weekly basis. The Project will contract domestic waste disposal services per the current established infrastructure.
- A location on the mine site will be designated as an outdoor storage or 'boneyard' area for placement of items that are not yet ready for disposal, but which may still be of use for spare parts. These items are likely to include equipment parts, vehicles, and pieces of equipment, and metal components. Materials remaining in the boneyard at the end of mine life will either be shipped off site for salvage value, recycled, or disposed of in the landfill if they meet the criteria for disposal at that location.

20.10.3 Wastewater Management

The wastewater disposal systems for office areas will be engineered, constructed, and maintained under the direction of a qualified professional.

20.10.4 Air and Noise Emissions Management

Since the mine has not yet been constructed, there are currently no emissions of smoke, dust, or noise associated with mining activities. However, once operational, emissions are expected to result from machinery and equipment use, as well as from ore and waste rock haulage (via trucks and conveyor belts), road operations, and vegetation clearing. These activities are anticipated to be the primary sources of dust emissions. Dust suppression measures, such as the use of water trucks on roads, are planned to mitigate these impacts.

20.10.5 Water Management

Water diversion structures will be constructed to keep surface water from flowing into the HLF, mine pits, waste dumps and other operational areas. Surface drainage from disturbed areas that have no potential to produce chemical or metal contamination will be directed into small ponds to allow sediments to settle out before discharging to the environment.

Sampling of surface water drainages in the Project area and monitoring wells will need to be carried out through the life of the mine, including the reclamation period and post-

closure, until it has been determined that reclamation has been successful in preventing long-term impacts to surface waters.

20.11 Socioeconomic Studies and Impacts

The town of Dolores Hidalgo is the largest nearby community, located about 15 miles (24 km) away from the Project, and has an important place in Mexican Independence history. Dolores Hidalgo is designated as a “Pueblo Mágico” (Magical Town) by the Mexican government. There are two sites in the region (Sanctuary of Atotonilco and San Miguel de Allende) that are designated as United Nations Educational, Scientific and Cultural Organization (UNESCO) World Heritage Sites for their cultural and architectural significance. The sites are about 40 miles (62 km) from the Project. The town of San Miguel de Allende has a large expat community composed of foreigners from a wide range of countries. The expat community is known to have concerns regarding water availability in the State of Guanajuato. The State of Guanajuato is a popular tourist destination.

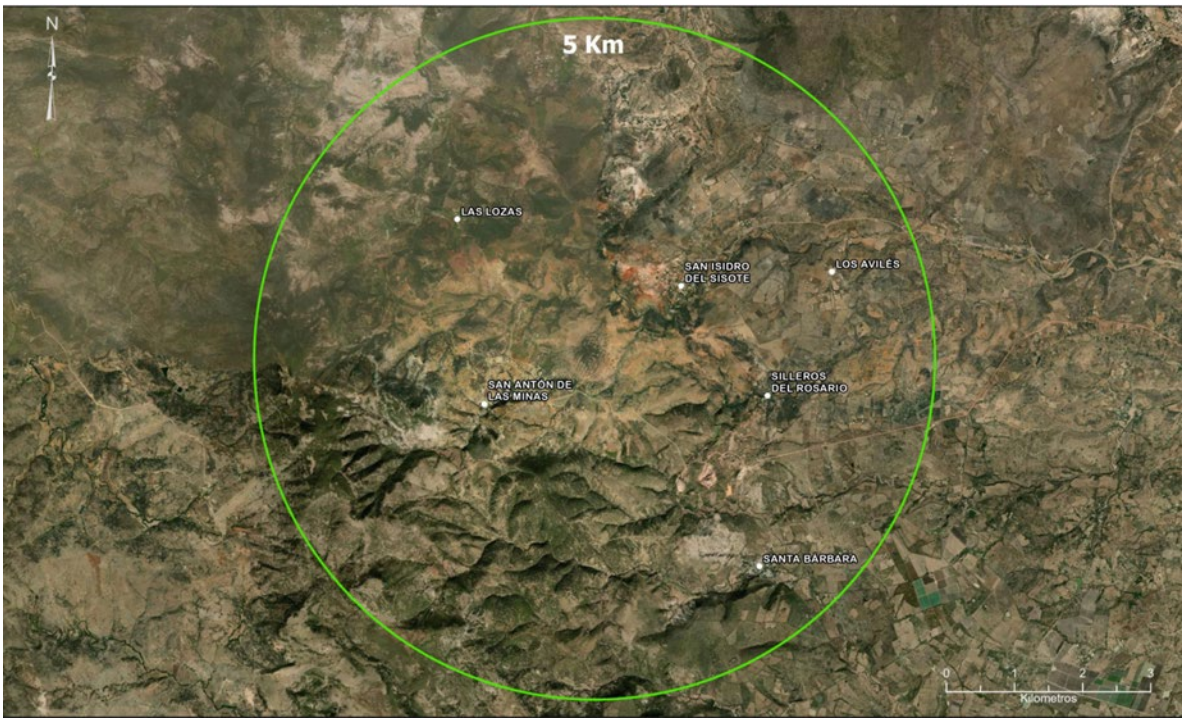
The last social baseline study was carried out in 2022 (McBride Sustainability). The study identified six communities that would be directly impacted by future mining operations (

Table 20.7 and Figure 20.4). The communities are all located within the Dolores Hidalgo municipality. Social surveys were carried out in each of the six communities, plus La Colmena, which is another small community further east than the radius of influence. The survey gathered information on social demographics, economics, education, health, mobility, the environmental, general wellbeing and perspectives.

Table 20.7 Communities Directly Impacted

Community	Number of Inhabitants	Number of Homes
San Antón de las Minas	550	138
Las Lozas	106	27
San Isidro del Sisote	441	110
Los Avilés	80	40
Silleros del Rosario	132	35
Santa Bárbara	493	123

Figure 20.4: Communities of Direct Impact



Source: Heliostar, 2025, filename: SAN ANTON DE LAS MINAS_GCSR.docx

The Project is in a rural area. The closest health clinic to the site is located in the town of San Anton de las Minas; however, it offers limited services. The closest hospital is in Dolores Hidalgo. Most of the direct impact communities have community centers and all have water and electricity. None have a sewage system. Cooking is done by about an equal combination of natural gas and wood. There are no Indigenous rightsholders. The problems most mentioned during the survey were lack of pavement (39%), lighting (19%), sewage (11%), lack of garbage service (3%) and water (0.6%). Lack of water and low water pressure were indicated, and few people considered the water source to be contaminated (5%). The peaceful environment was cited 45% as being the most enjoyed aspect of the area.

Five of the direct impact communities have kindergarten, primary and secondary schools (up to ninth grade, or 15 years old). It is estimated that 90 percent of adult males leave the communities after secondary school to work in the United States. Many of the households depend on income sent from the workers in the United States. Agriculture and livestock are the primary employment, with stores, commercial businesses, and bakeries as minor employment. Agricultural products include beans, avocadoes, limes, chiles, corn and squash. Much of the agriculture is for personal consumption and sustenance living. The majority of persons interviewed stated that there is no opportunity for employment and that the salaries are poor. About 80% of the persons interviewed do not have a paid job.

All the local communities have communal government (ejido) properties, although the Project is not located on ejidal property. The population identifies almost 100 percent as Roman Catholic, and the local priest based in San Antón de las Minas is an important interested party. The Company has identified the priest at the church in San Anton de las Minas as a neutral party regarding the Project.

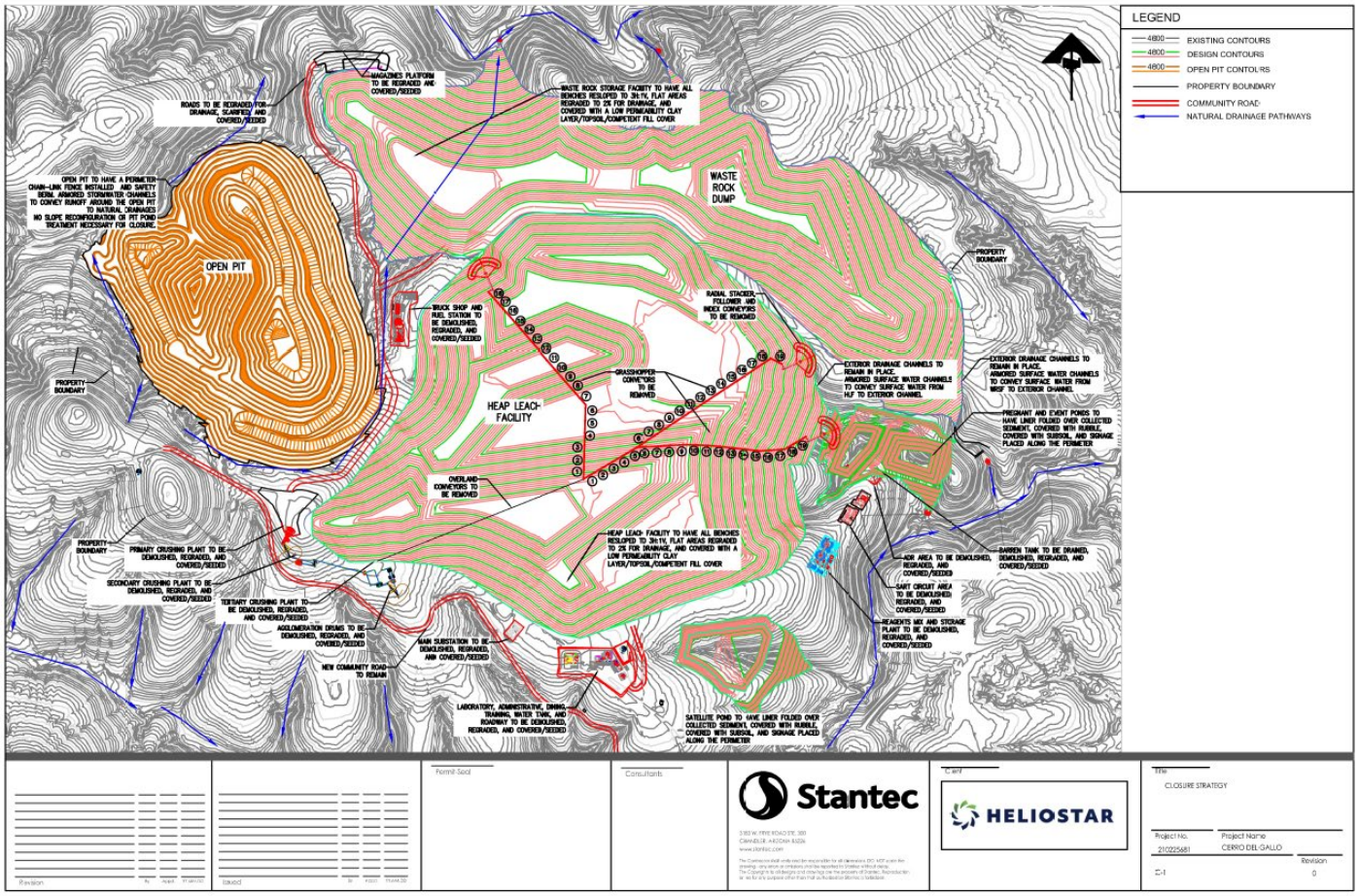
The Company has established agreements with several community members to use Company land for agriculture and livestock grazing.

The Company has an active social program for 2025 that includes activities such as participation in community events, educational workshops, community infrastructure improvements, donations of goods and services, and support of Red Cross activities. An update of the baseline study for the impacted communities will also be carried out. Some of the workshops will focus on environmental issues and human rights.

20.12 Closure and Reclamation Plan

Closure objectives include securing the site for physical safety of people, protecting wildlife, protecting surface and groundwater quality and quantity, minimizing erosion and controlling fugitive dust. Permanent impacts will be the open pit, the WRSF and the HLF. The effects of mining are irreversible, although through planned restoration and revegetation methods, some effects will be improved. Conceptual-level closure methods and associated costs were developed based on the current facilities layout (Figure 20.5). The closure costs assume that no impacts have occurred to the environment that will require long-term mitigation measures.

Figure 20.5: General Arrangement and Closure Strategy



To accomplish these objectives, the following key elements will be included in the reclamation plan:

1. Decommissioning equipment, management of contaminated water, demolition of physical structure and management of infrastructure;
2. Chemical stabilization, accomplished through rinsing of the HLF, and covering of PAG rock in the WRSF with a low permeability layer engineered closure cover;
3. Physical stabilization, accomplished through slope grooming, and the application of topsoil and revegetation;
4. Control of surface waters; and
5. Monitoring effluent chemistry from the pad and water draining the mine waste areas.

Closure will be accomplished in three stages:

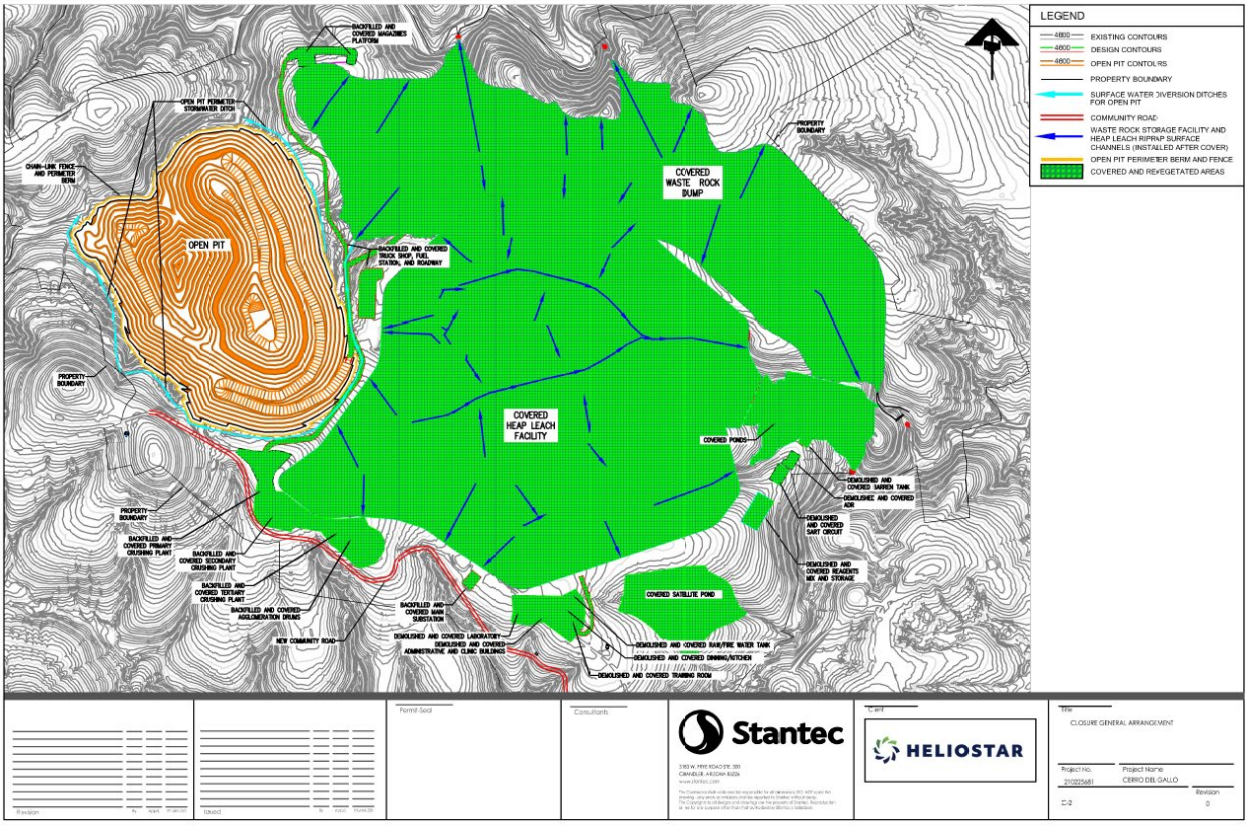
1. Concurrent: measures implemented during the operating life of the Project;
2. Final: measures implemented after cessation of operations; and,

3. Post-closure: provides for short-term maintenance and long-term monitoring of the closed facilities.

An outline of the key components of the closure and reclamation plan is given in this section. Further detailing of these components will be required before construction commences. During operation, the closure and reclamation plan will be revised further. Progressive reclamation will be undertaken during mining activities where possible, but the majority of work will occur after the completion of mining and final gold recovery. The reclamation land use objective will be to return the land to its traditional use, wherever practical, and establish a self-sustaining ecosystem that is compatible with a healthy environment.

The final site configuration after closure is presented on Figure 20.6, which shows the remaining facilities and areas of soil cover and revegetation.

Figure 20.6: Mine Closure Layout



20.12.1 HLF and WRSF Closure

Designs for operation and closure of the HLF and WRSF were provided by Golder (2020). The HLF and WRSF will be stacked in approximately eight-meter and 30-meter high lifts, respectively, with benches provided between lifts to create an average overall ore slope of 3H:1V, which will provide operational and post-closure stability of the heap and minimize grading during reclamation. The maximum HLF design height is 80 m. The stability analysis for the static and seismic conditions (design event of 475-year return period) indicated that the proposed designs for the HLF and WRSF are geotechnically stable under long-term, post-closure conditions (Golder, 2020).

The HLF will have underdrains installed where the pH of water transported beneath the heap can be monitored and, if the pH begins to rise, it will be used for process make-up water. The WRSF will also have underdrains installed to minimize the contact time between the waste and water. Collection ponds will be installed at the base of the waste rock dump major drainages to monitor any seepage and, if necessary, pump the seepage to portions of the HLF that are not under leach.

Reclamation grading will involve re-sloping between and on the benches to facilitate surface water drainage off of the reclaimed facilities.

The proposed closure covers for the HLF and WRSF will consist of topsoil, or a composite topsoil/soil, capable of supporting local vegetation and a low permeability clayey layer below to reduce the infiltration through the facilities. The cover will be placed over each entire facility, including slopes, benches and top platform (crest). This cover system will reduce seepage into the facilities to a negligible amount and will minimize potential soil erosion and sediment runoff to natural drainages. The cover system will be comprised of the following materials from top to bottom:

- Topsoil layer 0.2 m thick with a maximum permeability of $1x10^{-4}$ cm/sec
- Cover layer 1.0 m thick with a maximum permeability of $1x10^{-5}$ cm/sec
- Lower permeability clay layer 0.6m thick with a maximum permeability of $1x10^{-6}$ cm/sec

The solution management for closure of the HLF will occur in three phases:

- Residual leaching with cyanide; no additional ore placement
- Rinsing with fresh water
- Disposal of the process solution/rinse water

At the end of stacking and leaching, when it is determined that the HLF is no longer economical to operate, the HLF will be rinsed to remove residual metals and sodium cyanide. Rinsing is assumed to be three pore volumes of the spent ore. For the purposes

of closure costing, it is assumed that the recycled rinse water will be treated in a future water treatment plant so that rinsing is more efficient. At the completion of rinsing, the rinsed ore is anticipated to be a nonhazardous material. It is anticipated that during active leaching after the ore stacking has ceased, and during rinsing, water accumulating in the pond system will need to be treated to discharge standards. The treated water will be discharged to Arroyo las Cuevas.

At the end of the heap rinsing process and installation of the covers, the surface water runoff from the HLF and WRSF closure covers will be considered non-contact water. The surface water management system developed on the reclaimed benches will be connected to the surface water management systems constructed for operations of the HLF and WRSF. It is anticipated that most of the site-wide water management structures will remain in place for closure. It is proposed that the emergency pond and other water management ponds be incorporated into the post-closure water management system to collect residual flows from the HLF. As mentioned above, the residual discharge during rinsing will be treated; this includes the pond water that is incorporated into water management system, treated and reused to rinse the HLF. Seepage collected from the WRSF is also anticipated to require treatment. The timing of when seepage will meet “de minimis” standards has not been determined; however, the closure scenario assumes that after the HLF draindown period, and covers are placed on the HLF and WRSF, the seepage rates will diminish to flows that can either be managed in an evaporation pond or via a passive treatment system.

Monitoring will be performed during the post-closure period to verify that the water discharges will have acceptable quality to allow direct discharge into the natural drainages. Per NOM-155-SEMARNAT-2007, the numeric discharge standard for the final rinse water for WAD cyanide is less than 0.2 mg/L and the pH should be between five and 10. The discharges to the environment must comply with NOM-001-SEMARNAT-1996 permissible limits for discharge. Should seepage or runoff not meet standards, the water will need to continue to be treated in the water treatment plant. This water treatment plant will be constructed as part of the closure activities.

20.12.2 Open Pit

The open pit will be left in place with no backfill. It is anticipated that the pit walls will become unstable over time. The closure method will be to prevent access to the area as opposed to attempt to stabilize the pit walls. Berms will be constructed around the perimeter of the pit for safety and for surface water diversions. A v-ditch lined with rip-rap will be placed where needed for surface water run-on control. A chain-link fence with warning signs will be installed around the facility. The need for fencing will be re-evaluated at the end of the post-closure monitoring period.

The pit is assumed to stay dry except during short periods during the rainy season. The exploration drilling has indicated that the water table is below the bottom of the pit and water collected in the pit from precipitation is expected to evaporate or infiltrate.

20.12.3 Ponds and Water Management

Pond sediments will be placed on the HLF or an offsite waste facility, depending on characterization results and management requirements. Lined ponds will be perforated, filled, regraded, covered and revegetated. Should the liners be classified as hazardous, then the liners may be placed on the HLF or disposed of permanently in an appropriate offsite waste facility.

The HLF and WRSF will continue to produce contact water from stormwater runoff and seepage after operations cease. The draindown and seepage water is assumed to be acidic with high concentrations of metals and sulfate and will require water treatment prior to discharge to the environment. At this time the water generated from the HLF and WRSF will be managed through collection systems and conveyed to a water treatment plant to be constructed at closure.

Additional studies are needed to predict a closure water balance and future water quality prior to preparing a more detailed closure strategy. For the purposes of the closure plan, the capital costs for a water treatment plant have been included without operating costs and the timing of draindown rates has not been estimated.

20.12.4 Plant Site

A survey will be carried out at closure to document the quantities and types of materials at site at the time of closure. Chemicals and other unused materials will be transferred to other operations or sold.

Combustibles and other chemical products will be removed prior to closure. Hazardous wastes will be classified and placed in appropriate containers, per the environmental regulations of Mexico. The hazardous and non-hazardous wastes will be sent to appropriate facilities offsite for final disposal after being temporarily stored onsite.

Plant site buildings where cyanide was handled will need to be decommissioned. Decommissioning activities refer to decontamination and industrial cleaning of equipment, machinery and pipelines. Equipment that stored, processed and transported cyanide-containing solutions will be decontaminated by rinsing with recirculated water prior to being dismantled and removed from the site for permanent disposal. Rinse water will be treated at the future water treatment plant.

All plant and related facilities will be dismantled or demolished. Foundations will be removed and excavated areas will be filled to restore drainage. The closure plan assumes that a construction landfill located nearby will be available for disposal of non-hazardous materials. When possible, materials will be salvaged and resold.

20.12.5 Roads, Buildings and Other Support Facilities

Several roads will remain to access the property for closure and environmental monitoring. Internal roads will be leveled and graded to facilitate vegetation growth. Cover materials will be used, as needed.

The electrical infrastructure will be removed, and dewatering wells (if any) will be decommissioned and permanently closed.

20.12.6 Revegetation

It is anticipated that the seed mixes to be used will be composed of native plants. Native seeds will be collected and used in the production of rootstock at a future plant nursery. The closure plan assumes that the plant nursery will be constructed and operated by Company personnel as part of operations.

Restriction of livestock will be critical, and it is anticipated that the need for fencing around reclaimed areas will be evaluated at a later stage.

20.12.7 Post-Closure Monitoring

After the completion of final closure, the site will require regular maintenance for approximately 20 years post-closure or until there are no further signs of changing conditions. Initially, the site will be inspected every three months, and maintenance activities will be planned immediately following each wet season and following any unseasonal major storm events. The purpose of this is to ensure drainage and erosion control measures are working as planned, and to allow the recently revegetated areas to mature and properly take hold. Maintenance work will consist of light manual labor (ditch tending, rubble removal, and so forth), and light equipment (backhoe and bulldozer) work to regrade or groom any areas showing signs of distress or erosion. Success criteria will be developed for erosion, slope stability, water quality and vegetation.

Once the site stops showing signs of seasonal distress, the functionality of the facilities has been field-proofed and when the geochemical performance matches predictive modeling, periodic inspection and maintenance activities can be reduced in frequency; initially to annually and eventually to only after unusually high rainfall periods.

The quality of the water draining from the HLF will require monitoring and comparison to the predicted chemistry and discharge standards. If the measured water quality significantly varies from that predicted, in an unfavorable manner, then the geochemical model will be revised and new forecasts prepared. In the extreme case, additional rinsing and neutralization of the heap may be required. More likely, it will only be required to extend the short-term maintenance period.

During the initial, short-term draindown period, the ponds will remain in service for water management. Water collected in the ponds will be tested with each inspection cycle and if the water quality does not meet discharge standards, then that water will be treated at the water treatment plant and/or recirculated to the HLF. The ponds will likely accumulate sediments and precipitates as water accumulates and evaporates. These sediments will require periodic removal and can be buried within the HLF. The length of the pond life will depend upon the effectiveness of the erosion control measures and re-vegetation efforts.

20.12.8 Closure Cost Estimate

The closure cost estimate was estimated using closure activity costs applied to disturbance areas, and unit rates for demolition and remediation. The closure costs were based on preliminary closure activities and unit rates published by the Mexican Chamber of Construction Industry (Cámara Mexicana de la Industria de la Construcción) in 2025 for haulage, demolition and removal for construction projects. The closure cost was prepared in Mexican pesos and converted to US dollars.

The US Forest Service (USFS) recommendation of 35 percent for a preliminary cost estimate was used in the cost estimation (USFS 2004). Indirect costs were estimated at 47.5 percent of the direct expenses, and follows the guidance of the US Bureau of Land Management (BLM 2014). Indirect costs include field management, central administration, insurance, and contingency. The Mexican authority does not require a closure cost financial mechanism.

The closure cost was estimated to be about US$132M (Table 20.8).

Table 20.8 Closure Cost Estimate 2025

Item No.	Item Description	Costs (MXN)	Costs (US$)	Percentage of Total
1	Open Pit	$10,570,000	$563,000	0.4%
2	Waste Rock Storage Facility	$436,924,000	$23,278,000	17.6%
3	Heap Leach Facility	$836,381,000	$44,559,000	33.8%
4	Construction of Waste Water Treatment Plant	$18,770,000	$1,000,000	0.8%

5	Infrastructure Demobilization/Demolition	$130,196,000	$6,936,000	5.3%
6	Engineering and Design	$974,000	$52,000	0.04%
7	Closure Management (Direct Administration Costs)	$252,769,000	$13,467,000	10.2%
	Subtotal:	**$1,686,584,000**	**$89,855,000**	**68.1%**
Indirect Costs (Applied to Items 1 through 6)				
	Field Management (5% of Direct Costs)	$71,642,000	$3,817,000	2.9%
	Central Administration (6% of Direct Costs)	$85,970,000	$4,580,000	3.5%
	Insurance (1.5% of Direct Costs)	$21,493,000	$1,145,000	0.9%
	Contingency (35% of Direct Costs)	$501,494,000	$26,718,000	20.2%
8	Post Closure Care and Maintenance	$109,578,000	$5,838,000	4.4%

Total Closure Costs	**$2,476,761,000**	**$131,953,000**	**100%**

The QP notes that the closure water balance has not been developed to estimate the time or volume of water that requires treatment, nor have the water treatment plant specifications been developed.

20.13 Security

A study to evaluate security risks for the Project was carried out in 2025 (Heliostar). The study indicated that the State of Guanajuato has had increases in security problems, including attempted robbery at Guanajuato Silver's mining operations, caused by organized crime. The most affected towns include Celaya, Irapuato, Salamanca, Leon and Villagrán. None of those towns are within the direct or indirect influence radius of the Project. The study results indicated that the Project risks were acceptable based on the implementation of a mitigation plan.

21 Capital and Operating Costs

21.1 Summary

Capital and operating costs for the process and general and administrative components of the CdG Project were estimated by KCA, with input from Heliostar, while mining costs were provided by Hard Rock Consulting and are based on contract mining. The cost estimates are considered to have an accuracy of ±20% and are discussed in further detail in this section. All costs are stated in Q3 2025 US dollars.

The total Life-of-Mine (LOM) capital cost for the Project is estimated at US$355.6 million, excluding, value-added tax (IVA), and other taxes. IVA is applied to all capital costs at a rate of 16% and is assumed to be fully refundable. Capital requirements for the CdG Project are summarized in Table 21.1.

Table 21.1 LOM Capital Cost Summary

Description	Cost (US$M)
Pre-Production Capital	$ 178.9
Initial Fills	$ 0.9
Sustaining Capital – Mine & Process	$ 43.9
Closure	$ 131.9
Total excluding IVA	**$ 355.6**

Source: KCA (2025)

The average life of mine operating cost for the Project is US$13.82 per tonne of ore processed. Table 21.2 presents the LOM operating cost requirements for the CdG Project.

Table 21.2 LOM Operating Cost Summary

Description	LOM Cost (US$/t Processed)
Mine	$ 3.79
Process & Support Services	$ 9.12
Site G & A	$ 0.91
Total	**$ 13.82**

Source: KCA (2025)

IVA is not included in the operating costs.

21.2 Capital Costs

The capital cost estimates are based on the design presented in this report and include process facilities, infrastructure, construction indirects, mine contractor mobilization, and owner mining capital costs, assuming contract mining over the project life.

Costs were primarily estimated by KCA, with input from Heliostar for major construction contracts, unit rates, and infrastructure, and by Hard Rock Consulting for owner and contractor mining costs. Earthworks, concrete, and major piping quantities were estimated by KCA based on material take offs from the previous PEA. Equipment and material requirements are derived from the design information presented in this report. Capital costs were developed mainly from budgetary supplier quotes for major and most minor equipment, as well as contractor quotes for major construction contracts. Where project specific quotes were unavailable, estimates were based on recent KCA data or costs from Heliostar's other operations.

All equipment costs assume new equipment purchased or fabricated from manufacturers.

The total pre-production capital cost for the CdG Project is estimated at US$195.4 million, inclusive of process facilities, infrastructure, construction indirects, mine contractor mobilization, mine pre-production, initial fills, and working capital. Costs are presented in Q3 2025 US dollars, using an exchange rate of 18.7 MXN:US$1 where applicable.

Pre-production capital costs required for the CdG Project by area are presented in Table 21.3.

Table 21.3 Summary of Pre-Production Capital Costs

Plant Totals Costs	Total Supply Cost US$	Install US$	Grand Total US$
Total Direct Costs	**$97,620,900**	**$28,176,965**	**$125,797,865**
Sub Total with Spare Parts			**$131,466,306**
Contingency	$22,057,581		$22,282,581
Total Direct Costs with Contingency			**$153,748,887**
Mining Costs			**$1,400,000**
Indirect Costs			**$6,465,600**
Other Owner's Costs			**$3,549,690**
EPCM			**$13,817,150**
Sub Total Costs Pre-Production			**$178,981,327**
Initial Fills			**$869,747**
Working Capital (60 days)			**$15,630,000**
TOTAL COSTS (excluding IVA)			**$195,481,074**

Source: KCA (2025)

21.2.1 Mine Capital Costs

The estimate of mining capital assumes that contract mining will be used through the LOM. The contractor will provide their own shop and maintenance facilities for the mine.

21.2.1.1 Initial Mining Capital

Initial mining capital costs were estimated to include contractor mobilization, blasting consumables storage, engineering and office equipment, slope monitoring and surveying equipment, haul road development, and light vehicles.

Table 21.4 shows the estimated owners mining capital requirements during preproduction.

Table 21.4 Initial Mine Capital Costs (000's US$)

Initial Mine Capital	Units	Total
Contractor Mobilization	KUS$	350
Explosive Storage Bins/Magazines	KUS$	100
Slope Radar System	KUS$	500
Engineering & Office Equipment	KUS$	50
Survey Equipment	KUS$	150
Access Roads - Haul Roads	KUS$	100
Light Vehicles	KUS$	150
Contingency	KUS$	225
Total Initial Mine Capital	**KUS$**	**1,625**

Source: HRC (2025)

Explosive and blasting agent storage capital costs have been estimated based on estimation guides. The engineering and office equipment estimate allows for computers and planning software. The slope radar and survey equipment will be used for pit slope monitoring and daily surveying requirements. Costs for haul roads and site preparation have been estimated based on using dozers and graders to be used in developing mine access to the top of the hill and to level sites for mine offices and shop areas. No drilling and blasting is expected to be required for this work. Light vehicle capital expenditure for engineering and geological personnel are also included.

21.2.2 Process & Infrastructure

21.2.2.1 Process & Infrastructure Cost Basis

Process and infrastructure capital costs were estimated by KCA, with input from Heliostar for major construction contracts and infrastructure, and based on material take offs from previous project reports for heap leach and waste rock dump earthworks. Equipment and material requirements are derived from the design information presented in previous sections of this Report. Capital costs are primarily based on Project-specific budgetary quotes for major and most minor equipment, as well as contractor quotations for major construction contracts provided by Heliostar. Where Project specific quotes were unavailable, estimates were derived from recent KCA data or cost information from Heliostar's other operations. All capital cost estimates assume the purchase or fabrication of new equipment.

The process cost estimate has been developed by separating each area into the following disciplines, as applicable: Major earthworks and liner; civil (concrete); structural steel; platework; mechanical equipment; piping; electrical; instrumentation; infrastructure and buildings; supplier engineering; and commissioning and supervision.

Pre-production process and infrastructure costs by discipline are presented in Table 21.5.

Table 21.5 Summary of Process & Infrastructure Pre-Production Capital Costs by Discipline

Discipline Totals	Cost @ Source	Freight	Customs Fees & Duties	Total Supply Cost	Install	Grand Total
	US$	US$	US$	US$	US$	US$
Major Earthworks				$3,451,726	$14,921,305	$18,373,031
Civils (Supply & Install)	$5,456,171			$5,456,171	$923,259	$6,379,430
Structural Steelwork (Supply & Install)	$1,934,389			$1,934,389	$332,898	$2,267,287
Platework (Supply & Install)	$2,609,351			$2,609,351	$308,722	$2,918,072
Mechanical Equipment	$50,701,611	$4,231,738	$2,163,122	$57,096,471	$6,730,895	$63,827,366
Piping	$5,827,034	$0	$0	$5,827,034	$711,200	$6,538,234
Electrical	$7,191,114	$125,241	$62,620	$7,378,975	$1,035,272	$8,414,246
Instrumentation	$1,500,296	$165,033	$82,516	$1,747,844	$174,440	$1,922,284
Infrastructure & Buildings	$11,829,666	$245,447	$43,826	$12,118,939	$491,785	$12,610,724
Supplier Engineering					$2,422,000	$2,422,000
Commissioning & Supervision					$125,190	$125,190
Spare Parts				$5,668,441		$5,668,441
Contingency				$22,057,581		$22,057,581
Total Direct Costs	**$87,049,631**	**$4,767,458**	**$2,352,085**	**$125,346,922**	**$28,176,965**	**$153,523,887**

Source: KCA (2025)

Freight, customs, duties, and installation costs have been included for all disciplines. Freight costs were estimated at 11% of equipment costs based on bulk freight assumptions, unless supplier-quoted freight costs were available and applied. Installation costs are based on contractor unit rates or are included in turnkey supplier packages. Contractor costs include labor, tools, and minor support equipment required for equipment installation. Engineering, procurement, and construction management (EPCM), indirect costs, and initial fills inventory are also included in the capital cost estimate.

21.2.2.2 Major Earthworks & Liner

Major earthworks and liner costs include earthworks required to provide level areas for site facilities, access roads, and materials for the heap leach pad and process solution ponds. Earthworks quantities for process areas and building platforms were estimated by KCA, while quantities for the heap leach and waste rock dump facilities were derived from previous reports based on the preliminary site design. Unit costs for site earthworks and lining systems, including installation, are based on contractor quotations and are summarized in Table 21.6.

Table 21.6 Cerro del Gallo Earthworks/Liners/Materials Unit Costs

Description	Unit	Cost, US$/Unit
Clear & Grub	Hectares	$10,484.82
Topsoil / Growth Media Strip & Stockpile	m^3	Incl. in Clear & Grub
Site Preparation, Clearing	m^2	$0.64
Excavation - Type B	m^3	$4.19
Structural Fill of Cut Product	m^3	$2.15
Excavation - Type C	m^3	$7.66
Structural Fill 30 cm lifts 90-92% spread and comp imported.	m^3	$2.80
Anchor Trenches	m	$11.39
Storm Water Diversions	m	$2.94
Rip-Rap D50 = 200 mm	m^3	$12.51
Rip-Rap D50 = 300 mm	m^3	$12.51
Rip-Rap D50 = 400 mm	m^3	$12.51
Filter Fill	m^3	$12.80
Berm Structural Fill	m^3	$7.95
-Underliner (Clay)	m^3	$8.75
1.5 mm HDPE Single Side Textured (supply & install)	m^2	$3.94
1.5 mm HDPE Smooth (supply & install)	m^2	$3.62
HDPE Geonet (supply & install)	m^2	$5.81
Geotextile (supply & install)	m^2	$1.45
Pad Gravel Cover	m^3	$12.40

Source: KCA (2025)

21.2.2.3 Civils

Civil works include detailed earthworks and concrete. Concrete quantities were estimated based on layouts, comparable equipment installations, major equipment weights, and slab areas. Unit costs for concrete supply and installation, including production, delivery, and placement activities, were developed by KCA based on contractor quotations.

21.2.2.4 Structural Steel

Costs for structural steel, including grating, light, medium and heavy structural steel, and handrails for each area are based on material takeoffs estimated by KCA based on preliminary layouts, equipment weights and unit costs from a contractor quote received by Heliostar.

21.2.2.5 Platework

Platework includes costs for the supply and installation of steel tanks, bins and chutes. Platework costs are primarily based on supplier quotes for shop fabricated tanks and contractor quotes for larger field erected tanks, or have been included as part of vendor supply packages.

21.2.2.6 Mechanical Equipment

Mechanical equipment costs are based on a detailed equipment list developed for all major process equipment. Costs for major and most minor equipment items are derived from budgetary supplier quotations. Where Project-specific quotations were unavailable, allowances were applied based on recent KCA data and cost information from Heliostar's other operations. All costs assume the purchase or fabrication of new equipment.

Mechanical equipment costs include predominantly turnkey supply packages for the ADR, refinery, and cyanide systems; complete equipment supply packages for the crushing and reclaim systems, excluding the HPGR, which was quoted separately; and additional equipment packages from multiple suppliers. Installation costs are based on contractor unit rates provided by Heliostar and estimated installation hours or are included within turnkey supplier packages.

21.2.2.7 Piping

Major piping, including heap leach irrigation and gravity solution collection piping, was estimated based on material take-offs and supplier quotations. All other piping, fittings, and valve costs were estimated as a percentage of mechanical equipment costs or included within vendor supply packages. Piping supply factors ranging from 0% to 30% of mechanical equipment costs were applied by area, depending on system complexity. Piping included as part of equipment packages is included in the mechanical equipment cost.

21.2.2.8 Electrical

Major electrical equipment including transformers, substations, site powerlines, and motor control centers have been costed based on supplier quotes. Costs for miscellaneous electrical works, including supply and installation of electrical cable, cable trays, grounding and lightning protection are based on contractor quotes provided by Heliostar.

21.2.2.9 Instrumentation

Instrumentation costs have been estimated based on a percentage of the mechanical equipment costs. A rate ranging from 0% to 12% of the equipment cost was used to estimate instrumentation purchase costs for each area based upon recent KCA experience on similar projects. Instrumentation installation hours are estimated based on a factor of instrumentation equipment costs.

21.2.2.10 Infrastructure & Buildings

A list of buildings is presented in Table 21.7. Building costs are based on a combination of pre-engineered steel buildings, modular vendor-supplied trailers, and shipping containers. Allowances for office and dining furnishings, appliances, lockers, and shop tools were derived from costs at Heliostar's existing operations. Septic systems and leach

field costs were provided by Heliostar and are included in the cost estimate. The mining contractor will provide maintenance facilities for mining equipment.

Table 21.7 Cerro del Gallo Buildings

Description
Guard House
Explosives Storage Magazine
Modular Offices - Process
Modular Offices - Mine
Warehouse & Maintenance - Process Area
General Warehouse
Warehouse - Reagent Storage
Modular Dining Area
Modular Change Facility & Locker Storage
Clinic
Warehouse Storage Containers - Process
Warehouse Storage Containers - Mine/Crusher
Refinery
Laboratory

Source: KCA (2025)

Power during the first year of operation will be supplied by leased diesel generators at an estimated cost of US$0.38/kWh. Beginning in Year 2, grid power will be supplied via a 115-kV transmission line at an estimated cost of US$0.11/kWh. Preliminary powerline capital costs were estimated by CFE, with 50% assumed to be incurred during pre-production and the balance during the first year of operations.

Water supply will be provided by three groundwater wells located near the site. Costs for well drilling and construction, well pumps, and the raw water pipeline to the site tanks are based on contractor quotations.

The site perimeter will be enclosed with animal fencing, and the process ponds and process facilities will be enclosed with 2-m-high chain-link fencing. Fencing costs were provided by Heliostar based on costs from existing operations

21.2.2.11 Supplier Engineering & Supervision

Supplier engineering and construction and commissioning supervision were quoted for the ADR and laboratory packages. Where not directly quoted, supplier engineering for equipment items are assumed to be included in the equipment supply cost.

21.2.2.12 Process Mobile Equipment

Mobile equipment included in the capital cost estimate are detailed in Table 21.8.

Table 21.8 Process Mobile Equipment

Item	Description	Qty.
Light Vehicles	1 ton pickup truck	15
Leach Pad Truck	1 ton pickup truck	1
Service Truck	1 ton	1
Forklift	2.5 ton	2
Crane (60t)	60 ton	1
Light Plants	400 Watt, diesel fired	4
Dozer	CAT D6 or Equiv., LGP tracks	1
Boom Truck	17-20 tons	1
Fuel Truck		
Small Water Truck	3,000 gallons	1
Excavator	CAT 321D or Equiv.	1
Backhoe		1
Bobcat		2
Ambulance		1

Source: KCA (2025)

Costs for process mobile equipment are based on cost guides or other published data. Mobile equipment costs are considered in the mechanical equipment cost estimate.

21.2.3 Spare Parts

Spare parts are budgeted at 4% of the mechanical equipment costs, based on KCA experience.

21.2.4 Indirect Costs

Indirect costs include costs for items such as temporary construction facilities and support, surveying, temporary communication systems, temporary warehousing, temporary power and water, quality control, fenced yards, construction office, support equipment, security, vendor representatives, etc., and have been estimated based on a construction period of 16 months and Heliostar's experience on their past projects.

21.2.5 Other Owner's Construction Costs

Other Owner's construction costs are intended to cover the following items:

- Owner's costs for labor, offices, home office support, vehicles, travel and consultants during construction;
- Subscriptions, licence fees, etc.;
- Taxes and Permits;
- Work place health and safety costs during construction.

Other Owner's construction costs are estimated based on Heliostar's experience on past projects.

21.2.6 Initial Fills Inventory

The initial fills inventory consists of a supply of consumable items stored on site at the outset of operations. The list of consumables includes cyanide, cement, carbon, caustic, hydrochloric acid, flocculant, sulfuric acid, NaSH, smelting fluxes and antiscalant.

21.2.7 Engineering and Construction

The estimated costs for engineering, procurement and construction management (EPCM) for the development, construction, and commissioning are based on a percentage of the direct capital cost and assume Heliostar managing EPCM services internally. The total EPCM cost is estimated at 9.0% of the process and infrastructure direct costs without any added contingency.

The EPCM costs cover services and expenses for the following areas:

- Project Management;
- Detailed Engineering;
- Engineering Support;
- Procurement;
- Construction Management;
- Commissioning;
- Vendors Reps.

For some major equipment packages, costs associated with detailed engineering, commissioning, and installation supervision have been included in the vendor's quotes; these costs are reflected in the supplier engineering estimate of the capital costs.

21.2.8 Contingency

Contingency for the process and infrastructure has been applied to the total direct costs by discipline. Contingencies varying from 10% to 25% were used depending on the confidence level and are detailed in

Table 21.9.

Table 21.9 Process and Infrastructure Contingency

Direct Costs Contingency	%
Major Earthworks	25%
Civils (Supply & Install)	25%
Structural Steelwork	20%
Platework	15%
Mechanical Equipment	15%
Piping	15%
Electrical	15%
Instrumentation	20%
Infrastructure & Buildings	20%
Supplier Engineering	10%
Commissioning & Supervision	10%
Total Contingency on Direct Costs	**17.5%**

Source: KCA (2025)

Contingency for Indirect and Other Owner's costs have been applied at 20%.

21.2.9 Sustaining Capital Costs

Sustaining capital costs for the Project include costs to construct the second, third and fourth phases of the leach pad (in Years 1, 2 and 5, respectively) and to replace process area surface mobile equipment in years 5 and 11. Sustaining capital costs are approximately US$43.9 million and includes related contingencies. EPCM and construction indirect costs for the leach pad expansions are included. Sustaining capital costs are presented in Table 21.10.

Table 21.10 Sustaining Capital Costs by Year

SUSTAINING CAPITAL BY YEAR	YEAR 1			YEAR 2			YEAR 5			YEAR 11		
Description	**Supply (US$)**	**Install (US$)**	**Total (US$)**	**Supply (US$)**	**Install (US$)**	**Total (US$)**	**Supply (US$)**	**Install (US$)**	**Total (US$)**	**Supply (US$)**	**Install (US$)**	**Total (US$)**
Major Earthworks (including Mob.& Demob.)	$2,086,118	$7,692,275	$9,778,392	$1,368,170	$3,417,039	$4,785,209	$1,732,344	$4,906,208	$6,638,552			$0
Mechanical Equipment							$1,020,394		$1,020,394	$3,878,894		$3,878,894
Infrastructure	$9,012,736		$9,012,736									
Overall Plant & Infrastructure Totals			**$18,791,128**	**$1,368,170**	**$3,417,039**	**$4,785,209**	**$2,752,738**	**$4,906,208**	**$7,658,946**	**$3,878,894**	**$0**	**$3,878,894**
Direct Plant & Infrastructure Costs Contingency			$3,758,226			$957,042			$1,531,789			
Indirect Costs			$751,645			$191,408			$306,358			
EPCM			$751,645			$191,408			$306,358			
TOTAL SUSTAINING COSTS			**$24,052,644**	$1,368,170	$3,417,039	$6,125,068	$2,752,738	$4,906,208	$9,803,450	$3,878,894	$0	$3,878,894

21.2.10 Working Capital

Working capital is money that is used to cover operating costs from start-up until a positive cash flow is achieved. Once a positive cash flow is attained, Project expenses will be paid from earnings. Working capital for the Project is estimated to be US$15.6 million based on 60 days of operation and includes all mine, process and G&A operating costs.

21.2.11 Exclusions

The following have been excluded:

- Finance charges and interest during construction;
- IVA (value added tax). IVA is applied to all capital costs at a rate of 16% in the economic model and is considered to be fully refundable in the following year;
- Escalation costs;
- Currency exchange fluctuations; and
- Penalties or incentives.

21.3 Operating Costs

21.3.1 Mine Operating Costs

Mine operating costs have been estimated based on a contractor quote to perform and manage all drilling, blasting, loading, hauling and support equipment and re-handle of material at the crusher. To assist the contractors in providing proposals, HRC provided a production summary sheet with tonnages, volumes, average haulage distances, and estimated cycle times along with yearly layouts of the mine plan.

Mine owner costs include personnel and supply costs for the owner's employees including mine supervision, engineering, and geology/ore control. Support services are estimated from historic actuals and from base principles for equipment, consumables, supplies, services and manpower requirements based on the mine schedule.

Equipment fuel requirements are calculated based on required operating hours for each unit and haulage route profiles for the trucks. Diesel costs were estimated at US$1.10/L. The costs details by department over the life of the mine are shown in

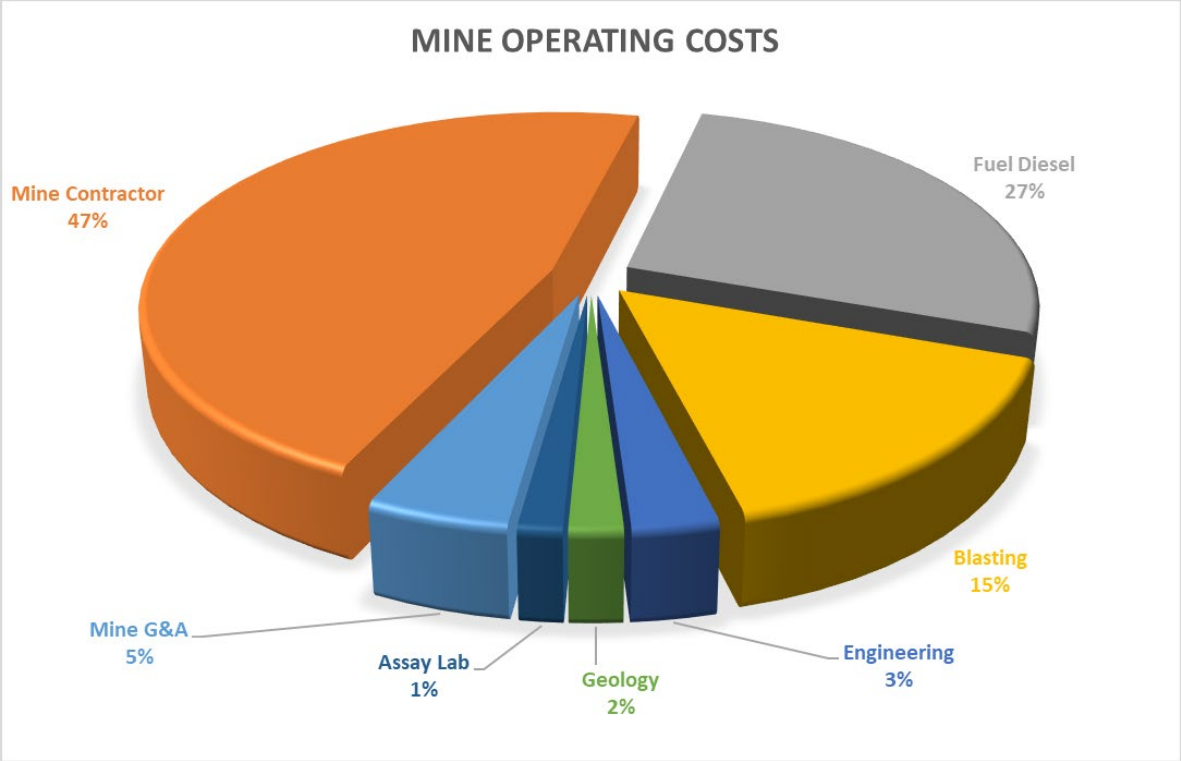
Table 21.11 and Figure 21.1 shows the distribution of these costs by department. The overall mining costs are estimated to be $2.88 per tonne mined

Table 21.11: Mine Operating Costs

Department	Operating Cost (US$/t mined)	Operating Cost (US$/t ore)
Mine general and administrative	$0.14	$0.18
Mine contractor	$1.34	$1.77
Fuel diesel	$0.77	$1.01
Blasting	$0.45	$0.59
Engineering	$0.08	$0.11
Geology	$0.05	$0.07
Assay laboratory	$0.04	$0.06
Total	***$2.88***	***$3.79***

Note: Figure prepared by Hard Rock Consulting, 2025

Figure 21.1: Mine Operating Cost Distribution



Note: Figure prepared by Hard Rock Consulting, 2025.

21.3.2 Process Operating Costs

Process operating costs for CdG have been estimated based on information presented in earlier sections of this Report. Mining costs were provided by Hard Rock Consulting at US$2.88 per tonne mined (LOM US$3.79 per tonne of ore) and are based on contract mining.

Process operating costs have been estimated by KCA from first principles. Labor costs were estimated using project specific staffing and salary, wage and benefit requirements provided by Heliostar based on their existing mining operations in Mexico. Unit consumptions of materials, supplies, power, water and delivered supply costs were also estimated. LOM average processing costs are estimated at US$9.12 per tonne ore.

General administrative costs (G&A) have been estimated by KCA with input from Heliostar. G&A costs include project specific labor and salary requirements and operating expenses including social contributions and land and water rights. G&A costs are estimated at US$0.91 per tonne ore.

Operating costs were estimated at approximately US$13.82 overall and based on 3rd quarter 2025 US dollars, and presented with no added contingency based upon the design and operating criteria present in this report. IVA is not included in the operating cost estimate.

The operating costs presented are based upon the ownership of all process production equipment and site facilities, including the onsite laboratory. The owner will employ and direct all operating maintenance and support personnel for all site activities except mining.

Operating costs estimates have been based upon information obtained from the following sources:

- Contractor mining costs from MDA;
- Owner's mine general costs from MDA;
- G&A costs estimated by KCA with input from Heliostar;
- Project metallurgical test work and process engineering;
- Supplier quotes for reagents and fuel provided by Heliostar;
- Recent KCA project file data; and
- Experience of KCA staff with other similar operations.

Where specific data do not exist, cost allowances have been based upon consumption and operating requirements from other similar properties for which reliable data exist. Freight costs have been estimated where delivered prices were not available.

21.3.3 Process and G&A Operating Costs

Average annual process and G&A operating costs are presented in Table 21.12.

Table 21.12 LOM Average Process, Support & G&A Operating Costs

OPERATING COST	LOM Average	
	Annual Costs, US$	US$ per Tonne Ore
Labor	$2,709,854	$0.47
Primary Crushing	$1,109,759	$0.19
Secondary Crushing	$2,334,585	$0.41
Tertiary Crushing (HPGR)	$4,267,945	$0.74
Agglomeration	$9,979,772	$1.74
Conveyor Stacking	$872,764	$0.15
Heap Leach Systems	$2,230,372	$0.39
SART	$14,860,370	$2.59
Recovery	$901,406	$0.16
Refinery	$399,618	$0.07
Reagents	$9,929,074	$1.73
Water Supply & Distribution	$462,465	$0.08
Laboratory	$836,106	$0.15
Support Services / Facilities	$1,458,689	$0.25
TOTAL COST PROCESS & SUPPORT	**$52,352,778**	**$9.12**
G&A	$5,205,800	$0.91
TOTAL COST	**$57,558,578**	**$10.02**

Source: KCA (2025)

21.3.3.1 Consumable Items

Operating supplies have been estimated based upon unit costs and consumption rates predicted by metallurgical tests and have been broken down by area. Freight costs are included in all operating supply and reagent estimates. Reagent consumptions have been derived from test work and from design criteria considerations. Other consumable items have been estimated by KCA based on KCA's experience with other similar operations.

Operating costs for consumable items have been distributed based on tonnage and metal production or smelting batches, as appropriate.

21.3.3.2 Heap Leach Consumables

Pipes, Fittings and Emitters – The heap pipe costs include expenses for broken pipe, fittings and valves, and abandoned tubing. The heap pipe costs are estimated to be US$0.05/t ore, and are based on previous detailed studies conducted by KCA on similar projects.

Sodium Cyanide (NaCN) – Delivered sodium cyanide is quoted at US$2.17/kg. Cyanide is primarily consumed in the heap leach with a portion of the consumed cyanide being regenerated as part of the SART process. Overall cyanide consumption is estimated at 0.71 kg/t ore.

Cement – Cement is consumed at an average rate of 10 kg/t ore with a range of 5 – 20 kg/t depending on ore type and is used for pH control and agglomeration at the heap. A delivered price of US$0.16/t has been quoted.

Antiscale Agent (Scale Inhibitor) – Antiscalant consumption is based on a dosage range of 0 to 20 ppm to the suctions of the barren and pregnant pumps. A delivered price of US$2.50/kg has been used based on recent supplier quotes in KCA's files.

21.3.3.3 Recovery Plant and SART Consumables

Carbon – Carbon is used in the adsorption circuit to adsorb metal values from pregnant leach solution and is assumed to be consumed at a rate of 4% per strip. Carbon is quoted at US$5.68/kg

Caustic – Caustic consumption will be 336 kg per strip. Caustic will also be consumed in the SART plant and will be used at a rate of 0.03kg/t ore. Caustic has been quoted at US$0.65/kg.

Hydrochloric Acid (HCl) – Hydrochloric acid is used in the acid washing circuit to remove scale from carbon which reduces the carbons activity and ability to adsorb metal. HCl is assumed to be consumed at a rate of 1,000 L/strip and had been quoted at US$0.41/L.

Flocculant – Flocculant is used in the SART circuit at the copper precipitation and gypsum thickeners and is consumed at an average rate of 0.018 kg/t ore. Flocculant has been quoted at US$4.60/kg

Sulfuric Acid (H_2SO_4) – Sulfuric acid is consumed at an average rate of 2.72 kg/t ore for as part of the SART process. Sulfuric acid is quoted at US$0.35/kg

Lime – Will be consumed in the SART plant at a rate of 1.94 kg/t ore and was quoted at $0.195/kg.

NaSH – NaSH is used as part of the SART process and is consumed at an average rate of 0.62 kg/t ore. NaSH has been quoted at US$1.80/kg

21.3.3.4 Mobile Support Equipment

Mobile and support equipment are required for the process and include two fork lifts, one 17 tonne boom truck, one backhoe, sixteen pickup trucks, one maintenance truck, one 60 tonne rough terrain crane, one excavator, two bobcats and an ambulance. The costs to operate and maintain each piece of equipment have been estimated primarily using

published information and project specific fuel costs. Where published information was not available, allowances were made based on KCA's experience from similar operations.

21.3.3.5 Wear, Overhaul, Maintenance and Maintenance Supplies

Overhaul and maintenance of equipment along with miscellaneous operating supplies for each area have been estimated as allowances based on tonnes of ore processed. The allowances for each area were developed based on published data as well as KCA's experience with similar operations.

21.3.3.6 Power

Power usage for the process and process-related infrastructure was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with estimated on-stream times to determine the average energy usage and cost. The total attached power for the process and infrastructure is estimated at 12.3 MW with an average demand of 6.9 MW. Power for the Project will be provided by a 115 kV powerline at an estimated average cost of US$0.11 per kWh.

21.3.3.7 Personnel and Staffing

Staffing requirements for process and administration personnel have been estimated by KCA based on experience with similar sized operations with input from Heliostar on wages and salary information based on their existing mining operations. Staffing will be primarily by Mexican nationals with an emphasis of hiring workers from the local community. Total process personnel are estimated at 142 persons including 24 laboratory workers. G&A labor is estimated at 25 persons. Mining labor will be provided by the mining contractor and is considered in the mining cost estimate.

Personnel requirements and costs are summarized in

Table 21.13.

For continuous operations, there will be three crews working 12-hour shifts. Supervision and technical staff will operate on a flexible schedule to suit operational requirements.

Table 21.13 Cerro del Gallo Staffing Levels and Salary Schedules

Description	Number of Workers	Cost US$/yr
Process Supervision	10	$552,844
Crushing & Reclaim	21	$335,323
Heap Leach	29	$467,275
Recovery Plant / SART	25	$386,028
Maintenance	33	$588,191
Subtotal Process	**118**	**$2,329,661**
Laboratory	24	$491,292
Subtotal Laboratory	**24**	**$491,292**
G&A	25	$1,523,453
Subtotal G&A	**25**	**$1,523,453**
TOTAL	**167**	**$4,344,406**

Source: KCA (2025)

21.3.4 General and Administrative

General and administrative costs (G&A) have been estimated by Heliostar based on their existing operations in Mexico. G&A operating costs include the following items:

- Land Payments;
- Communications;
- IT outside services;
- Computers, printers, software;
- Personnel transport - daily to/from site;
- Travel Costs;
- Personnel housing & meals;
- Access roads maintenance;
- Community relations;
- Outside environmental support & lab services;
- Environmental supplies;
- Legal;
- Outside accounting support – auditors;
- Office supplies;
- Outside consultants;
- Portable toilet service;
- Safety & clinic supplies;
- Insurance;
- Miscellaneous.

21.3.5 Operating Cost Summary

The operating costs include the cost of operations related to mining, processing, and general administration activities. Operating cost estimates were derived from actual

historical costs, mining contract quotes, Heliostar's 2025 operating budget, and Hard Rock Consulting's and KCA's in-house database of projects and studies including experience from similar operations.

Operating cost estimates use terms that are non-International Financial Reporting Standards measures:

- All-in sustaining costs (AISC): as set out in the World Gold Council in its 2018 guidance note. AISC are the sum of operating costs (as defined and calculated above), royalty expenses, sustaining capital, corporate expenses and reclamation cost accretion related to current operations. Corporate expenses include general and administrative expenses, net of transaction related costs, severance expenses for management changes and interest income. AISC excludes growth capital expenditures, growth exploration expenditures, reclamation cost accretion not related to current operations, interest expense, debt repayment and taxes;
- Cash operating costs: include mine site operating costs such as mining, processing and administration, but exclude royalty expenses, depreciation and depletion and share based payment expenses and reclamation costs.

The LOM average cash operating cost is projected to be US$1,067/oz AuEq sold. The LOM average base case total operating cost (including royalties and production taxes) is expected to be US$1,252/oz AuEq. The total AISC summary per tonne of mill feed and per ounce of gold equivalent is expected to be US$19.25/t and US$1,390/oz AuEq respectively, as shown in Table 21.14.

Table 21.14: Total Operating Cost Summary

Operating Costs	Operating Cost (US$/oz AuEq)	Operating Cost (US$/t ore)	Operating Cost (US$/t mined)
Total mining	274.02	3.79	2.88
Total processing	658.44	9.12	
Total site general and administrative	65.61	0.91	
Refinery and transport	68.55	0.95	
Cash operating costs	**1,066.62**	**14.77**	
Production taxes	80.29	1.11	
Royalties	105.12	1.46	
Total cash costs	**1,252.03**	**17.33**	
Sustaining capital costs	138.20	1.91	
Total AISC	***1,390.23***	***19.25***	

22 Economic Analysis

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates;
- Assumed commodity prices and exchange rates;
- Proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution;
- Sustaining costs and proposed operating costs;
- Assumptions as to closure costs and closure requirements;
- Assumptions as to environmental, permitting and social risks.
- Additional risks to the forward-looking information include:
- Changes to costs of production from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade or recovery rates;
- Geotechnical or hydrogeological considerations during mining being different from what was assumed;
- Failure of plant, equipment or processes to operate as anticipated;
- Ability to maintain the social license to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates;
- Changes to tax rates.

22.1 Methodology Used

The Project has been evaluated using a constant US dollar, after-tax discounted cashflow methodology based on a 5% discount rate. For personnel costs and material sourced in Mexico, an exchange rate of 18.7 pesos per US dollar was assumed. This valuation method requires projecting material balances estimated from operations and calculating economic analysis. Cashflows are calculated from sales of metal, less cash outflows such as operating costs, capital costs, working capital changes, royalties, any applicable

taxes and reclamation costs. Resulting annual cash flows are used to calculate the net present value (NPV) and internal rate of return (IRR) of the Project. Tax calculations involve complex variables that can only be accurately determined during operations, and as such, the actual post-tax results may differ from those estimated.

22.2 Financial Model Parameters

The economic analysis was performed assuming a base case gold selling price of US$2,300/oz. Gold metal prices are elevated in the economic analysis to reflect current metal price trends. The base case silver selling price is assumed to be US$27/oz and the base case copper price is assumed to be $4.00/lb.

No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- No pre-production period is planned as mineralized material outcrops at surface with very little waste stripping required, a 0.08:1 strip ratio is planned for year 1;
- The production life is forecast at 15.3 years;
- Cost estimates are in constant Q3 2025 US dollars for capital and operating costs, with no inflation or escalation factors considered;
- Results are based on 100% ownership with a 1% government NSR on revenue from gold and silver production;
- Capital costs are funded with 100% equity (no financing assumed);
- All cash flows are discounted to the start of the construction period using a mid-period discounting convention;
- All metal products will be sold in the same year they are produced;
- Project revenue will be derived from the sale of gold and silver doré and a copper and silver precipitate from the SART plant.

22.3 Taxes

22.3.1 Taxes and Duties

The Project has been evaluated on a post-tax basis to provide an approximate value of potential economics. The Project was assumed to be subject to the following tax regimes:

- The Mexican corporate income tax system (Federal Income Tax) consists of 30% income tax. Federal income tax is applied on Project income after deductions of eligible expenses including depreciation of assets, earthworks and indirect

construction costs, exploration costs, special mining tax, extraordinary mining duty and any losses carried forward;

- Mining tax in Mexico (Special Mining Tax) consists of 8.5% on earnings before interest, taxes, depreciation, and amortization. The special mining duty is applied on Project income after deduction of eligible exploration, earthworks and indirect costs expenses. Income subject to the special mining tax does not allow deductions for depreciation or allow losses carried forward.

At the assumed metal prices, total payments are estimated to be US$442.8 M over the LOM.

The Mexican value-added tax (Impuesto al Valor Agregado) is outside the economic valuation of this Project. The value-added tax is a 16% value added tax applied to all goods and services and is considered to be fully refundable. For the economic model, value-added taxes are not considered in the capital or operating cost estimate as it is assumed that value-added taxes paid versus value-added tax credits will be a net zero value during the period in which they occur.

22.3.2 Royalties

Royalties payable for the CdG include a 1.0% royalty due to the Mexican government as an “Extraordinary Mining Duty”. The 1.0% extraordinary mining duty represents US$25.5 M over the LOM and is included in the Project economics. Royalties are also payable to current claim owners which average 3.7% for the current reserve. The mining claims royalty represents US$108.3 M over the LOM and is also included in the Project economics.

22.4 Economic Analysis

The financial analysis for the Project shows an after-tax net present value at a discount rate of 5% of US$423.9 M, an after-tax internal rate of return of 33.1%, and a payback period of 2.28 years.

Table 22.1 summarizes the projected cashflow; net present value at varying rates; internal rate of return; years of positive cash flows to repay the negative cash flow (payback period); multiple of positive cash flows compared to the maximum negative cash flow (payback multiple) for the project on both an after-tax and before-tax basis.

Table 22.1: Summary Economic Results

Project Valuation Overview	Units	After Tax	Before Tax
Total cashflow	US$ M	724.08	1,166.86
NPV @ 5.0% (base case)	***US$ M***	***423.92***	***699.38***
NPV @ 7.5%;	US$ M	326.76	549.45
NPV @ 10.0%;	US$ M	252.23	434.91
Internal rate of return	***%***	***33.1***	***44.9***
Payback period	Years	2.28	1.78
Payback multiple		4.51	6.72
Total initial capital	US$ M	195.48	195.48

The projected total lifespan of the Project is 15.3 years. Approximately 1.496 Moz of gold, 45.067 Moz of silver and 94.7 kt of copper are projected to be mined, with 0.888 Moz gold, 22.208 Moz of silver and 58.5kt of copper recovered and produced for sale. The economic analysis was completed on an annual cashflow basis. The production schedule and cashflow output is shown in Table 22.2.

Table 22.2 Cashflow Statement on Annual Basis

Item	Units	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	LOM
MINE PRODUCTION																				
Total Ore Mined	***kt***			***5,645***	***5,997***	***6,000***	***6,000***	***6,000***	***6,035***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***2,184***	***91,895***
Au grade	g/t			0.70	0.81	0.43	0.39	0.64	0.41	0.35	0.37	0.41	0.44	0.45	0.47	0.50	0.55	0.62	0.68	0.51
Ag Garde	g/t			12.83	13.78	17.37	22.68	21.12	15.77	16.34	15.51	15.61	15.11	14.71	13.56	12.75	12.16	11.27	9.98	15.25
Cu Grade	%			0.06	0.07	0.09	0.12	0.11	0.10	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.12	0.10
Total Waste	kt			474	266	1,187	926	494	5,182	4,480	3,712	3,110	2,563	2,045	1,636	1,241	1,071	830	87	29,305
Total Tonnes Mined	***kt***			***6,119***	***6,263***	***7,187***	***6,926***	***6,494***	***11,217***	***10,480***	***9,712***	***9,110***	***8,579***	***8,045***	***7,636***	***7,241***	***7,088***	***6,830***	***2,271***	***121,200***
Strip ratio	ratio			0.08	0.04	0.20	0.15	0.08	0.86	0.75	0.62	0.52	0.43	0.34	0.27	0.21	0.18	0.14	0.04	0.32
Other Tonnes	kt			189	199	180	180	180	305	180	180	180	180	180	180	180	180	180	217	3,070
Total Tonnes Moved	***kt***			***6,308***	***6,462***	***7,367***	***7,106***	***6,674***	***11,522***	***10,660***	***9,892***	***9,290***	***8,759***	***8,225***	***7,816***	***7,421***	***7,268***	***7,010***	***2,489***	***124,270***
PROCESS PRODUCTION																				
Tonnes Ore to Heap	**kt**			***5,607***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***6,000***	***6,000***	***6,016***	***6,000***	***2,222***	***91,895***
Au grade	g/t			0.71	0.81	0.43	0.39	0.64	0.41	0.35	0.37	0.41	0.44	0.45	0.47	0.50	0.55	0.62	0.68	0.51
Ag Garde	g/t			12.79	13.80	17.37	22.68	21.12	15.78	16.34	15.51	15.61	15.11	14.71	13.56	12.75	12.16	11.27	10.07	15.25
Cu Grade	%			0.06	0.07	0.09	0.12	0.11	0.10	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.12	0.10
INCOME STATEMENT																				
Au Ozs Placed on Pad	koz			127.2	157.2	83.1	74.8	123.2	79.4	68.3	70.6	78.6	84.2	87.4	89.7	95.9	106.4	120.5	48.9	1,495.5
Au Ozs Recovered	koz			68.9	94.8	64.4	44.2	64.2	57.9	42.3	41.0	44.7	52.5	49.4	50.5	53.7	59.2	66.6	33.7	887.9
Au Cumm. Recovery %	koz			54.2%	57.6%	62.1%	61.6%	59.5%	61.2%	61.2%	60.9%	60.6%	60.7%	60.4%	60.0%	59.7%	59.4%	59.0%	59.4%	59.4%
Ag Ozs Placed on Pad	koz			2,305.7	2,668.6	3,350.9	4,375.8	4,074.5	3,053.2	3,152.8	2,992.4	3,011.8	2,923.5	2,837.2	2,615.8	2,460.2	2,351.5	2,174.1	719.1	45,067.1
Ag Ozs Recovered	koz			1,069.0	1,534.2	1,649.3	2,106.0	2,065.6	1,654.9	1,704.6	1,546.6	1,455.4	1,494.7	1,287.9	1,171.8	1,095.0	1,038.9	947.3	386.7	22,207.9
Ag Cumm. Recovery %	%			46.4%	52.3%	51.1%	50.1%	50.2%	50.8%	51.3%	51.3%	51.0%	51.0%	50.6%	50.2%	49.8%	49.5%	49.2%	49.3%	49.3%
Cu t Placed on Pad	kt			3.4	4.0	5.2	7.3	6.5	6.2	6.9	6.8	6.8	6.8	6.6	6.4	6.4	6.3	6.4	2.7	94.7
Cu t Recovered	kt			1.0	2.4	2.6	4.4	4.1	3.7	4.3	4.5	4.4	4.8	4.2	4.0	4.0	4.0	4.0	2.1	58.5
Cu Cumm. Recovery %	%			30.1%	46.6%	47.9%	52.2%	55.0%	55.7%	56.8%	58.1%	58.9%	60.2%	60.6%	60.9%	61.1%	61.2%	61.4%	61.8%	61.8%
Au Revenue	US$ x 1,000			156,708	215,753	146,575	100,225	145,972	131,650	95,859	92,873	101,444	119,176	112,129	114,638	121,888	134,585	151,433	76,624	2,017,533
Ag Revenue	US$ x 1,000			26,007	37,322	40,122	51,233	50,249	40,258	41,467	37,625	35,405	36,362	31,331	28,505	26,639	25,274	23,045	9,407	540,252
Cu Revenue	US$ x 1,000			7,878	18,778	20,168	33,980	32,179	28,559	33,092	34,833	34,226	37,353	32,796	31,426	31,353	31,389	31,354	16,016	455,380
Gross Revenue	***US$ x 1,000***			***190,593***	***271,852***	***206,865***	***185,438***	***228,400***	***200,467***	***170,418***	***165,331***	***171,075***	***192,891***	***176,256***	***174,570***	***179,880***	***191,249***	***205,832***	***102,047***	***3,013,165***
Dore Refining & Shipping	US$ x 1,000			(815)	(1,125)	(794)	(592)	(812)	(722)	(551)	(529)	(566)	(654)	(610)	(616)	(647)	(706)	(783)	(392)	(10,914)
Smelter Treatment and Shipping Charge	US$ x 1,000			(1,601)	(3,401)	(3,647)	(5,867)	(5,588)	(4,889)	(5,576)	(5,770)	(5,645)	(6,124)	(5,368)	(5,120)	(5,079)	(5,063)	(5,023)	(2,527)	(76,290)
Net Revenue - Before Royalties	**US$ x 1,000**			**188,177**	**267,326**	**202,424**	**178,979**	**222,000**	**194,856**	**164,290**	**159,032**	**164,864**	**186,112**	**170,278**	**168,834**	**174,154**	**185,480**	**200,025**	**99,128**	**2,925,961**
Royalties - Extraordinary Mining Duty	US$ x 1,000			(1,819)	(2,520)	(1,859)	(1,509)	(1,954)	(1,712)	(1,368)	(1,300)	(1,363)	(1,549)	(1,429)	(1,425)	(1,479)	(1,592)	(1,737)	(856)	(25,469)
Royalties	US$ x 1,000			(6,963)	(9,891)	(7,490)	(6,622)	(8,214)	(7,210)	(6,079)	(5,884)	(6,100)	(6,886)	(6,300)	(6,247)	(6,444)	(6,863)	(7,401)	(3,668)	(108,261)
Net Revenue	***US$ x 1,000***			***179,396***	***254,916***	***193,075***	***170,848***	***211,832***	***185,934***	***156,844***	***151,848***	***157,401***	***177,677***	***162,550***	***161,162***	***166,231***	***177,025***	***190,887***	***94,604***	***2,792,231***

Item	Units	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	LOM
OPERATING EXPENSES																				
Mining																				
Mine G&A	US$ x 1,000			1,063	1,067	1,071	1,069	1,066	1,102	1,094	1,088	1,084	1,083	1,077	1,074	1,071	1,073	1,068	394	16,542
Mine Contractor	US$ x 1,000			8,061	8,257	9,531	8,770	8,213	14,468	13,363	12,469	12,182	11,496	11,005	10,436	10,597	10,368	10,067	3,709	162,991
Fuel Diesel	US$ x 1,000			5,225	5,366	5,614	5,387	5,310	6,892	6,625	6,567	6,493	6,406	6,311	6,249	6,189	6,179	6,126	2,252	93,192
Blasting	US$ x 1,000			2,971	3,042	3,329	3,248	3,114	4,584	4,351	4,112	3,926	3,764	3,595	3,468	3,346	3,301	3,218	1,109	54,476
Engineering	US$ x 1,000			637	640	645	643	640	676	669	663	659	657	651	648	646	646	643	236	10,001
Geology	US$ x 1,000			398	400	403	402	400	423	419	415	412	411	407	405	403	404	401	147	6,249
Assay Lab	US$ x 1,000			1,013	1,076	1,077	1,077	79	79	79	79	79	79	79	79	79	79	79	29	5,140
Total Mining	**US$ x 1,000**			**19,368**	**19,848**	**21,670**	**20,596**	**18,821**	**28,223**	**26,599**	**25,394**	**24,834**	**23,896**	**23,124**	**22,359**	**22,331**	**22,050**	**21,602**	**7,876**	**348,591**
Processing																				
Plant G&A	US$ x 1,000			429	430	429	429	429	430	429	429	429	430	429	429	429	430	429	159	6,602
Crushing	US$ x 1,000			14,497	7,952	7,930	7,930	7,930	7,952	7,930	7,930	7,930	7,952	7,930	7,930	7,930	7,952	7,930	2,937	128,540
Conveyor Stacking	US$ x 1,000			1,979	1,181	1,178	1,178	1,178	1,181	1,178	1,178	1,178	1,181	1,178	1,178	1,178	1,181	1,178	436	18,921
Heap	US$ x 1,000			5,222	2,220	2,214	2,214	2,214	2,220	2,214	2,214	2,214	2,220	2,214	2,214	2,214	2,220	2,214	820	37,068
Reagents	US$ x 1,000			11,502	11,391	11,350	10,773	10,652	11,366	10,920	10,244	9,928	9,733	9,533	9,404	9,544	9,720	9,589	3,378	159,030
Agglomeration	US$ x 1,000			10,304	10,415	10,387	10,387	10,387	10,415	10,387	10,387	10,387	10,415	10,387	10,387	10,387	10,415	10,387	3,846	159,676
Recovery Plant	US$ x 1,000			1,334	1,155	1,152	1,152	1,152	1,155	1,152	1,152	1,152	1,155	1,152	1,152	1,152	1,155	1,152	426	17,895
SART plant	US$ x 1,000			15,693	15,734	15,691	15,691	15,691	15,734	15,691	15,691	15,691	15,734	15,691	15,691	15,691	15,734	15,691	5,811	241,350
Refinery	US$ x 1,000			709	396	395	395	395	396	395	395	395	396	395	395	395	396	395	146	6,394
Water Supply & Distribution	US$ x 1,000			1,017	445	444	444	444	445	444	444	444	445	444	444	444	445	444	164	7,399
Support Services/Plant Mtce.	US$ x 1,000			2,376	2,216	2,210	2,210	2,210	2,216	2,210	2,210	2,210	2,216	2,210	2,210	2,210	2,216	2,210	818	34,154
Assay Lab	US$ x 1,000			1,652	1,322	1,319	1,319	1,319	1,322	1,319	1,319	1,319	1,322	1,319	1,319	1,319	1,322	1,319	488	20,615
Total Processing	**US$ x 1,000**			**66,714**	**54,858**	**54,699**	**54,121**	**54,000**	**54,833**	**54,268**	**53,592**	**53,276**	**53,200**	**52,881**	**52,753**	**52,893**	**53,187**	**52,938**	**19,430**	**837,644**
Site General & Administration																				
Admin	US$ x 1,000			2,716	2,723	2,716	2,716	2,716	2,723	2,716	2,716	2,716	2,723	2,716	2,716	2,716	2,723	2,716	1,006	41,772
Human Relations	US$ x 1,000			503	504	503	503	503	504	503	503	503	504	503	503	503	504	503	186	7,731
Security & Safety	US$ x 1,000			536	537	536	536	536	537	536	536	536	537	536	536	536	537	536	198	8,238
Accounting	US$ x 1,000			353	354	353	353	353	354	353	353	353	354	353	353	353	354	353	131	5,428
Purchasing	US$ x 1,000			280	280	280	280	280	280	280	280	280	280	280	280	280	280	280	104	4,300
Environmental	US$ x 1,000			1,040	1,043	1,040	1,040	1,040	1,043	1,040	1,040	1,040	1,043	1,040	1,040	1,040	1,043	1,040	385	16,000
Total G&A	**US$ x 1,000**			**5,427**	**5,442**	**5,427**	**5,427**	**5,427**	**5,442**	**5,427**	**5,427**	**5,427**	**5,442**	**5,427**	**5,427**	**5,427**	**5,442**	**5,427**	**2,010**	**83,469**
Special Mining Tax	**US$ x 1,000**			**2,271**	**10,618**	**6,072**	**5,767**	**9,680**	**7,017**	**5,038**	**5,003**	**5,721**	**7,659**	**6,481**	**6,532**	**7,023**	**7,991**	**9,268**	**0**	102,141
Cash Operating Costs	***US$ x 1,000***			***93,780***	***90,766***	***87,867***	***85,911***	***87,928***	***95,515***	***91,332***	***89,415***	***89,259***	***90,196***	***87,913***	***87,071***	***87,673***	***88,669***	***89,235***	***29,316***	***1,371,846***
EBITDA	***US$ x 1,000***			***85,616***	***164,150***	***105,209***	***84,938***	***123,904***	***90,419***	***65,512***	***62,433***	***68,143***	***87,481***	***74,636***	***74,091***	***78,558***	***88,357***	***101,653***	***65,287***	***1,420,385***

Note: EBITDA = earnings before interest, taxes, depreciation, and amortization

Item	Units	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	LOM
Depreciation	US$ x 1,000			51,171	39,851	29,847	22,357	19,198	14,385	10,779	8,077	6,053	4,536	4,370	3,276	2,456	1,841	1,380	4,135	223,711
Reclamation Deduction	US$ x 1,000			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	131,953	131,953
Income - before NOL & Perc Depletion	**US$ x 1,000**			**34,445**	**124,299**	**75,362**	**62,581**	**104,705**	**76,034**	**54,733**	**54,355**	**62,090**	**82,944**	**70,266**	**70,816**	**76,103**	**86,516**	**100,273**	**(70,801)**	**1,064,721**
Corporate Income Tax	US$ x 1,000			(10,319)	(37,290)	(22,608)	(18,774)	(31,412)	(22,810)	(16,420)	(16,307)	(18,627)	(24,883)	(21,080)	(21,245)	(22,831)	(25,955)	(30,082)	0	(340,641)
Taxable Income, less Tax	**US$ x 1,000**			**24,077**	**87,009**	**52,753**	**43,807**	**73,294**	**53,224**	**38,313**	**38,049**	**43,463**	**58,061**	**49,186**	**49,571**	**53,272**	**60,561**	**70,191**	**(70,801)**	**724,029**
CASH FLOW CALCULATION																				
Adjustments for Non Cash Items																				
Depreciation/Reclamation/Salvage	US$ x 1,000			51,171	39,851	29,847	22,357	19,198	14,385	10,779	8,077	6,053	4,536	4,370	3,276	2,456	1,841	1,380	136,088	355,664
Net Operating Loss Adjustment	US$ x 1,000			50	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	50
Total Adjustments for Non Cash Items	**US$ x 1,000**			**51,221**	**39,851**	**29,847**	**22,357**	**19,198**	**14,385**	**10,779**	**8,077**	**6,053**	**4,536**	**4,370**	**3,276**	**2,456**	**1,841**	**1,380**	**136,088**	**355,714**
CAPITAL																				
Investment - Mine	US$ x 1,000	0	1,400																	1,400
Investment - Plant	US$ x 1,000	62,899	62,899																	125,798
Investment - G&A	US$ x 1,000	1,521	2,028																	3,550
Capital Indirects & Contingency	US$ x 1,000	21,170	27,933																	49,104
Total Capital	**US$ x 1,000**	**85,590**	**94,261**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**179,851**
Sustaining Capital - Plant	US$ x 1,000			18,791	4,785	0	0	7,659	0	0	0	0	0	3,879	0	0	0	0	0	35,114
Sustaining Capital - Indirects& Continge	US$ x 1,000			5,262	1,340	0	0	2,145	0	0	0	0	0	0	0	0	0	0	0	8,746
Reclamation	US$ x 1,000			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	131,953	131,953
Total Capital & Sustaining	**US$ x 1,000**	**85,590**	**94,261**	**24,053**	**6,125**	**0**	**0**	**9,803**	**0**	**0**	**0**	**0**	**0**	**3,879**	**0**	**0**	**0**	**0**	**131,953**	**355,664**
Working capital	US$ x 1,000		0	15,630	0	0	0	0	0	0	0	0	0	0	0	0	0	0	(15,630)	0
Total Capital & Working Capital	**US$ x 1,000**	**85,590**	**94,261**	**39,683**	**6,125**	**0**	**0**	**9,803**	**0**	**0**	**0**	**0**	**0**	**3,879**	**0**	**0**	**0**	**0**	**116,323**	**355,664**
Beginning Cash	***US$ x 1,000***	***0***	***(85,590)***	***(179,851)***	***(144,236)***	***(23,501)***	***59,099***	***125,262***	***207,951***	***275,560***	***324,652***	***370,778***	***420,294***	***482,891***	***532,568***	***585,415***	***641,142***	***703,544***	***775,115***	
Period Net Cash Flow	***US$ x 1,000***	***(85,590)***	***(94,261)***	***35,615***	***120,735***	***82,600***	***66,163***	***82,689***	***67,609***	***49,092***	***46,126***	***49,516***	***62,597***	***49,677***	***52,847***	***55,727***	***62,402***	***71,571***	***(51,036)***	***724,079***
Ending Cash	***US$ x 1,000***	***(85,590)***	***(179,851)***	***(144,236)***	***(23,501)***	***59,099***	***125,262***	***207,951***	***275,560***	***324,652***	***370,778***	***420,294***	***482,891***	***532,568***	***585,415***	***641,142***	***703,544***	***775,115***	***724,079***	***724,079***

22.5 Sensitivity Analysis

A sensitivity analysis was performed on the base case to determine Project sensitivity to gold, silver and copper prices and grade, operating costs and capital costs.

22.5.1 Metal Price Sensitivity Analysis

The Project, like almost all precious metals projects, is very responsive to changes in the price of its chief commodity, gold. From the base case price of US$2,300/oz, a change in the average gold price of 10% (US$230/oz Au) would change the NPV at a 5% discount rate by 18.7%, or approximately US$7.9 M (Figure 22.1).

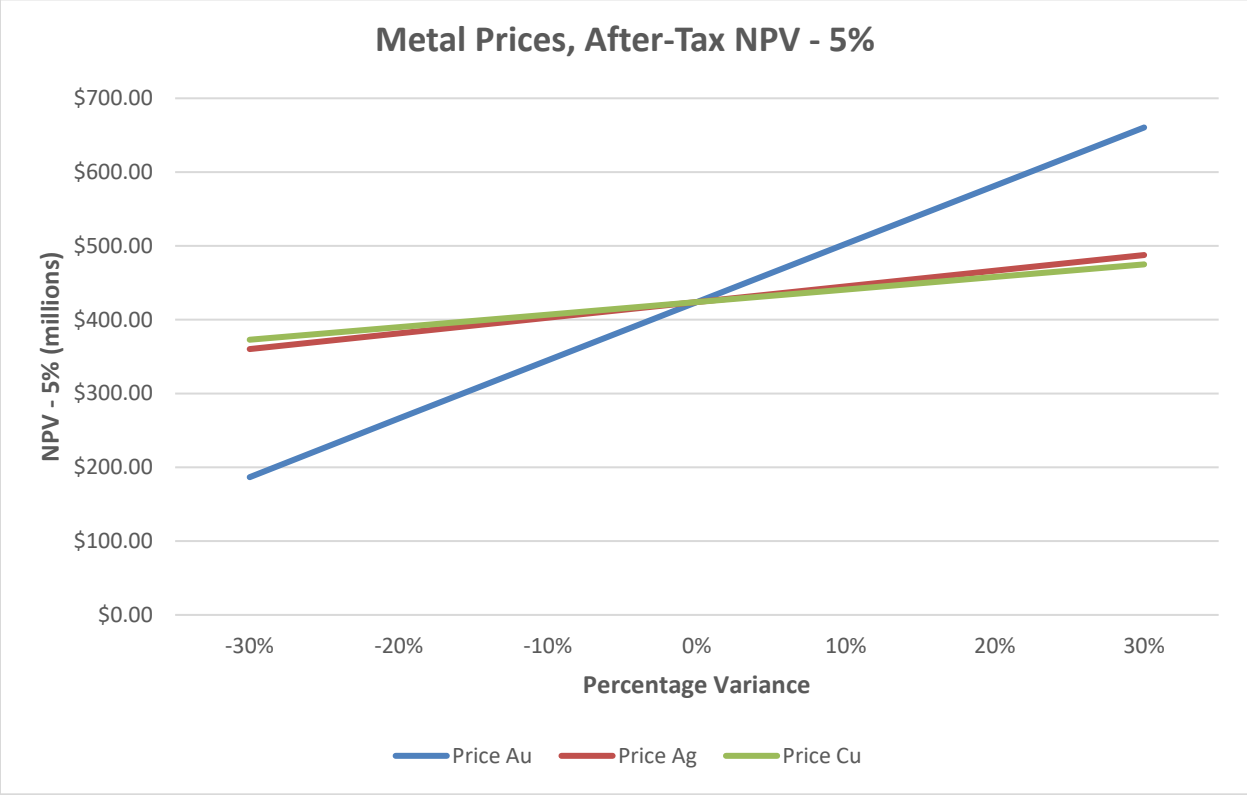
As shown in **Error! Reference source not found.**, the NPV is less sensitive to the silver and copper prices as they only represent approximately 17.9% and 15.1% of the gross revenue respectively. Table 22.3 shows the economic sensitivities, due to the change in gold price, in the net cash flow, the net present value at 5%, the internal rate of return, the payback period, and the payback multiple.

Table 22.3: Gold Price Sensitivity Analysis

Au Price (US$/oz Au)	Net Cash Flow (US$ M)	After-Tax NPV At A 5.0% Discount Rate (US$ M)	IRR (%)	Payback Period (years)	Payback Multiple
500	-410.0	-282.9	-	-	0.0
700	-186.5	-152.8	-	-	0.2
900	-43.4	-60.6	-	9.5	0.8
1100	66.1	9.9	6.1%	5.6	1.3
1300	176.6	79.9	12.4%	3.9	1.8
1500	286.0	148.8	17.3%	3.1	2.4
1700	395.4	217.6	21.6%	3.5	2.9
1900	505.3	286.8	25.7%	2.9	3.4
2100	614.7	355.4	29.5%	2.6	4.0
2300	***724.1***	***423.9***	***33.1%***	***2.3***	***4.5***
2500	833.4	492.5	36.7%	2.0	5.0
2700	942.8	561.0	40.1%	1.9	5.6
2900	1,052.2	629.6	43.5%	1.8	6.1
3100	1,161.6	698.2	46.8%	1.7	6.6
3300	1,270.9	766.7	50.0%	1.6	7.2
3500	1,380.3	835.3	53.2%	1.5	7.7
3700	1,489.7	903.8	56.3%	1.4	8.2
3900	1,599.0	972.4	59.3%	1.4	8.8
4100	1,708.4	1,041.0	62.3%	1.3	9.3

Note: Base case is highlighted.

Figure 22.1: Metal Price Sensitivity Analysis



Note: Figure prepared by Hard Rock Consulting, 2025.

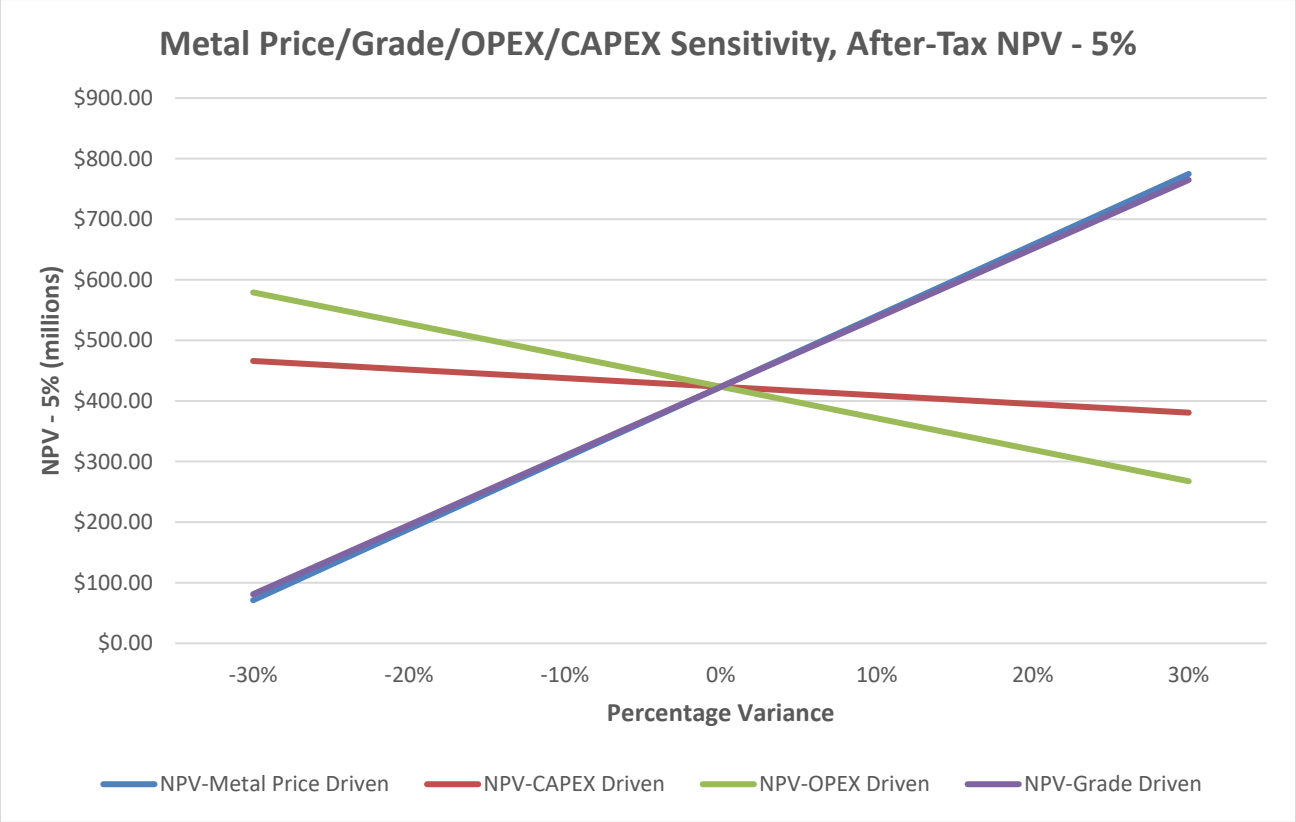
22.5.2 Grade, Operating and Capital Costs Sensitivity Analysis

The Project is most sensitive to grade experiencing an approximate 27% change in the NPV at a 5% discount rate for each 10% increase or decrease in grade.

The Project is also significantly sensitive to the cost of operations, incurring an approximately 12.2% decline in the NPV at a 5% discount rate for each increase of 10% in the operating costs.

The Project is less sensitive to variances in the cost of capital, experiencing an approximate 3.4% decline in the NPV at a 5% discount rate for each increase of 10% in the capital costs, as shown in Figure 22.2.

Figure 22.2: Project Metal Price, Grade, Operating Cost and Capital Cost Sensitivity Analysis



Note: Figure prepared by Hard Rock Consulting, 2025. CAPEX = capital cost estimate; OPEX = operating cost estimate.

23 Adjacent Properties

Heliostar advises the authors that there are no adjacent properties having any relevance to the Cerro del Gallo project.

24 Other Relevant Data and Information

24.1 Hydrology and Hydrogeology

The project is located within the Hydrological Region RH-12 “Lerma Santiago”, the RH-12 is made up of 6 Hydrological Sub-regions, and the project is part of the Hydrological Subregion of “La Laja.” The average monthly and annual precipitation, as well as their maximum and minimum values, are presented in Table 24.1.

Table 24.1: Cerro del Gallo Average Precipitation

Month	Precipitation Average	More wet year	Dryer Year
Jan	15.00	9.00	0.00
Feb	11.80	0.00	0.00
Mar	8.40	28.00	3.00
Apr	10.80	9.00	0.00
May	33.80	196.50	0.00
Jun	99.60	393.50	0.00
Jul	138.10	34.00	11.00
Aug	108.90	191.00	89.00
Sep	102.80	231.00	12.00
Oct	37.10	36.50	0.00
Nov	10.30	17.00	0.00
Dec	9.60	6.10	0.00
Anual	**586.20 mm**	**1,151.60 mm**	**115.00 mm**

Source: PEA (2021)

In 2013, Primero, through SRK, requested the support of Cardona Benavides y Asociados S.C. to carry out exploration work to identify two or more sites for drilling wells. The regional structures that were identified in the study area of CdG and its immediate vicinity, included the fault system: i) NW-SE and ii) NE-SW. The conclusions of the study allowed the identification of a structure that controls the flow of groundwater in the study area. It trends NW-SE and is called “Graben de Dolores.”

Based on the geological knowledge of the area of interest defined from the bibliographic analysis, geological and hydrogeological verification paths and geophysical exploration, from a qualitative point of view it is possible to propose that the following hydrogeological units exist for the analyzed area:

1. Free aquifer composed of granular material (mainly sands);
2. Semiconfining aquifer composed of fine material (silt and clays mainly);
3. Semi-confined aquifer composed of granular material and/or fractured rock.

According to the objectives of the work plan, as well as the hydrogeological knowledge of the area, the groundwater yield potential is considered to be related to two main factors:

- Thickness of unconsolidated materials of good to high permeability;
- Presence of areas with good fracture development in consolidated rocks.

The combination of these factors made it possible to identify the zones which, from the qualitative point of view, would offer the best conditions to capture groundwater. Information used for water table identification included hydrogeological, resistivity of geological materials at different depths, geological structures, and stream catchment areas. The results allowed to identify 3 zones with better possibilities for collecting groundwater, called NORTH, CENTER, and SOUTH.

In 2014, Primero drilled a water well on the suggested target from the study and drilled the SAM # 1 well 450 meters deep with 12-inch steel casing and flow test of around 10 L/s. The water well location is 3.9 km from the proposed mine water tank location. Results for the well pump tests are presented in Table 24.2 and indicate stable flow of 10.8 L/s and 12.0 L/s for two separate tests conducted at 235 meters and 250 meters.

Table 24.2: Cerro del Gallo Well (450-meter Depth) Pump Test Results

Well Pump Results to 235 m Depth

Stage	RPM	Consumption (L/s)	Pumping Level (m) A	Abatement (m) A	Specific Water Flow (L/s/m)	Accrued Time (min)	Note
1	1200	6.2	153.35	41.73	0.15	252	(1)
2	1300	8.83	184.63	73.01	0.12	460	(1)
3	1400	15.08 (?)	> 230.00			465	(2)
4	1350	10.81	219.24	107.62	0.10	620	(1)

Nota (A) The pumping and abatement level data are already corrected with respect to the ground surface level.
Note (1) Stage stabilized – complete.
Nota (2) Stage not stabilized – not complete.

Stage	RPM	Consumption (L/s)	pH	Temp (°C)	Electrical Conductivity (µS/cm)	Observations
1	1200	6.2	7.1-7.3	27.8-32.9	514-476	Turbid water, no sand
2	1300	8.83	7.3-7.2	32.9-32.5	492-467	Turbid water, no sand
3	1400	?	7.3-7.2	32.7-32.8	466-462	Clean water, no sand
4	1350	10.81	7.1-7.2	32.7-32.8	462	Clean water, no sand

Well Pump Results to 250 m Depth

Stage	RPM	Consumption (L/s)	Pumping Level (m) A	Abatement (m) A	Specific Water Flow (L/s/m)	Accrued Time (min)	Note
1	1200	6.74	150.55	37.85	0.18	210	*(1)*
2	1300	8.45	172.25	59.55	0.14	225	*(1)*
3	1350	10.19	188.00	75.3	0.14	710	*(1)*
4	1500	21.93	233.05	120.35	0.18	5	*(2)*
5	1450	11.96	241.77	129.07	0.09	25	*(2)*

Nota (A) The pumping and abatement level data are already corrected with respect to the ground surface level.
Note (1) Stage stabilized – complete.
Nota (2) Stage not stabilized – not complete.

Source: Primero (2014)

In 2015, Primero bought a parcel 7 km away from the footprint and within Zone South, this parcel has an existing water well with 204-meter depth and 12-inch casing and flow test of 16 L/s. Results for the well pump tests are presented in Table 24.3.

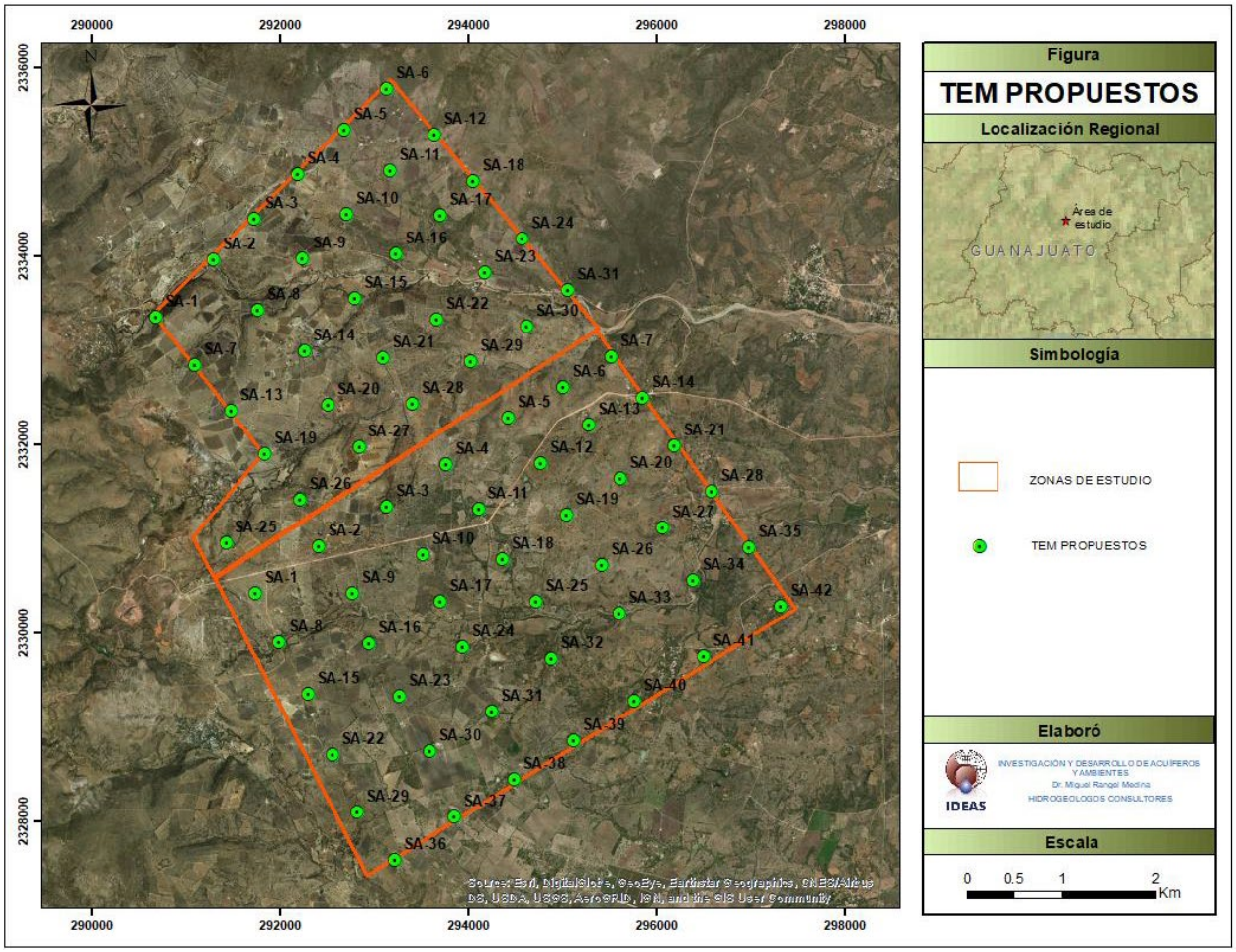
Table 24.3 Cerro del Gallo Well (204m Depth) Pump Test Results

Day	Hour	RPM Pump	Piezómeter In.	Piezómeter M.	Consumption Lps	N.D.	Abatement	Orifice	Observations
21	15:00	1500	5		17.00	130	10	5”	Turbid water
21	16:00	1500	5		17.00	150	30	5”	Turbid water
21	17:00	1500		0.32	14.19	151	31	5”	Turbid water
21	18:00	1500		0.32	14.19	151	31	4”	Turbid water
21	19:00	1500		0.32	14.19	151	31	4”	Turbid water
21	20:00	1500		0.32	14.19	151	31	4”	Turbid water
21	21:00	1500		0.32	13.03	151	31	4”	Turbid water
21	22:00	1500		0.27	13.03	156	36	4”	Turbid water
21	23:00	1500		0.27	13.03	156	36	4”	Turbid water
22	00:00	1500		0.27	13.03	158	38	4”	Turbid water
22	01:00	1500		0.27	13.03	158	38	4”	Turbid water
22	02:00	1500		0.27	13.03	158	38	4”	Turbid water
22	03:00	1600	5		17.00	168	48	5”	Turbid water
22	04:00	1600	5		17.00	168	48	5”	Turbid water
22	05:00	1600	5		17.00	168	48	5”	Turbid water
22	06:00	1600	5		17.00	169	49	5”	Turbid water
22	07:00	1600	5		17.00	169	49	5”	Turbid water
22	08:00	1700	7		20.50	173	53	5”	Less turbid water
22	09:00	1700	7		20.50	185	65	5”	Less turbid water
22	10:00	1700	7		20.50	190	70	5”	Clear water
22	11:00	Intermittent			-	-	-	-	-
22	12:00	1500		0.41	16.06	190	70	4”	Clear water
22	13:00	1500		0.41	16.06	190	70	4”	Clear water
22	14:00	1500		0.41	16.06	190	70	4”	Clear water
22	15:00	1500		0.41	16.06	190	70	4”	Clear water

Source: Primero (2015)

In 2019, Argonaut requested Investigación y Desarrollo de Acuíferos y Ambiente, S.A. DE C.V. (IDEAS), based in Hermosillo, Sonora, to review all the existing data and studies to define a third or even a fourth well with a single or combined capacity of about 25-30 L/s (additional water capacity required to meet project needs). The Technical Memo provided by IDEAS concluded that given the existing well flow capacities of 12 and 16 L/s, an additional 2 or 3 wells will be required to meet water flow needs. IDEAS recommended a detailed study using a type of magnetic resonance sounding imaging as a non-invasive geophysical method for groundwater investigation, which may yield more suitable locations for the water wells. The proposed test locations by IDEAS includes two separate zones with probe points every 500 meters, total of 73 points, as depicted in Figure 24.1.

Figure 24.1: Cerro del Gallo IDEAS Proposed Test Water Well Locations



Source: IDEAS (2019)

24.2 Project Implementation

The project will be developed in a manner similar to most other heap leach projects. Initial design, including basic design and procurement of long lead time items such as the HPGR and crushers, will be conducted during the early stages. Detailed engineering and procurement will be followed by the construction phase. The project implementation will most likely start after permit authorizations. Heliostar has elected to manage development of this Project internally with assistance from external consultants.

Typically, the detailed design phase of the project is separated into two parts: an initial basic and detailed engineering phase followed by final detailed engineering. The initial design phase of the project will include finalization of:

- P&ID's;
- Flow sheets;

- General arrangement drawings;
- Heap leach pad and pond earthworks; and
- Control philosophy.

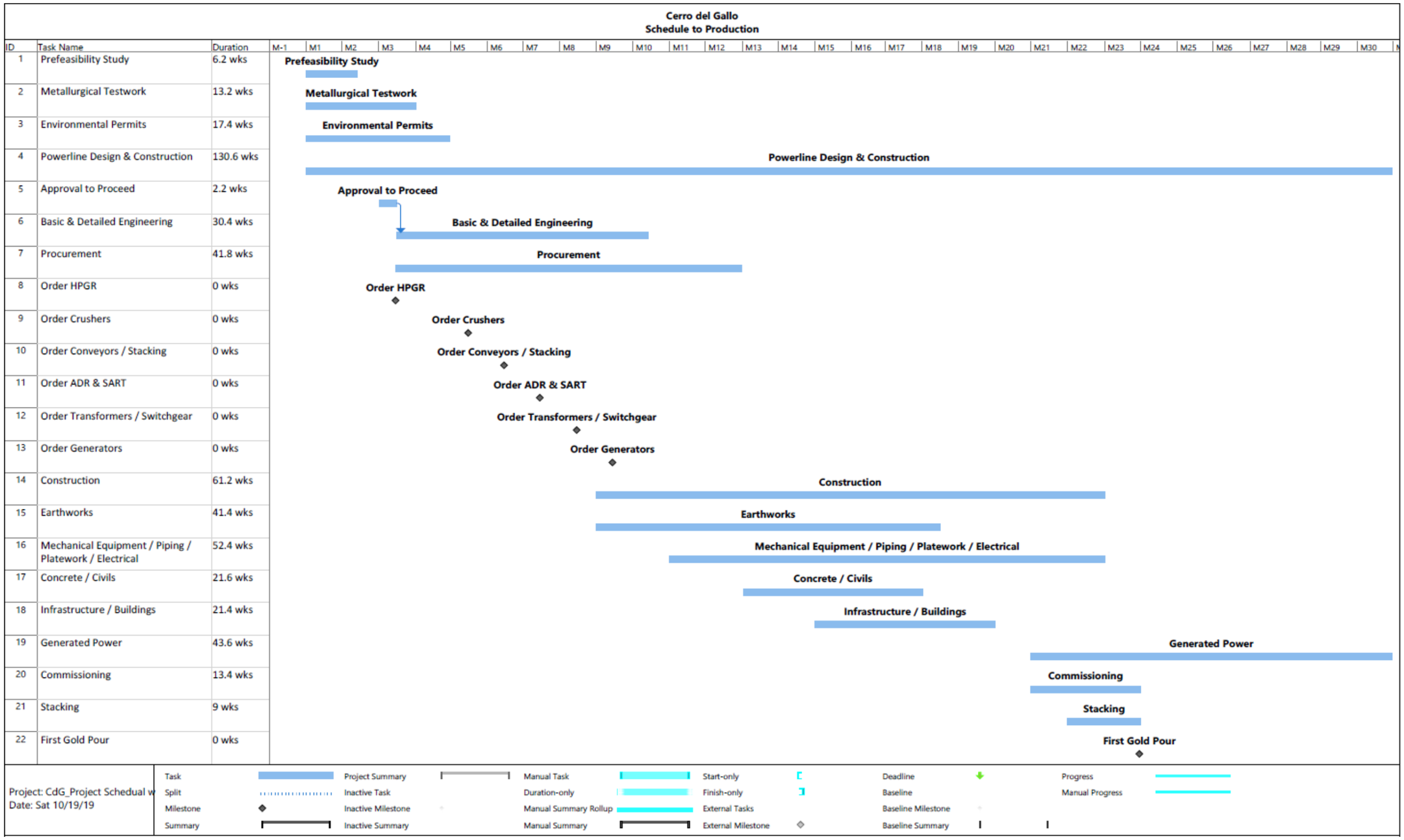
Specifications for long lead time equipment, a logistic study, finalization of site geotechnical work, detailed engineering for the leach pad and ponds, and detailed engineering for common infrastructure items will be completed early in this period. The heap leach facility and fresh water storage system earthworks will be designed to a level with sufficient detail to allow finalization of construction equipment requirements. Additionally, design of the power line will be completed in this period.

Final detailed engineering work will progress in areas and disciplines in a similar sequence to the initial design phase. Typically, this will include: earthworks, plate work, structural, civil, mechanical, piping, electrical, and instrumentation. Earthworks represent a large portion of the work both during initial construction and during future construction. Emphasis will be placed on completion of the earthwork design to facilitate start of the earthworks.

Final drawings for the various disciplines will be required. Some areas such as the crusher, process plant, the power generator package, and the stacking gear, will be packaged contracts. In these types of contracts, the vendor will supply detailed design and will also be responsible for a majority of the site work.

Assuming no delays due to permitting or social issues, approximately 21 months from start of engineering to first gold pour is anticipated. A preliminary schedule to production for the project is shown in **Error! Reference source not found.**.

Figure 24.2: Cerro del Gallo Project Schedule to Production



Source: KCA (2019)

24.3 Opportunities and Risks

There are opportunities and risks that have been identified in various areas of the project.

24.3.1 Mineral Resource Growth and Mineral Resource Conversion

Opportunities:

- If additional land can be acquired, there is the potential to increase total reserves as currently there are land constraints which limit the size of the heap leach and waste dump facilities.

Risks:

- The leach pad, as proposed, is in close proximity to the planned pit. If the pit limits are expanded toward the HLF as a result of subsequent mine planning, the volume capacity of the designed heap could fall below 92 Mt, requiring the development of a second HLF.

24.3.2 Metallurgy and Processing

Opportunities:

- As designed, the leach pad lining system consists of a manufactured crushed gravel overliner material placed above the geomembrane liner to promote solution flow into the gravity collection system. If the minimum permeability of the ore is at or above $1x10^{-1}$ cm/sec, the ore may be used as overliner.
- The treatment rate of 6 Mt/yr used in the study results in a mine life approaching 16 years. There is an opportunity to expand the process rate at some time in the early years of the operation to enhance the project economics.

Risks:

- Based on a limited geotechnical investigation, there is likely sufficient material within the property limits to source adequate clayey low permeable soil liner bedding material needed to be placed beneath the geomembrane. However, this is based on visual observations and discussions with the site geologist. An extensive investigation has not been completed, including test pits or boreholes to estimate the quality or quantity of clayey materials. There are also two nearby clay mines providing clay for tile production that may be sources for some of the required liner bedding material. A geotechnical investigation to quantify the volume of clayey materials on/near site is scheduled for 2020. GCL is an acceptable replacement for soil liner bedding over the majority of the HLF. Procurement and transporting the GCL may increase the overall HLF PFS construction cost.

- The HLF as designed is at its capacity within the CDG site. No additional capacity exists beyond the 92 Mt design, and an increase in leach feed will require purchasing additional property for the development of a second HLF.
- Preliminary geochemical test results indicate that the mine waste may be acid generating and if this is confirmed and quantified with further testing, then additional design measures may need to be added to the WRD design, such as larger contact water ponds, and possibly a base liner.
- Sulfide in the heap leach facility could turn acidic due to the sulfide content.
- Sodium cyanide starvation during initial leaching due to copper dissolution could delay gold and silver production.
- Potential need to lower the lift height from 8 m to 6 m due to possible cyanide starvation, which would increase leach solution requirements.
- Elevated levels of copper in the ore material may have a significant effect on potential economic extraction if not managed properly.
- Some gold may be present in the SART product due to poor washing. The value received for any gold in the SART product may be lower than that received from the sale of doré.
- The level of penalty elements in the SART product may be higher than currently estimated which could lead to higher treatment costs.
- Agglomeration must be closely followed to maintain heap permeability; poor agglomeration techniques could lead to permeability issues and/or increased cement requirements.
- The SART plant will treat all of the pregnant leach solution. Potential gold losses from treating pregnant solution in SART are variable and it is difficult to predict the amount of loss from previous testwork.
- The cover system as designed for the HLF and WRD consists from top to bottom an approximate 0.2-meter thick layer of topsoil, 1.0-meter thick layer of cover, and a 0.6-meter thick layer of clay. Based on the existing geotechnical information at the site, there may be insufficient clayey material on site to supply the clay portion of the system.

24.3.3 Mining

Opportunities:

- The oxidized portion of the deposit near the surface is substantially fractured. This may result in lower than expected drilling and blasting costs in that portion of the deposit, though blast hole drilling will be required for ore control purposes.
- A switch to owner-operated mining could reduce LOM mining costs.
- Given the bulk nature of the ore zones a switch to 6m mining levels versus the planned 5m levels should be analyzed to possibly lower the mine operating costs.

Risks:

- The steep terrain for mining in the upper benches may prove challenging for initial mining. This may delay some ore supply and increase development costs if not properly managed.
- Weather delays have not been included in the production estimate, and heavy rains may cause some delays with mining operations. Low spots on active benches should be maintained that can be used as sumps in case major rain events occur. Waste material inside the pit may be sought for use as gravel for haulage roads to help maintain traction during such events.

24.3.4 Water Supply

Risks:

- Although the Owner has sufficient water rights, only two wells have been tested and there is not sufficient water produced from these wells for the planned operation. Additional wells are required and a plan for the location and quantity of the wells needed is to be completed.

24.3.5 Power Supply

Opportunities:

- The current plan is to use generated power for the first year of operation, switching to line power in year two. If the supply of line power can be completed earlier, then there is the potential for savings in operating costs.

Risks:

- If the supply of line power is delayed, then the overall operating costs could increase due to extended site generation of power for the project.

24.4 Cautionary Statements

24.4.1 Forward Looking Information

This document contains “forward-looking information” as defined in applicable securities laws. Forward looking information includes, but is not limited to, statements with respect to the PFS, including but not limited to future production, costs and expenses of the Project; estimates of Mineral Reserves and Mineral Resources; commodity prices and exchange rates; mine production plans; projected mining and process recovery rates; mining dilution assumptions; sustaining costs and operating costs; interpretations and assumptions regarding joint venture and potential contract terms; closure costs and requirements; government regulations and permitting timelines; requirements for additional capital; environmental, permitting and social risks; and general business and economic conditions. Often, but not always, forward-looking information can be identified by the use of words such as “plans”, “expects”, “is expected”, “budget”, “scheduled”, “estimates”, “continues”, “forecasts”, “projects”, “predicts”, “intends”, “anticipates” or “believes”, or variations of, or the negatives of, such words and phrases, or statements that certain actions, events or results “may”, “could”, “would”, “should”, “might” or “will” be taken, occur or be achieved.

Forward-looking information is based on a number of assumptions which may prove to be incorrect, including, but not limited to, the availability of financing for production, development and exploration activities; the timelines for exploration and development activities on the Project; the availability of certain consumables and services; assumptions made in mineral resource and mineral reserve estimates, including geological interpretation grade, recovery rates, price assumption, and operational costs; and general business and economic conditions. Forward-looking information involves known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by the forward-looking information. These risks, uncertainties and other factors include, but are not limited to, the assumptions underlying the production estimates not being realized, changes to the cost of production, variations in quantity of mineralized material, grade or recovery rates, geotechnical or hydrogeological considerations during mining differing from what has been assumed, failure of plant, equipment or processes, changes to availability of power or the power rates used in the cost estimates, changes to salvage values, ability to maintain social license, changes to interest or tax rates, decrease of future gold prices, cost of labor, supplies, fuel and equipment rising, the availability of financing on attractive terms, actual results of current exploration, changes in project parameters, exchange rate fluctuations, delays and costs inherent to consulting and accommodating rights of local communities, environmental risks, reclamation expenses, title risks, regulatory risks and

uncertainties with respect to obtaining necessary permits or delays in obtaining same, and other risks involved in the gold production, development and exploration industry.

All forward-looking information herein is qualified by this cautionary statement. Accordingly, readers should not place undue reliance on forward-looking information. Heliostar and the authors of this Technical Report undertake no obligation to update publicly or otherwise revise any forward-looking information whether as a result of new information or future events or otherwise, except as may be required by applicable law.

24.4.2 Non-IFRS Measures

Heliostar has included certain non-International Financial Reporting Standards (IFRS) performance measures as detailed below. In the gold mining industry, these are common performance measures but may not be comparable to similar measures presented by other issuers and the non-IFRS measures do not have any standardized meaning. Accordingly, it is intended to provide additional information and should not be considered in isolation or as a substitute for measures of performance prepared in accordance with IFRS.

- All-in sustaining costs (AISC): as set out in the World Gold Council in its 2018 guidance note. AISC are the sum of operating costs (as defined and calculated above), royalty expenses, sustaining capital, corporate expenses and reclamation cost accretion related to current operations. Corporate expenses include general and administrative expenses, net of transaction related costs, severance expenses for management changes and interest income. AISC excludes growth capital expenditures, growth exploration expenditures, reclamation cost accretion not related to current operations, interest expense, debt repayment and taxes;
- Cash operating costs: include mine site operating costs such as mining, processing and administration, but exclude royalty expenses, depreciation and depletion and share based payment expenses and reclamation costs.

25 Interpretations and Conclusions

The work that has been completed to date has demonstrated that the CdG open pit mine and heap leach is a technically feasible and economically viable project. It is the conclusion of the QP's that the work completed in preparation of this technical report included adequate detail and information to declare Mineral Reserves. Standard industry practices, equipment and design methods were used in the PFS.

This report concludes:

- The base case for the Project has been developed with sufficient detail to underpin a decision to continue to move the Project through subsequent stages of development.

- The production schedule targeted a consistent total mine tonnage of approximately six million tonnes per year after the first year. The first year is approximately five and a half million tonnes. At this rate the expected mine life will be about 15.3 years.

- Pit reserves are divided into four different materials for processing: oxide, mixed oxide, mixed sulfide and sulfide.

- The mine life and total tonnes processed are limited by land constraints for the heap leach and waste dump facilities and metallurgical testing at depth.

- For the four material types, results of the metallurgical testing show gold recoveries in the range of 58% to 74%, silver recoveries range from 40% to 79%, and copper recoveries range from 22% to 59%.

- Tertiary crushing with two stages of conventional crushing and one stage of HPGR crushing are used to produce the final crush size of 80% passing 4 to 6 mm.

- Based on the production schedule and field adjusted recovery for the four material types, the average projected gold, silver and copper recoveries are 60%, 52% and 44%, respectively.

- Copper and silver will be recovered in a SART plant by sulfide precipitation; gold and some silver will be recovered by conventional carbon adsorption.

- During operations, control of cyanide levels and careful monitoring of copper in pregnant and barren leach solutions will be required to minimize cyanide consumption and maximize gold and silver recoveries.

- Cement for agglomeration constitutes a significant portion of the operating costs. Careful monitoring of heap permeability and ongoing test programs at site will be required.

- The CdG Project requires supplies and infrastructure items such as administration buildings, laboratory, warehouse, roads, powerline and MCC, water wells and water line, reagent storage, a complete service truck shop; and dining area.

- The economic analysis indicates that the profitability of the potential operation will be driven by gold price, operating costs and capital costs. Given the lower grade nature of the deposit and the strip ratio, well over half of the revenues are consumed by the operating costs. Therefore, a focus on controlling costs will be important in maintaining robust project economics.

- The PFS design of the HLF consists of a geomembrane lined pad and process ponds and is intended to operate as a zero-discharge system to groundwater and surface water. The HLF is designed with a geomembrane lined satellite pond that is available to store solution, if necessary, during the 1 in 100 wet year climate as described in Section 17.2.3.

- The HLF provides suitable capacity to store and leach the 92 Mt of ore identified through this study. The HLF is designed to North American standards used for the design of these facilities, as described in Section 17. The ultimate HLF configuration meets the minimum factors of safety contained in Section 17.2.4 with respect to geotechnical stability. The HLF has been designed to withstand a reasonably foreseeable earthquake.

- For the planned HLF design, the 92 Mt of ore storage capacity requires a maximum heap height of 80 meters. The heap leach pad, as designed, is at its limit with respect to storage capacity for the proposed site, constrained by surrounding topography and the planned location of the proposed pit and the WRD.

- The PFS design of the WRD has an excess capacity to store 54.4 Mt of waste rock with only 29.3Mt being planned to be stored in the facility for this study. An additional 3.4 Mt of waste rock identified in this study is planned for use as

structural fills for construction of the HLF. The ultimate WRD configuration meets the minimum factors of safety contained in Section 16.2 with respect to geotechnical stability

- For the planned WRD design, the 55 Mt of waste rock storage capacity requires a maximum waste rock dump height of 100 meters. The WRD, as conceptually designed, is at its limit with respect to storage capacity for the proposed site, constrained by the property boundary to the north and east, the pit to the west, and the HLF to the south.

- The WRD, as designed, is assumed to be progressively reclaimed during operations to reduce the amount of contact water that may need to be managed during operations. The HLF as designed is assumed to be rinsed in order to remove unwanted constituents from the ore, which may include metals and cyanide prior to receiving the closure cover.

25.1 Geology and Mineralization

The CdG deposit is located in central Mexico within the Mesa Central physiographic province that includes the Guerrero Composite Terrane which is characterized by submarine and subaerial volcanic and sedimentary successions that range in age from Jurassic to Middle–Late Cretaceous. The oldest rocks in the CdG region are a Triassic to Cretaceous age, deformed and regionally metamorphosed volcano-sedimentary sequence of the Esperanza Formation, described as carbonaceous and calcareous shale interbedded with arenite, limestone and andesite to basaltic flows, all weakly metamorphosed to phyllites, slates and marble. A swarm of basaltic dykes intrude rocks of the Esperanza Formation predominantly along northwest trending fracture zones. The stratigraphy strikes broadly east-west and dips northerly at 30-40°, except in the vicinity of CdG where dips steepen to greater than 60°.

CdG (the actual hill) is cored by a steep sided stock of granodiorite to monzogranitic composition with a surface exposure measuring approximately 600 meters in a north-south direction and 300 meters wide (**Error! Reference source not found.**). Gold and copper are spatially and genetically associated with this intrusive complex occurring largely as an up-right quartz vein stockwork/breccia system formed peripheral to the intrusive (**Error! Reference source not found.**; **Error! Reference source not found.**). CdG fits well into a copper-gold porphyry mineralization model in that mineralization is related to volatile phases of a felsic intrusive complex. This mineralization is genetically and spatially associated with this intrusive and is proximally zoned concentrically around

it. Copper locally overlaps the gold zone and extends outward from the intrusive farther than the gold system. Silver appears to be structurally related and occurs as overlapping epithermal veins and is clearly younger than the gold-copper system.

Several post-mineral faults locally offset mineralization and act as hard boundaries to mineralization.

Hydrothermal alteration is zoned concentrically around the main intrusive complex and intensity decreases outwards. Potassic alteration is the dominant style of alteration in the core of the hydrothermal system which is centered on the Main Intrusive. Within and outward from the main intrusive there is a general absence of primary textures due to replacement by silica. Tuffaceous wall rocks in contact with the intrusive system have experienced pervasive hydrothermal alteration to form fine grained replacement assemblages of albite, K-feldspar, biotite, sericite, quartz, sphene, rutile and pyrite. Hydrothermal propylitic alteration overprints potassic alteration within both the felsic intrusives and surrounding wall-rocks and extends laterally beyond the zone of potassic alteration surrounding CdG.

Much of the mineralization is fracture controlled and related to K-feldspar, biotite, pyrrhotite, and chalcopyrite, but locally forms thicker granular textured veins dominated by quartz with minor biotite, chlorite, calcite, pyrite, pyrrhotite, chalcopyrite, molybdenite and bismuthinite.

The geological setting and local geology are adequately known to support PFS-level mineral resource estimation and mine planning. Core and cuttings should be relogged to a common nomenclature to improve the current geological models. The extent of the alteration is not well documented in the geology logs which precludes developing a 3-D alteration model. The QP recommends that drill core and cuttings be relogged for alteration as well.

25.2 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

Drilling was completed mostly in 2004 through 2008 by Cerro Resources with both core (77 holes; 37,359.97 m) and reverse circulation (RC) (282 holes; 60,100.80 m) drill types. Primero drilled 51 core holes (12,829.16 m) in 2013 and Argonaut drilled 18 core holes (1484.5 m) in 2018.

Mineralization generally displays a concentric distribution around the carapace of the intrusive, drilling was predominantly performed on nominal 50-meter fences, oriented NNE-SSW to intersect a known north-westerly trending structural corridor. Holes were normally angled between 55 and 65º, either to the northeast or southwest, depending on the position of the drill fence (or drill section). Specifically, drill fences were oriented to alternate every other section between NNE bearing and SSW bearing. These holes can locally crisscross at depth. Drill spacing within the resource area averages approximately 50 meters.

RC drill samples obtained by CdG were logged at five-foot (1.52-meter) intervals on-site at the time of drilling. A split from each interval was sieved, washed and logged visually, using a hand lens. Logging criteria included: lithology, alteration, degree of oxidation, and mineralization.

Geological logging of core included lithological, alteration, structural, and mineralization logging. Core was then sawed in half with one half was sent for assay and the other half retained in the original core box.

After completion of the hole, final hole locations were determined by a contract surveyor using a total station instrument. Down-hole surveys were completed for all core and RC holes on the project. Initially, down-hole surveying was done using an Eastman single shot camera. After 2004, a Reflex EZ-Shot electronic solid-state single shot survey tool was implemented.

The QP concluded that drilling at CdG was completed using methods and procedures that are commonly used in the industry. Collar surveys, downhole surveys, and core orientation were completed using instruments that are widely used in the industry.

Geological logging was performed by two operators using two different nomenclatures that are not consistent. The QPs and Heliostar personnel used the data for lithology and oxide state to generate new descriptors. The QPs consider this to be an acceptable “fix” for this stage of the project, but recommend that all core and RC cuttings be relogged using a standard nomenclature for the next stage of the project.

Sample lengths are appropriate and consistent with industry procedures.

25.3 Environmental, Permitting and Social Interpretation and Conclusions

The following observations and conclusions have been developed based on the site visit, discussions between the QP and Company personnel, and review of available information.

Environmental baseline studies have indicated that the Project is in a rural region of high flora biodiversity. The QP received a summary of the baseline studies for flora and fauna without supporting workplans or field procedures, precluding a determination of the quality of the work. Due to the use of the area and the nearby community, the QP does not expect a high fauna diversity.

Surface water quality generally meets Mexican permissible limits for discharge standards. Groundwater quality does not meet potable water quality standards, and elevated arsenic and lead, possibly associated with mineralization, were noted during some sampling events. In addition, the QP notes that the data acquired may not represent the aquifer characteristics. The sampling method and the quality of the well construction can preclude obtaining a representative water quality sample. The QP notes that Mexico does not have water quality standards for naturally-occurring groundwater; the standards are related to discharge to water bodies and drinking water usage.

There are no established protected areas within the Project or in the immediate area, nor any Indigenous rightsholders; however, a determination is pending as to whether there is a federally protected drainage.

The mineralization for the Project includes sulfide minerals that have the potential to produce acidic conditions and metals leaching. Geochemistry studies of the future mining wastes (that is, spent ore and waste rock) have been carried out based on static and kinetic testing to characterize potential environmental impacts during operations and post-closure. The results indicated that 70% of the waste rock and all of the spent ore will be classified as a hazardous waste and require special handling based on Mexican regulations for mining waste. Leaching of some metals may be problematic in the waste rock, and the spent ore may discharge elevated concentrations of arsenic, cadmium, and selenium. During operations, leachate from the HLF will be managed with a leachate collection system. The WRSF design does not include a liner, and future seepage and surface runoff will need to be managed. The management options will be evaluated as the Project advances.

A permitting process was started by a previous owner, but has lapsed. The Company will need to restart the permitting process with a new MIA application. At the time of this report, the Company had not determined whether any of the historic baseline data would

be used, but had indicated that new data would also be collected to support the permit application.

The determination of whether any drainage within the Project is within a federally-protected zone is pending. If the drainage is within the federal zone, then the Company plans to include a diversion channel in the mine design to mitigate disturbance to the drainage.

The closure plan is at a conceptual stage and includes a variety of assumptions. Closure costs could increase as closure planning advances.

No material issues related to the environmental, permitting or social aspects of the Project were noted; however, the permitting process in Mexico has been significantly delayed under the current political administration.

26 Recommendations

Based on discussions with Heliostar's technical staff and the Qualified Persons responsible for this Technical Report, the following recommendations have been made to advance the CdG Project:

26.1 General Recommendations

- A feasibility study should be conducted if the Owner wants to reduce the outlined risks and increase the accuracies of the cost estimates. This study cost would be approximately US$750,000.

- Conduct additional SART tests to optimize reagent requirements and better define elements associated with the final product. The cost would be approximately US$100,000.

- A detailed groundwater investigation to determine suitable locations for additional water wells. The current water well supply is not adequate to meet the projected water requirements for the project. The cost would be approximately US$25,000.

- Metallurgical testing should be conducted on the material deeper in the pit. The majority of the material at depth is classified as Sulfide. The testwork program should include both column leach tests with comparable parameters to those utilized for the design in this report and flotation tests. There is core in the shed from the Argonaut drilling program, but it would need to be demonstrated that the sulfides have not been altered metallurgically due to oxidation while in storage. The metallurgical program would cost approximately US$50,000

26.2 Heliostar Recommendations

Assay Database

- Review 2013 Ag and Cu assays done by the 0.5g ICP method vs the 5g AA method / 3 acid digestion (SGS AAS21E). Statistically compare the two methods by re-assaying some of the pulps using both methods. (This work is in progress.)

Drill Hole Database

- Audit the remaining 75% of down-hole survey data. Cost approximately US$5,000.

Drilling

- Infill drill the NNE part of the deposit, in an around the peak of the hill. This cost is estimated to be US$80,000.

- Select an area of perceived higher-grade variance and reduce the drill spacing to 25 x 25m or less to assess the short scale variance. Recommend this area to be within the years 1 – 2 pit plan as an aid to mine planning and ore control reconciliation. This is estimated to be 2,000 to 3,000 m of RC drilling or approximately US$250,000.

Density

- Collect approximately one hundred more density measurements in the oxide profile. This program would cost approximately US$25,000.

Modeling

- The oxide domains, intrusive lithology and grade models generated in Leapfrog are overly complex given the nominal 50 x 50m drill-hole spacing. These need to be smoothed. This would cost less than US$5,000.

- Develop the structural planes into domains and assess the validity of the implied fault displacements using geostatistics. This would cost roughly US$5,000 to $10,000.

- The surface topography should be extended, particularly to the east and NE where higher silver grades are not currently being captured in the reporting. This could be done in-house for less than US$5,000.

- Model statistics could be further upgraded by: (1) further assess the multi-variate statistics to better understand the relationship between the three potential

economic metals; (2) complete further assessment of the level of grade smoothing in the estimate. This would likely cost less than US$5,000.

- Silver model: silver has been estimated with the same radial framework as Au and Cu, though it is known that there is a secondary late-stage silver event. Further assessment is warranted of whether the current estimation framework is optimal. The initial review of this would cost about US$2,500.

26.3 Data Collection and Evaluation Recommendations

- The QP recommends that all available core be relogged using consistent nomenclature. A second phase of relogging would include checking available RC chip trays to conform to the core nomenclature.

- The QP also recommends that all available core drilling be relogged for oxidation type and subsequently all RC chip trays be checked to conform to the oxidation nomenclature.

- The QP recommends additional density data be obtained from future drill programs.

- The QP estimates that this will require two geologists and four support personnel six months to complete. The cost would be approximately US$25,000.

- The QP recommends the relogging campaign include controls for silver mineralization and correlation of silver veining if applicable.

- Once the re-logging is completed, the data must be incorporated into a new geological model. The QP estimates three months for one senior geologist to complete this. The cost would be approximately US$15,000.

26.4 Environmental, Permitting and Social Recommendations

Based on the observations and conclusions, the Environmental, Permitting and Social QP makes the following recommendations for the next phase of project advancement:

- Set up a comprehensive library and relational database for the Project.

- Continue to advance the baseline studies. Include a hydrogeologic study that characterizes the hydraulic properties for the purposes of the environmental permitting and estimation of in-flow to the open pit. Establish monitor wells upgradient, downgradient and cross-gradient of the mining facilities. Establish processes and procedures for long-term environmental monitoring.

- Any additional drilling should consider a combination of exploration, geotechnical and hydrogeologic purposes. Carry out hydraulic testing and installation of either vibrating wire piezometers and/or monitor wells to monitor water levels.

- Continue the activities planned for social engagement. Include initiatives that are collaborative efforts between local and state government and shared responsibility.

The estimated cost of the recommended studies for the next phase is US$1M.

Recommendations to continue to advance the Project in a later phase are as follows:

- Advance the geochemistry studies to assess the geochemical source terms based upon the current mine design. Assess the need and methods for special handling and management of waste rock lithologies that will be acid-generating or leach metals during operations and/or closure. Include the prediction of water quality in the future pit lake, should the hydrogeologic study indicate that a pit lake will form after closure.

- Start barrel testing of the waste rock and HLF materials.

- Advance the closure planning. A feasible, yet proactive reclamation plan should be developed for the HLF and WRD. This plan will consist of a closure cover design, quantities necessary for closure, permanent stormwater facilities design, and monitoring requirements. Sediment and erosion control and storm water management will be challenging throughout operations. To reduce potential sediment transportation and erosion of operational slopes, it is recommended to implement a concurrent reclamation plan to actively reclaim completed HLF and WRD slopes during operation. This will include re-sloping, placement of cover soil, placement of a low permeable soil liner, placement of topsoil, and revegetation. In addition to reducing sediment and erosion management efforts, placement of a low permeable cover over the HLF and WRD during operation will reduce the risk of excess solution to manage during rinsing.

- Develop a closure water balance for the heap leach facility to estimate the cost of HLF rinse water and draindown solution treatment. Plan for pilot studies of closure cover options.

- Evaluate the possibility to construct a passive water treatment system instead of an active water treatment (that is, construction of a water treatment plant), should discharge management be required post-closure.

- Develop a community and interested parties engagement plan with processes and procedures for recording, acting upon and resolving issues as they arise. A formal grievance mechanism should be described in the plan. The community has indicated that visual impacts from mining are a concern. Create visual graphics that show the final closure scenario from the viewpoint of the town of San Anton de las Minas. These graphics would be used to illustrate the view so that the local community understands the changes expected over time.

26.5 Geotechnical Recommendations

- Perform a detailed geotechnical characterization of the available clays within the property limits of the CDG mine to evaluate if there are sufficient clayey materials on site to be used as the low permeable soil liner bedding beneath the leach pad and ponds geomembrane liner. If future investigations identify an insufficient volume of clayey materials on site (liner bedding), then a geosynthetic clay liner (GCL) may be substituted over a majority of the leach pad. The use of a GCL instead of a locally sourced clayey soil material will likely increase the HLF's construction cost. If the HLF PFS construction cost estimate is based on the use of soil liner bedding, then add a 30% contingency to the HLF cost. If the owner elects to base the HLF PFS cost on the use of a GCL instead of soil liner, then reduce the contingency to 25%. The cost would be approximately US$100,000.
- The close proximity of the proposed HLF to the proposed pit may pose a stability risk of the pit walls to the Phase 3 heap leach pad expansion as it is currently proposed. Further stability analyses will be needed to evaluate if the risk is acceptable or if further stabilization efforts will be needed. The cost would be approximately US$30,000.

- If the geochemical investigation identified ARD (acid rock drainage) as a potential issue, then performing in-situ infiltration testing throughout the WRD foundation to characterize the permeability of the foundation soils is recommended. The foundation soils permeabilities are necessary to characterize the amount of acid waste rock drainage that would infiltrate into the foundation soils and subsequently, the underlying aquifer. Based on this quantity of infiltration, alternative design measures may be necessary. The cost would be approximately US$100,000.

- A meteorological station has been constructed near the leach pad site to gather climate data to compare with the data from the weather station used for the water balance. The weather data from near site will be used to better refine the predictions of the HLF process fluid water balance. More accurate precipitation

and evaporation data for the site will provide a higher confidence in estimating process fluid consumption and pond storage capacities. This comparison is necessary to confirm the climate variables used for the study. The cost would be approximately US$40,000. To update the site climate and water balance.

- To facilitate feasibility and detailed design of the HLF and WRD, additional geotechnical field investigation and laboratory testing programs are recommended. Subsurface conditions should be characterized in sufficient detail to provide a high level of confidence in the stratigraphy, ground water, structure, and preferential paths of flow in the upper 30 meters of the subsurface and summarized in a feasibility study level geotechnical report. This can be accomplished through the advancement of geotechnical boreholes located within the footprint of the facility. The depth and quantity of boreholes will depend on the final design of the facilities. Additionally, a shallow subsurface investigation should be performed to characterize the near surface soil conditions across the entire site to better delineate vegetative root mass and depth, stripping depth of topsoil and weak soils, near surface bedrock, and mass grading requirements. This can be performed using a track-mounted excavator or by hand-excavation of exploratory test pits. The cost would be approximately US$200,000.

- Further consideration should be given to evaluating the hydrogeologic conditions that exist at the HLF and WRD sites. Install ground water monitoring wells through installation of slotted PVC well casings in future geotechnical boreholes. Additionally, a comprehensive ground water spring study is recommended to better delineate and quantify year-round or seasonal spring flows below the proposed HLF and WRD sites. This study will assist is adequately sizing underdrain collection piping installed below each facility's foundation. The cost would be approximately US$200,000.

- Monitoring of existing stream flows should be considered to measure sediment transportation in existing streams. This will provide valuable input for designing sediment control structures downstream of proposed improvements. The cost would be approximately US$20,000.

27 References

ALS Ammtec, 2012, Metallurgical Testwork, Report No. A13846, Metallurgical Testwork conducted upon Gold Ore HPGR Feed Samples from Cerro del Gallo Project, March 2012, prepared for Kings Minerals NL/Sedgman

ALS Ammtec, 2011, Heap Leach Amenability Testwork, Report No. A13259, Heap Leach Amenability Test work conducted upon Composites of Gold Ore Samples from Cerro Dal Gallo Gold/Silver Project, October 2011, prepared for King Minerals NL

Campa, M.F., and Coney, P.J., 1983. Tectono-stratigraphic terranes and mineral resource distributions in Mexico, Canadian Journal of Earth Sciences, 1983, 20(6): 1040-1051

Centro de estudios Mineralógicos, 2015, Informe Petrocalcográfico Solicitado por Primero Mining Proyectos Cerro del Gallo y Ventanas-Mala Noche; November 2015, Report prepared by Centro de estudios Mineralógicos Geoatacama.

Champion, D., 2005. Prospects look good in North Queensland. AusGeo News September 2005, 79: pp3-6

CONAGUA, 2024, Actualización de la disponibilidad media anual de agua en el acuífero Cuenca Alta del Rio Laja (1108), estado de Guanajuato, 39 p.

Consejo de Recursos Minerales, 1992. Monografía geológico-minera del estado de Guanajuato. Secretaria de Energía, Minas E Industria Paraestatal 35, pp55-57

Defilippi, C., Dyer, T., Minard, T., Arkel, B., and Zurkic, N. 2020, Pre-Feasibility Study NI43-101 Technical Report Cerro del Gallo Heap Leach Project Guanajuato, Mexico, 31 January 2020, NI 43-101 Technical Report prepared by Kappes, Cassiday & Associates, Reno NV, for Argonaut Gold, 351 p.

Dickinson and Lawton, 2001. Carboniferous to Cretaceous assembly and fragmentation of Mexico: Geological Society of America Bulletin, v. 113, p. 1142-1160

Golder Associates, 2006. NI43-101 Technical Report on the San Antón Property, Mexico – 021-05763009, 2 October 2006, NI 43-101 Technical Report prepared by Golder Associates Pty Ltd.

Golder, 2019. Pre-Feasibility Design Report: Heap Leach and Waste Rock Dump Facilities, Cerro del Gallo Project. Prepared for Argonaut Gold, Inc. by Golder Associates Inc. Dated December 2019.

Golder, 2020, Heap Leach and Waste Rock Dump Facilities Pre-Feasibility Study Design Report: Unpublished report prepared for Argonaut Gold Inc., February, 1094 p.

Groves, I., 2007, San Antón Project Carmen-Providencia Espíritu Santos, Ave de Gracia Epithermal Quartz Vein Systems Geological Report 1:2500 Scale Mapping Project 24th April 2007 to 21st May 2007; 12 June 2007, Report prepared by Insight Geology Pty Ltd for San Anton De Las Minas Se de CV; 32 p.

Groves, I., 2008. San Antón Project, Cerro del Gallo Cu-Au-Ag Deposit, Guanajuato State, Mexico. Unpublished report by Insight Geology Pty Ltd for San Antón de las Minas S.A. de C.V., January 2008

Groves, I., and Carman, C., 2007, San Anton Project, Multielement Stream and Soil Geochemistry, Guanajuato State Mexico; May 2007, Report prepared by Insight Geology Pty Ltd for San Antón de las Minas S.A. de C.V, 61 p. (two volumes)

Hart, C.J.R., 2005. Classifying, distinguishing and exploring for intrusion-related gold systems. The Gangue 87:pp1-9

Heliostar, 2025, Estudio de seguridad, análisis y evaluación de riesgos para el Proyecto San Antón de las Minas – CDG: Internal study, June, 24 p.

Herrera y Sato, S.C., 2024, Manifestación de Impacto Ambiental, Modalidad Regional, Proyecto Recinto Minero Cerro del Gallo: Draft document prepared for San Antón de las Minas, S.A. de C.V., April.

Independent Metallurgical Laboratories PTY LTD, 2006. Cerro del Gallo Testwork Program (SA002, SA007, SA010 & SA011), April 2006, prepared for Kings Minerals NL

Independent Metallurgical Laboratories PTY LTD, 2006. Cerro del Gallo Testwork Program (SA057, SA058, SA061, SA062, SA071 & SA078), April 2006, prepared for Kings Minerals NL

Kappes, Cassiday & Associates, Pending. Cerro del Gallo Project, HPGR Crushed Bottle Roll Leach Test Work, Report of Metallurgical Test Work. Report ID KCA0190035_CDG02_01 prepared for San Antón de Las Minas S.A. de C.V.

Kappes, Cassiday & Associates, 2019. Cerro del Gallo Project, Locked Cycle HPGR and Conventional Crushing Test Work, Report of Metallurgical Test Work. July 2019. Report ID KCA0180034_CDG01_01 prepared for San Antón de Las Minas S.A. de C.V.

Kappes, Cassiday & Associates, 2020, Pre-Feasibility Study, NI43-101 Technical Report, Cerro del Gallo Heap Leach Project, Guanajuato, Mexico: Report prepared for Argonaut Gold, 1 January, 351 p.

Kappes, Cassiday & Associates, 2020. Cerro del Gallo Project, Column Test Work with SART, Report of Metallurgical Test Work. March 2020. Report ID KCA0190078_CDG02_03 prepared for San Antón de Las Minas S.A. de C.V.

Kappes, Cassiday & Associates, 2020. Cerro del Gallo Project, Bottle Roll Leach Test Work, Report of Metallurgical Test Work. December 2020. Report ID KCA0190035_CDG04_01 prepared for San Antón de Las Minas S.A. de C.V.

Lapierre et. al., 1992. A crustal section of an intra-oceanic island arc: The late Jurassic-early Cretaceous Guanajuato magmatic sequence, central Mexico: Earth and Planetary Sciences Letters, v. 108, p. 61-77, doi: 10.1016/0012-821X(92)90060-9

Martínez Reyes, J.J., Miranda Avilés, R., and Puy Alquiza, M.J., 2014, Análisis Petrográfico Y Mineragráfico Muestras Del Proyecto “CERRO DEL GALLO”; April 2014, Report prepared by Laboratorio de Petrología, Departamento De Minas, Metalurgia Y Geología, Universidad de Guanajuato for Primero Mining.

Mason, D.R., 2005a. Petrographic descriptions for eleven rock samples from the San Antón Project, Mexico. March 2005. Unpublished report no. 3064 by Mason Geoscience Pty Ltd for KMN NL

Mason, D.R., 2005b. Petrographic descriptions for rock samples from Cerro del Gallo porphyry system (San Antón), and from Valenciana, central Mexico. August 2005. Unpublished report no. 3104 by Mason Geoscience Pty Ltd for KMN NL.

Mason, D.R., 2006a. Petrographic descriptions and interpretation for four drill chip skarn rock samples from the Cerro del Gallo porphyry system (Mexico). January 2006. Unpublished report no. 3149 by Mason Geoscience Pty Ltd for KMN NL

Mason, D.R., 2006b. Petrographic study of 12 drill core rock samples from the Cerro del Gallo gold prospect, Mexico. June 2006. Unpublished report no. 3196 by Mason Geoscience Pty Ltd for KMN NL

Mason, D.R., 2007. Petrographic study of ten drill core rock samples from Cerro del Gallo porphyry system (San Antón). Unpublished report by Mason Geoscience Pty Ltd for KMN NL

McPhar Geosurveys, 2005, Final Report on a Helicopter-borne Geophysical Survey San Anton Property Mexico; April 2005, Report prepared by McPhar Geosurveys Ltd. for Kings Minerals NL., 34 p.

McBride Sustainability, 2022, Estudio de Línea de Base Social, Proyecto Mina Cerro del Gallo: unpublished report prepared for Argonaut Gold, 156 p.

Mine Development Associates, 2012 Feasibility Mine Study Update, Cerro del Gallo, Guanajuato, Mexico. June 2012, prepared for Cerro Resources NL

Perez, E., 2018. Estudio petrograficas y minerografico Cerro del Gallo, Guanajuato, Mexico. Unpublished report by Dr. Efrain Perez Segura prepared for Argonaut Gold Co.

PH Consultores Ambientales S de RL (PHCA). 2014. “Caracterización Climatológica Del Sitio Del Proyecto Minero Cerro Del Gallo”. April 27, 2014 Revision.

Randall, J.A., Saldana, E.A., and Clark, K.F., 1994. Exploration in a volcano-plutonic center at Guanajuato, Mexico. Economic Geology. v.89, pp1722-1751

Rowins, S.M., 2000a. Preliminary results of a geochemical investigation of porphyry Cu-Au-Mo (Ag-Pb-Zn) and epithermal Ag-Au mineralization at the San Antón deposit, Guanajuato Mexico. Unpublished report to Luismin S.A. de C.V. August 2000.

Rowins, S.M., 2000b. A model for the genesis of “reduced” porphyry copper-gold deposits. The Gangue: October 2000: Issue 67, 1-7

Sanford Information Systems, 2019. Unpublished internal report on the Cerro del Gallo database audit prepared for Argonaut Gold Co. August 2019.

Sedgman, 2012, Technical Report, First Stage Heap Leach Feasibility Study, Cerro del Gallo Gold Silver Project, Guanajuato, Mexico, June 2012

Sedgman, 2012, Definitive Feasibility Phase Report, Cerro del Gallo Gold Silver Project, May 2012, prepared for Cerro Resources NL

SGS, 2015, An Investigation into the Recovery of Gold from the Cerro del Gallo Deposit, April 2015, prepared for Primero Mining

SGS, 2013, Cerro del Gallo Testwork Job No: 0168BH, July 2013, prepared for Argonaut Gold Inc.

SGS, 2011, High Pressure Grinding Rolls Testwork for Cerro Resources Cerro del Gallo Project, Job No: 10817, September 2011, prepared for Cerro Resources

SGS, 2011, Heap Leaching Testwork on Cerro de Gallo Oxide Gold Ore Samples – Comparison of Crushing Techniques, Job No: 10652, May 2011, prepared for Kings Minerals NL

SGS, 2009, Column Leach Testwork on Cerro del Gallo Composite SA-276, Job No: 10494, December 2009, prepared for Kings Minerals NL

SGS, 2009, Cerro del Gallo Heap Leach Amenability Testwork, Job No: 10467, September 2009, prepared for Kings Minerals NL

Siima, 2023, Estudio Hidrogeológico y de Línea Base para Minera Cerro del Gallo. Diciembre 2023

Smailbegovic, A., 2013, Remote Sensing and Field Evaluation of Cerro Del Gallo (CDG) project in Guanajuato, Mexico; July 2013, Report by Amer Smailbegovic, Consulting Geophysicist to Primero Mining Corp., 59 p. slide deck.

SRK Consulting, 2016, Geochemical Evaluation of Waste Rock and Ore: San Anton de las Minas: unpublished report prepared for San Anton de las Minas SA de CV, October, 543 p.

Swiridiuk, P., 2005a, Cerro del Gallo 3D Induced Polarization Survey, August 2005, Report prepared by AIMEX Geophysics for Kings Minerals N.L., 81 p.

Swiridiuk, P., 2005b, Cerro Del Gallo 3D IP Fine Mesh Model Results and Magnetic Modeling; October 2005, Report Prepared by AIMEX Geophysics for Kings Minerals N.L., 47 p.

Tardy et. Al., 1994. The Guerrero Suspect Terrane (Western Mexico) and coeval arc terranes (the Greater Antilles and the Western Cordillera of Colombia) – a Late Mesozoic intraoceanic arc accreted to cratonal America during the Cretaceous: Tectonophysics, v. 230, p. 49-73

Technical Report, 2015, Cerro del Gallo Metallurgical Testwork 2014, BBA document no. 3221002-000000-40-ERA-001/R00. April 2015, prepared for Primero Mining

Technical Report, 2012, First Stage Heap Leach Feasibility Study, Cerro del Gallo Gold Silver Project, Guanajuato, Mexico. June 2012, prepared for Cerro Resources NL

Technical Report, 2011, Feasibility Study and Preliminary Assessment, Cerro del Gallo Project, Guanajuato, Mexico, May 2011, prepared for Cerro Resources NL

Technical Report, 2010, Preliminary Assessment, Cerro del Gallo Project, Guanajuato, Mexico. April 2010, prepared for San Antón Resource Corporation

Technical Report, 2008, Cerro del Gallo deposit within the San Antón Property Mexico, Prepared by San Antón Resource Corporation Inc. Ottawa, Canada 2008

The Mines Group, 2019, Cerro del Gallo Pit Slope Stability Review, Memorandum. September 2019, prepared for Argonaut Gold

The Mines Group, 2011, Feasibility Level Pit Slope Stability Study, Cerro del Gallo Project, Guanajuato, Mexico. January 2011, prepared for San Antón Resources

Thompson, J.F.H., Sillitoe, R.H., Baker, T., Lang, J.R., and Mortensen, J.K., 1999. Intrusion-related gold deposits associated with tungsten-tin provinces. Mineralium Deposita, 34: pp323-334.

Townend, R., 2006. Screening, TBE separations of four composite samples, Optical/SEM examination of four polished sections and XRD of gangue. Unpublished report by Roger Townend and Associates, Consulting Mineralogists

Wave Geophysics, 2015 Cerro del Gallo Project Guanajuato, México; 31 May 2015, Presentation Prepared by Wave Geophysics for Primero Mining Corporation, 10 slides.

Western Environmental Testing Laboratory, 2019. Cerro del Gallo Project, Analytical, Report, October 2019. Order ID 19090229 prepared for KCA.

Western Mining Services, 2014. A description of ASTER Satellite data over the San Anton District, Guanajuato, Mexico; Nov 2014, report (PowerPoint presentation) prepared for Primero Mining Corp., 8 p.

United States Department of the Interior, BLM. 2014. BLM Alaska mining reclamation bonding guide, September.

United States Forest Service, United States Department of Agriculture. 2004. Training Guide for Reclamation Bond Estimation and Administration for Mineral Plans of Operations Authorized and administered under 36 CFR 228A, April, 52 p. and 7 appendices).

Zonge, 2010, Dipole-Dipole Complex Resistivity IP (CRIP), Natural Source AMT, Gravity, Ground Magnetic and Radiometric geophysical surveys; Report by Zonge International, Inc.

28 Date and Signature Page

Effective Date of report: December 11, 2025

"Travis J. Manning"
Travis Manning, PE
Date Signed:
January 23, 2025


Marvin Silva, PE
Date Signed:
January 23, 2025

"Todd Minard"
Todd Minard, PE
Date Signed:
January 23, 2025

"Timothy O. Kuhl"
Timothy O. Kuhl, RM SME, PG
Date Signed:
January 23, 2025

"Ted Eggleston"
Ted Eggleston, RM SME, PG
Date Signed:
January 23, 2025

"Jeffery Choquette"
Jeff Choquette, PE
Date Signed:
January 23, 2025

"Dawn Garcia"
Dawn Garcia, CPG
Date Signed:
January 23, 2025

29 Certificates of Authors

Travis J. Manning, P. E.

I, Travis J. Manning, P.E., as an author of this report entitled "NI43-101 Technical Report Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico" with an Effective Date of 11 December 2025, do hereby certify that:

1. I am Senior Engineer for Kappes, Cassiday & Associates located at 7950 Security Circle, Reno, Nevada 89506;
2. I graduated with a Bachelor of Science degree in Metallurgical Engineering from the University of Nevada in 2002;
3. I am a Registered Member of the Society for Mining, Metallurgy and Exploration (4138289 RM);
4. I am a Professional Engineer in the State of Utah (No. 6880159-2202);
5. I have worked as a Metallurgical Engineer for 21 years;
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101. I am independent of Heliostar Metals. and related companies applying all of the tests in section 1.5 of National Instrument 43-101. I have had no prior involvement with the Cerro del Gallo Project;
7. I am responsible for Sections 2, 4, 5, 13, 17, 18, 19, 23 and 27 and portions of Sections 1, 3, 6, 21, 24, 25 and 26;
8. I visited the Cerro del Gallo Site on 23 July 2025;
9. As of the effective date of this report, to the best of my knowledge, information and belief, the part of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading;
10. I have read National Instrument 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.

Dated 23rd day of January 2026

Travis J. Manning



AECOM

CERTIFICATE OF QUALIFIED PERSON

I, Todd E. Minard, P.E. (California and Nevada), of Reno, Nevada, USA, Sr Consultant with AECOM Technical Services Inc, as an author of this report entitled ""NI43-101 Technical Report Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico", with an Effective Date of 11 December 2025, do hereby certify that:

1) I am employed as a Senior Consultant at AECOM Technical Services Inc, with an office at 4741 Caughlin Parkway Suite #4, Reno, USA, 895.

2) I am a Registered Professional Engineer registered with the State of Nevada Board of Professional Engineers and Land Surveyors (License Number: 013199, Discipline: CE, Expiration Date: Dec 31, 2027).

3) I am a graduate of California State University, Chico with a Bachelor of Science (Civil Engineering) in 1993. I have practiced my profession continuously since 1993.

4) I am familiar with the National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, professional registration and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

5) I personally visited the site in February 2014, October 2019, and July 2025.

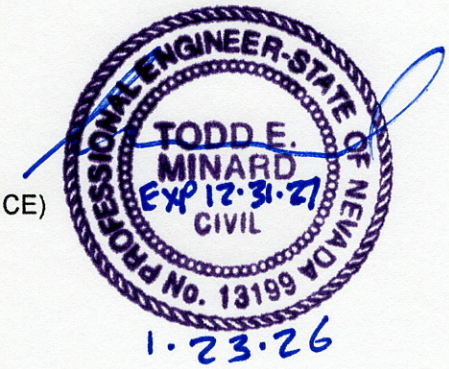
6) I am responsible for Sections 6.2, 12.4, 16.6, 17.2.3, 17.2.4 and 26.5 of the Technical Report

7) I am independent of the Issuer as described in Section 1.5 of NI 43-101.

8) I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23st day of January 2026



Todd E. Minard, P.E. (Nevada 013199 CE)
AECOM

Dr. Marvin Silva, Ph.D., P.E., P.Eng.

I, Dr. Marvin Silva, Ph.D., P.E., P.Eng., as an author (Qualified Person) of this report entitled "NI 43-101 Technical Report Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico" with an Effective Date of 11 December 2025, do hereby certify that:

1. I am Principal Geotechnical Engineer for Tailings Heap Solutions LLC, located at 1014 N Saguaro St, Chandler, AZ 85224;
2. I graduated with a Doctor of Philosophy (Ph.D.) in Geotechnical/Geo-Environmental Engineering from the University of Alberta in 1999;
3. I am a Professional Engineer (P.E.) in Arizona (43472) and a Professional Engineer (P.Eng.) in Alberta (52477);
4. I have worked as a Geotechnical Engineer for 25 years, with specialized experience in geotechnical characterization, rock slope engineering, and mine infrastructure;
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education and advanced academic training in geotechnical engineering, my professional engineering licensure, and my more than 25 years of directly relevant experience in mine geotechnics, including open pit slope characterization and stability assessment; development and review of geotechnical models; interpretation of drill core and geotechnical logging; evaluation of rock mass conditions, structural controls, and hydrogeologic influences on slope performance; and application of accepted industry methods for slope stability and design criteria, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101. I am independent of Heliostar Metals and related companies, applying all of the tests in Section 1.5 of NI 43-101, and I have had no prior involvement with the Cerro del Gallo Project other than the work described in this Technical Report;
6. I am responsible for Sections 13/14/15/16, and Section 25 (Interpretation and Conclusions) / Section 26 (Recommendations) as they relate to pit slopes; plus applicable portions of Section 1–3 and 2 (site visits);
7. I visited the Cerro del Gallo Site on 23 July 2025;
8. As of the effective date of this report, to the best of my knowledge, information and belief, the part of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading;
9. I have read National Instrument 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form;
10. I consent to the filing of this Technical Report and to the use of my name in connection with this Technical Report and any extracts from or summaries of the Technical Report that are included in news releases, presentations, or filings made in accordance with applicable securities laws.

Dated 23rd day of January 2026



Marvin Silva

Mr. Timothy O. Kuhl, RM SME

I, Timothy O. Kuhl, RM SME, PGeo., as an author of this report entitled “NI43-101 Technical Report Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico” with an Effective Date of 11 December 2025, do hereby certify that:

1. I am Principal Geologist for Mine Technical Services Ltd located at 18124 Wedge Parkway, #924, Reno, Nevada 89511;
2. I graduated with a Bachelor of Science degree in Geological Engineering (1975) and a MSc degree in Geology (1982) from South Dakota School of Mines and Technology;
3. I am a Registered Member of the Society for Mining, Metallurgy and Exploration (1802300);
4. I have worked as a Geologist for 45 years;
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101. I am independent of Heliostar Metals. and related companies applying all of the tests in section 1.5 of National Instrument 43-101. I have had no prior involvement with the Cerro del Gallo project;
6. I am responsible for Sections 1.7, 14, and 26.3 and portions of Sections 2 and 27;
7. I visited the Cerro del Gallo Site on 18 to 20 June 2025;
8. As of the effective date of this report, to the best of my knowledge, information and belief, the part of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading;
9. I have read National Instrument 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.

Dated 23rd day of January 2026



Timothy O. Kuhl

CERTIFICATE OF QUALIFIED PERSON

I, Ted Eggleston, RM SME, am employed as Principal Geologist, with Mine Technical Services Ltd., with a street address at 4110 Twin Falls Drive, Reno, NV, 89511.

This certificate applies to the technical report titled “NI43-101 Technical Report Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico” that has an effective date of 11 December, 2025 (the “technical report”).

I am a Registered Member (RM) of the Society of Mining, Metallurgy, and Exploration (SME), registration number 115851 RM. I graduated with a Bachelor of Science degree in geology from the Western Colorado University in 1976 and both an MSc (1982, geology) and Ph.D. (1987; geology) from New Mexico Institute of Mining and Technology;

I have practiced my profession continuously since 1976. I have been directly involved in gold and base metal exploration and mining projects in the United States, Chile, Perú, Canada, Ecuador, Colombia, Ghana, Burkina Faso, and Mexico. I have been involved with exploration and data acquisition and with in the evaluation of data quality, geologic modeling, resource modeling, and estimation for gold, base metal, and industrial mineral projects in North and South America, Africa, and the Asia Pacific.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101) for those sections of the technical report that I am responsible for preparing.

I most recently visited the Cerro del Gallo Operations 18 to 20 June 2025, a duration of three days.

I am responsible for Sections 1.4, 1.5, 1.6, 2.2 (discussion of site visit by Dr. Ted Eggleston only), 7, 8, 9, 10, 11, 12.2, 12.3, 25.1, 25.2, 26.3, 27 of the technical report.

I am independent of Heliostar Metals Ltd. as independence is described by Section 1.5 of NI 43–101.

I have had no prior involvement with the Cerro del Gallo project.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 23 January, 2026

“Signed”

Ted Eggleston, RM SME.

Jeffery Choquette, P. E.

I, Jeffery Choquette, P.E., as an author of this report entitled “NI43-101 Technical Report Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico” with an Effective Date of 11 December 2025, do hereby certify that:

1. I, Jeffery Choquette, P.E., am employed as a Principal Engineer with Hard Rock Consulting LLC, with an office address at 13918 E. Mississippi Ave., Suite 474, Aurora, CO 80012;
2. I am a graduate of Montana College of Mineral Science and Technology and received a Bachelor of Science degree in Mining Engineering in 1995;
3. I am a QP Member in Mining and Ore Reserves in good standing of the Mining and Metallurgical Society of America (No. 01425QP);
4. I am a Registered Professional Engineer in the State of Montana (No. 12265);
5. I have 29-plus years of domestic and international experience in project development, resource and reserve modeling, mine operations, mine engineering, project evaluation, and financial analysis. I have worked for mining and exploration companies for 15 years and as a consulting engineer for 14 years. I have been involved in industrial minerals, base metals and precious metal mining projects in the United States, Canada, Mexico and South America;
6. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101. I am independent of Heliostar Metals. and related companies applying all of the tests in section 1.5 of National Instrument 43-101. I have had no prior involvement with the Cerro del Gallo project;
7. I am responsible for Sections 1.8, 1.9, 1.14 (mining costs), 1.15, 15, 16 (excepting 16.2), 21.2.1, 21.3.1, 22, 24.3.3, 24.4 and portions of Sections 2 and 27;
8. I visited the Cerro del Gallo Site on 23 July 2025;
9. As of the effective date of this report, to the best of my knowledge, information and belief, the part of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading;
10. I have read National Instrument 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.

Dated 23rd day of January 2026





Jeffery Choquette

Stantec

CERTIFICATE OF QUALIFIED PERSON

DAWN H. GARCIA

I, Dawn H. Garcia, state that:

(a) I am a Senior Associate at:
Stantec Consulting Services Inc.
One South Church Avenue, Suite 2100
Tucson, Arizona, USA

(b) This certificate applies to the technical report titled "NI43-101 Technical Report, Pre-Feasibility Study of Cerro del Gallo Project, Guanajuato, Mexico" with an effective date of 11 December 2025 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows: I am a graduate of Bradley University with a bachelor's degree in Geological Sciences in 1982 and a graduate of California State University, Long Beach, with a master's degree in Geology in 1995. I am a licensed Professional Geologist in Arizona (License No. 26034) and am certified as a Professional Geologist (CPG) with the American Institute of Professional Geologists (Membership Number 08313). I am also a registered member of the Society for Mining, Metallurgy & Exploration (Membership No. 4135993). I have practiced my profession as an environmental geologist and hydrogeologist for over 35 years. I have over 20 years of experience in the mining industry. My relevant experience for the purpose of this Technical Report is:
- Acted as the Qualified Person for the Environmental, Permitting and Social section for 20 public disclosure technical reports and more than 20 detailed environmental and permitting reviews.
- Conducted environmental, socio-economic, or water-related tasks for over 50 mineral development, mineral processing, and mining operations.

(d) My most recent personal inspection of the Cerro del Gallo Project occurred on July 23 and 24, 2025, and was for a duration of two days.

(e) I am responsible for Sections 1.1, 1.2, 1.13, 1.17 (environmental only), 2.1, 2.2, 2.3, 20, 25.3, 26.4, and 27 of the Technical Report.

(f) I am independent of the issuer as described in Section 1.5 of NI 43-101.

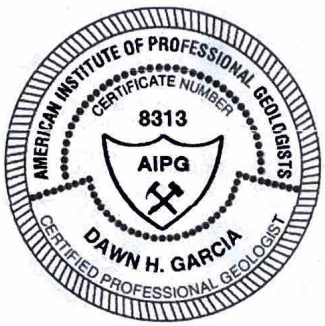
(g) I have not had previous involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the 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.

Dated at Tucson, Arizona, USA this 25th day of January 2026.

Dawn H. Garcia, PG, CPG



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